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OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XIX.

MAY, 1890, TO FEBRUARY, 1891,

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PREFACE.

It was the intention of the Council that this volume should include not only the proceedings and papers of the New York meetings of the Institute and of the International sessions held at Pittsburgh, but also, by the kind permission of the Council of the Iron and Steel Institute, the proceedings and papers of the New York meeting of that society; and in addition to these, the Address and proceedings connected with the inauguration of the Holley Memorial, and finally, a complete account of the extended excursions in which the members of both societies and of the Verein Deutscher Eisenhüttenleute participated as guests of the General Reception Committee representing members of the various technical and professional societies and of the iron and steel trade of the United States.

The dimensions of this book furnish a self-evident explanation of the necessary abandonment of a part of this design. The length of the technical papers it contains and of the discussions annexed to many of them was increased considerably during the process of publication by the contributors themselves. Moreover, unavoidable delays in obtaining the final revised copies from distant and scattered sources made it impracticable, until the larger part of the sheets had been printed, to estimate exactly the space required for the remainder; and when at last the papers and discussions were all in hand, it was too late to divide the volume into two, and the only recourse left was the sacrifice of much of the preliminary matter,* and the omission of the Holley Memorial Address, which will be published separately.

The introductory part of this volume, therefore, will be found to contain merely the outline of the proceedings of the various meetings, and a very brief and cursory account of the labors of the General Reception Committee. While these deficiencies are sincerely regretted, it is believed, nevertheless, that the most valuable and satisfactory souvenir of this remarkable and delightful episode in the history of the Institute is furnished by the series of important professional contributions for which it gave occasion.

July, 1881.

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* This is the less to be lamented, in view of the announcement in the *Journal of the Iron and Steel Institute* (No. II., 1890, p. 645) that the Council of that society has decided to issue a special memorial volume, giving an account of the visits and excursions of its members during their stay in the United States.

OFFICERS.

For the Year Ending February, 1891.

PRESIDENT,*

ABRAM S. HEWITT, New York City.

VICE-PRESIDENTS:

FOR TERM ENDING FEBRUARY, 1891.†

ECKLEY B. COXE, Drifton, Pa. — CHARLES MACDONALD, New York City. — PERCIVAL ROBERTS, JR., Philadelphia, Pa.

FOR TERM ENDING FEBRUARY, 1892.

HENRY M. HOWE, Boston, Mass. — S. F. EMMONS, Washington, D. C. — H. S. MUNROE, New York City.

MANAGERS:

FOR TERM ENDING FEBRUARY, 1891.‡

ROBERT FORSYTH, Chicago, Ill. — KENNETH ROBERTSON, Birmingham, Ala. — C. M. ROLKER, New York City.

FOR TERM ENDING FEBRUARY, 1892.

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FOR TERM ENDING FEBRUARY, 1893.

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---

* John Birkinbine, Philadelphia, Pa., was elected President in February, 1890.

† S. B. Christy, Berkeley, Cal.; E. C. Pechin, Roanoke, Va., and J. C. F. Randolph, New York City, were elected Vice-Presidents in February, 1891.

‡ J. B. Mackintosh, New York City; J. A. Porter, Durango, Col., and J. C. Smock, Trenton, N. J., were elected Managers in February, 1891; and, Mr. Mackintosh having died, J. C. Bayles, New York City, was appointed Manager to fill the vacancy thus occasioned.

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PROCEEDINGS OF THE FIFTY-SEVENTH MEETING, NEW YORK CITY, SEPTEMBER, 1890.

Local Committee.

J. F. Lewis; *Chairman;* James A. Burden, John Stanton, J. F Hollaway, James E. Denton, H. S. Munroe, J. B. Mackintosh, F. M. Pierce, W. S. Church, J. C. Bayles, George B. Wilkinson, David Williams, W. H. Wiley, H. W. Bulkley, W. H. Adams, E. Gybbon Spilsbury, George W. Maynard, Wm. Kent, Thomas Egleston, Charles Macdonald, Kenneth Robertson.

Hotel-headquarters were at the Park Avenue Hotel. Sessions were held at Chickering Hall.

The opening session of the meeting was held at 3 P.M., on Monday, September 29th. After preliminary announcements by the Local Committee, President Abram S. Hewitt made some brief remarks, postponing to a later period the delivery of the formal Presidential Address.*

The Secretary announced that the Council had decided to hold in February, 1891, no meeting for the reading and discussion of papers, but only a meeting for the purpose of opening ballots cast by mail, declaring the result of the election of officers and receiving the Annual Report of the Council: the reason of this decision being, that the great length and the laborious character of the series of meetings now to be held, and the extensive excursions and entertainments connected therewith, would make the organization of another meeting so soon after a difficult and doubtful matter.

The following papers were presented:

Explosions from Unknown Causes, by J. C. Bayles, East Orange, N. J.

Magnetic Concentration at the Michigamme Iron Mine, Lake Superior, by John C. Fowle, Michigamme, Mich.

Ore-Dressing by Electricity at the Tilly Foster Mine, by F. H. McDowell, New York City.

The Magnetization of Iron-Ore, by Clemens Jones, Hokendauqua, Pa.

The Ball-Norton Electro-Magnetic Separator, by C. M. Ball, Troy, N. Y.

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* By reason of Mr. Hewitt's illness, this address was not read, but only distributed in print. See page 475 of the present volume.

A working-model of this machine was exhibited in the hall at the close of the session.

At the second session, held at 8 P.M., on Monday, lectures were delivered, illustrated with lantern views, by Prof. W. B. Potter, of St. Louis, Mo., on the Iron Mountain Mine, and by W. F. Durfee, of Birdsboro', Pa., on American and Foreign Practice with the Diamond Drill.*

The third session was held at 10 A.M., on Tuesday, September 30th. The following papers were presented:

Physical and Chemical Equations of the Open-Hearth Process, by H. H. Campbell, Steelton, Pa.

Electric Power-Transmission in Mining Operations, by H. C. Spaulding, Boston, Mass.

Amalgamation at the Comstock Lode, Nevada; a Historical Sketch of Milling Operations at Washoe, and an Account of the Treatment of Tailings at the Lyon Mill, Dayton, by A. D. Hodges, Jr., Boston, Mass.

Notes on the Excavation of the New Croton Aqueduct, by J. P. Carson, New York City.

The fourth session was held at 2 P.M., on Tuesday, when the following papers were presented:

Notes on Recent Improvements in German Steel-Works and Rolling-Mills, by R. M. Daelen, Düsseldorff, Germany.

Machinery for the Charging of Heating- and Melting-Furnaces, by S. T. Wellman, Thurlow, Pa.

A Suspended Feed-Table for Rolling-Mills, by James Morgan, Pittsburgh, Pa.

Pneumatic Hoisting, by H. A. Wheeler, St. Louis, Mo.

The Walker Anti-Vibration Regulating Shutter, or Slide, for Guibal and other Enclosed Fans, by Edward R. Walker, Weybridge, England.

The Department of Metallurgy and Economic Geology in the National Museum, by F. P. Dewey, Washington, D. C.

Notes on Coal-Mining in Oregon, by R. H. Norton, Portland, Oregon.

The Resources of the Black Hills and Big Horn Country, Wyoming, by H. M. Chance, Philadelphia, Pa.

* These lectures have not been furnished in manuscript for publication.

The Potosí, Bolivia, Silver-District, by Arthur F. Wendt, New York City.

The Pratt Mines of the Tennessee Coal, Iron, and Railroad Company, by Erskine Ramsay, Pratt Mines, Ala.

The Paint-Ore Mines of Lehigh Gap, Pennsylvania, by C. E. Hesse, Bethlehem, Pa.

Analyses of Lake Superior Iron-Ores, by George W. Goetz, Milwaukee, Wis.

The fifth and final session was held at 8 P.M., on Tuesday, when Mr. A. Fteley, Chief Engineer of the New York Aqueduct, presented a series of lantern-views, with interesting explanations, in further illustration of the paper of Mr. Carson, presented at the third session.

The following papers were also presented, with illustrations upon the screen:

The Copper Resources of the United States, by James Douglas, New York City.

The New Iron-Breaker at Drifton, with Remarks on the Preparation of Anthracite Coal, by Eckley B. Coxe, Drifton, Pa.

In addition to the papers above named, all of which were either presented in printed form or read at length, the following papers were read by title and subsequently printed for distribution:

Latest Developments in Compressed-Air Motors for Tramways, by D. S. Jacobus, Hoboken, N. J.

Cast-Iron Tools for Cutting Metals, by Oberlin Smith, Bridgeton, N. J.

On Sulphur in Bessemer Steel, by John W. Cabot, Bellaire, O.

Interesting Vein-Phenomena in Boulder County, Colorado, by John B. Farish, Denver, Colo.

Notes on the Progress of Mining in China, by Ellis Clark, Philadelphia, Pa.

The Geology of the Haile Mine, South Carolina, by A. Thies, Concord, N. C., and A. Mezger, New York City.

The Thies Process of Treating Auriferous Sulphides, at the Haile Gold-Mine, Lancaster County, South Carolina, by A. Thies, Concord, N. C., and William B. Phillips, University of Alabama, Tuscaloosa, Ala.

The Genesis of the Edgar Thomson Blast-Furnaces, by William P. Shinn, Pittsburgh, Pa.

Progress in Magnetic Concentration of Iron-Ore, by John Birkinbine, Philadelphia, Pa.

Some Ontario Magnetites, by T. D. Ledyard, Toronto, Ont.

The Presidential Address of Hon. A. S. Hewitt was illustrated with a magnificent map of the United States east of the Rocky Mountains, suspended in the hall. Upon this map, prepared by Mr. Wheaton B. Kunhardt, the location of every iron blast-furnace and the distribution of every variety of iron-ore, and of the productive coal-fields throughout the country was indicated.

The following lists include the persons elected at the sessions of the meeting, or by ballot through the mail between the Washington meeting of February, 1890, and the June meeting of 1891.

Elected June, 1890, by Mail.

MEMBERS.

William H. Beck,	Washington, D. C.
P. T. Berg,	Munhall, Pa.
Alex. Bowie,	Gallup, New Mexico.
William L. Brockway,	Jersey City, N. J.
Arthur R. Browne,	West Coast, Tasmania.
W. M. Chauvenet,	St. Louis, Mo.
Harry W. Coffin,	Tuscarora, Nev.
W. O. Comstock,	New York City.
Frank G. Darlington,	Indianapolis, Ind.
Britton Davis,	Corralitos, Mexico.
G. D. Delprat,	London, Eng.
E. T. Dumble,	Austin, Texas.
H. T. Ellicott,	Paris, France.
John C. Fowle,	Michigamme, Mich.
Seth Frink,	Rowley, Colo.
Ernest E. Goodell,	Marysville, Mont.
F. P. Gracey,	Clarksville, Tenn.
James H. Henley,	Marysville, Mont.
W. H. Jaques,	Bethlehem, Pa.
Washitaro Kasahara,	Tokio, Japan.
W. H. Kemler,	Pittsburgh, Pa.
Tong Sing Kow,	Shanghai, China.
Ren Kurimoto,	Tokio, Japan.
James Malcolmson,	Michoacan, Mexico.
Henry Morris,	Johannesburg, Transvaal, S. Africa.
Joseph Morris,	Medellin, U. S. Colombia, S. A.
Albert E. Noble,	Jenifer, Ala.
S. A. Parnall,	Ishpeming, Mich.
John A. Partridge,	Washington, D. C.
Horace See,	New York City, N. Y.
H. C. Spaulding,	Boston, Mass.

James A. Sperry, ... Cooney, New Mexico.
A. B. Stollmeyer, ... Trinidad, Brit. West Indies.
Ambrose Swasey, ... Cleveland, O.
Scitaro Uchida, ... Tokio, Japan.

ASSOCIATES.

A. M. Bigelow, ... New York City, N. Y.
James Norfleet, ... Fort Payne, Ala.
Percival Bentley Waugh, ... Sutton, Surrey, Eng.

Elected at the New York Meeting, October, 1890.

MEMBERS.

William R. Appleby, ... Jersey City, N. J.
J. B. Archer, ... New York City.
J. F. Armstrong, ... Ishpeming, Mich.
C. M. Ball, ... West Troy, N. Y.
Henry C. Banon, ... Lenardo, Colo.
D. M. Barringer, ... Philadelphia, Pa.
Walter A. Barrows, Jr., ... Sharpsville, Pa.
George B. Bartlett, ... South Chicago, Ill.
George Fisher Beardsley, . . . Captain's Flat, N. S. Wales.
Richard Blackstone, ... Central City, S. Dakota.
Robert M. Blankenship, ... Richmond, Va.
Seth Bullock, ... Deadwood, S. Dakota.
William Clarke Catlin, ... South Chicago, Ill.
F. M. F. Cazin, ... New York City, N. Y.
C. F. Chandler, ... New York City, N. Y.
E. W. Codington,, ... Bartow, Fla.
E. T. Cox, ... New York City, N. Y.
Fred. Crabtree, ... McKeesport, Pa.
Lewis E. Dunham, ... Ashland, Wis.
E. B. Eckman, ... Columbia, Pa.
Fritz Eilers, ... Homestead, Pa.
Sydney H. Farrar, ... Johannesburg, Transvaal, S. Africa.
Henry G. Fiske, ... New York City, N. Y.
F. H. Foote, ... South Chicago, Ill.
C. B. Fowler, ... Leetonia, O.
Charles D. Fuller, ... Latrobe, Pa.
Samuel T. Godbe, ... Pioche, Nev.
Anton Hardt, ... Wellsboro', Pa.
Max J. Hartung, ... Sydney, N. S. Wales.
Arthur Hendy, ... Denver, Colo.
Conrad E. Hesse, ... Mt. Carmel, Pa.
George J. Hicks, ... Pittsburgh, Pa.
John L. Hoffman, ... Duquesne, Pa.
Frank E. Hopke, ... Hastings-on-the-Hudson, N. Y.
M. R. Hunt, ... Ashland, Wis.
Henry A. Judd, ... Bilesville, N. C.
Tom Cobb King, ... Ironaton, Ala.
F. H. Knight, ... Easton, Pa.
Theo. Knutzen, ... Deadwood, S. Dakota.

Elmer H. Lawall,	Wilkesbarre, Pa.
Thomas McDonald,	Duquesne, Pa.
C. H. Macnutt,	Aspen, Colo.
Robert Mannesmann,	New York City, N. Y.
Nicholas Manzavino,	Smyrna, Turkey.
T. Spencer Miller,	New York City, N. Y.
Charles B. Murray,	Buena Vista, Va.
W. H. Nimick,	Pittsburgh, Pa.
Stephen N. Noble,	Ironaton, Ala.
William Oothout, Jr.,	San Francisco, Cal.
Frank A. Ray,	Buchtel, O.
W. J. Richards,	Wilkesbarre, Pa.
Marvin F. Scaife,	Mt. Carmel, Pa.
C. G. E. Schmiterlow,	Steelton, Pa.
James Scott,	Pittsburgh, Pa.
John Shennan,	Clifton, Ariz.
Frank O. Stetson,	Salem, Va.
Charles L. Taylor,	Pittsburgh, Pa.
John M. Thomas,	Niles, O.
Carl Trautvetter,	Sharpsville, Pa.
Edward M. Van Corrtlandt,	Cana, U. S. of Colombia, S. A.
Ralph W. Watson,	Calumet, Mich.
Cabell Whitehead,	Washington, D. C.

ASSOCIATES.

E. Davenport Cleland,	Broken Hill, N. S. Wales.
Francis R. Fava, Jr.,	Washington, D. C.
Henry M. Fish,	Joliet, Ill.
Walter M. Francis,	Philadelphia, Pa.
Walter Richardson Francis,	Clausthal, Germany.
G. H. Gile,	Oshkosh, Wis.
W. H. Helms,	Port Henry, N. Y.
H. D. Johnson,	South Bend, Ind.
A. Lanfear Norrie,	New York City.
D. L. Plumer,	Wausau, Wis.
Alex. T. Porter,	Troy, N. Y.
N. Taylor,	Wilkesbarre, Pa.

Elected February, 1891, by Mail.

MEMBERS.

John S. Adriance,	Huntingdon, Pa.
James Archbald, Jr.,	Scranton, Pa.
Emmett Armstrong,	Harrisburg, Pa.
Alvah H. Axford,	Custer, Idaho.
Franklin Bache,	Santiago, Cuba.
Juan B. Blasquez,	Pachuca, Mexico.
H. Brauns,	Dortmund, Germany.
Alfred H. Bucherer,	Lockport, N. Y.
Jos. L. Budd,	Braddock, Pa.
Charles Bullman,	Plainfield, N. J.
F. M. Coghlan,	San Luis Potosi, Mexico.

F. S. Cook,	Yonkers, N. Y.
R. M. Daelen,	Dusseldorf, Germany.
William P. Darling,	Buffalo, N. Y.
Thomas G. Davey,	Harrietville, Victoria, Australia.
Frank Drake,	Negaunee, Mich.
Vincent Ferguson,	Buchtel, O.
William C. A. Ferguson,	Brooklyn, N. Y.
James W. Fox,	Big Stone Gap, Va.
Percy C. Gilchrist,	London, Eng.
R. B. Green,	Ishpeming, Mich.
H. Hamilton Gunn,	Singapore, East India.
Robert Dalton Hale,	Broken Hill, N. S. Wales.
Harry Robert Hall,	Parryville, Pa.
J. C. Henderson,	New York City.
Francis Hinton,	Milwaukee, Wis.
Charles W. Hunt,	New York City.
T. Kitsunesaki,	Tokio, Japan.
Walter J. Koehler,	Broken Hill, N. S. Wales.
Emil Krabler,	Altenessen, Germany.
Story B. Ladd,	Washington, D. C.
Werner Langguth,	Deadwood, S. Dakota.
James E. Lawton,	Cumberland Gap, Tenn.
William McNeill Leete,	Sierra City, Cal.
Walter Leisenring,	Sandy Run, Pa.
Sir William Thomas Lewis,	Mardy, Wales.
Pei Gustaf Lidner,	Shullsburg, Wis.
William E. Loane,	Aspen, Colo.
George J. Lobdell,	Wilmington, Del.
Samuel McClure,	Sharon, Pa.
G. B. McCormack,	Pratt Mines, Ala.
Adolph Mezger,	Charlotte, N. C.
Charles Herbert Miller,	Rockhill Furnace, Pa.
Charles S. Morse,	Bridgeport, Conn.
Karl Neitzel,	Elkhorn, Mont.
Matthew Penhale,	Black Lake, Can.
Arnold K. Reese,	Sparrows Point, Md.
A. J. Reilly,	Pratt Mines, Ala.
George B. Reynolds,	Bombay, India.
John W. Richards,	Aspen, Colo.
C. A. O. Rosell,	Washington, D. C.
Arthur Earnest Savage,	Broken Hill, N. S. Wales.
T. Schlink,	Mulheim-on-Ruhr, Germany.
E. Schroedter,	Dusseldorf, Germany.
William G. Sharp,	Castle Gate, Utah.
Charles Henry Snow,	Dubuque, Ia.
Michael Druck Sohon,	South Bethlehem, Pa.
D. McNeely Stauffer,	New York City.
Charles Wade Stickney,	Butte, Mont.
Alex. Thielen,	Ruhrort, Prussia.
Edward Titcomb,	Nogales, Ariz.
T. Tsuno,	Shimeneken, Japan.
J. S. Unger,	Braddock, Pa.
Howard F. Welsh,	Arthur, Tenn.
Earl A. Wheeler,	Sharon, Pa.
Edward A. Whitmore,	Aspen, Colo.

Granger Whitney,	Santiago, Cuba.
Jasper Whiting,	South Chicago, Ill.
William Wilkins,	Santiago, Cuba.
David E. Williams,	Philadelphia, Pa.

ASSOCIATES.

Charles H. Bosher,	New York City.
William McHose Boyer,	Reading, Pa.
George J. Newton,	Pottstown, Pa.
Leo Von Rosenberg,	New York City.
Charles P. Scrivener,	Austin, Tex.
U. P. Swinburne,	Newcastle-on-Tyne, Eng
M. O. Wilson,	New York City.
R. T. Wilson,	New York City.

CHANGE OF STATUS.

F. W. Hardwick,	Exmouth, Eng.
Marmaduke B. Holt,	New York City.
N. B. Smith,	Birmingham, Ala.

Elected May, 1891, by Mail.

MEMBERS.

Charles M. Allen,	Butte, Montana.
Henry A. Bang,	New York City.
Hermann C. Bellinger,	Saltillo, Mexico.
Alex. Leslie Black,	New Orleans, La.
James Henry Carpenter,	Reading, Pa.
Benjamin Franklin Conner,	Columbia, Pa.
Otto C. Davidson,	Commonwealth, Wis.
Arthur O. Granger,	Cartersville, Ga.
Simeon C. Hazleton,	Sandy, Utah.
William D. Hoffman,	Brewster, N. Y.
William H. Hoffman,	Brewster, N. Y.
E. C. Huxley,	Boston, Mass.
Robert Emmet Jennings,	Jersey City, N. J.
Squier H. Macomber,	Marysville, Mont.
Joseph Massenez,	Höerde, Germany.
August A. Miller,	Roanoke, Va.
J. B. Nau,	New York City.
Lionel R. Nettre,	London, Eng.
Hiram Phillips,	Columbia, Pa.
Beverly S. Randolph,	Frostburgh, Md.
Charles Q. Rawling,	Wheeling, W. Va.
Samuel B. Robbins,	Great Falls, Mont.
Frederick Bowes Scott,	London, Eng.
H. E. Smith,	Milwaukee, Wis.
William H. Wahl,	Philadelphia, Pa.
Moriz Weis,	Sparrows Point, Md.
James W. R. Young,	Mount Gilead, N. C.

ASSOCIATE.

William S. Ackerman, ... Paterson, N. J.

CHANGE OF STATUS.

Stuart W. Cramer, ... Charlotte, N. C.
E. L. Zukoski, ... Coulterville, Cal.

The Holley Memorial.

On Thursday afternoon, October 2d, the memorial bust of Alexander Lyman Holley was unveiled in Washington Square. The memorial address was delivered in Chickering Hall, before the ceremony, by Mr. James Dredge, editor of London *Engineering*. This address, together with the other proceedings of the occasion, will be separately published.

Excursions and Entertainments.

The excursions and entertainments of this meeting were practically merged in those connected with the meeting of the Iron and Steel Institute of Great Britain, which immediately followed. The members of that society, and of the *Verein Deutscher Eisenhüttenleute*, the American societies of civil, mechanical, and electrical engineers, and of other technical societies, were invited to participate in the sessions of this Institute, and the Iron and Steel Institute was equally liberal in hospitality.

This Institute maintained, for the convenience of members and guests, during the period of both meetings in New York and the subsequent period occupied by the Pittsburgh International Sessions and the various local and general excursions—about six weeks in all—a Bureau of Information at the Park Avenue Hotel, with a force of clerks and messengers, and books, maps, railway schedules, post-office, telephone, and facilities for cashing drafts, purchasing railway tickets, type-writing, forwarding mail or baggage, etc. Assistance was generously given by Messrs. Thomas Cook & Sons, Messrs. Wyckoff, Seamans & Benedict (agents of the Remington type-writer), the Western Union Telegraph Company, and the American and Metropolitan Telephone and Telegraph companies.

The excursions provided by the New York local committee comprised the following:

Wednesday, October 1st —Excursion up the Hudson river on the steamer Sandy Hook, with a collation, served *en route*.

Thursday, October 2d.—Visit to the works of the Consolidated Gas Company. In the evening a reception to the members of the *Verein Deutscher Eisenhüttenleute* was given by the Liederkranz Society. On the same evening the annual dinner of the Iron and Steel Institute took place at Delmonico's.

Friday, October 3d.—Visit to the electrical works of New York City. Three excursions, as follows:

1. By steamboat up the East river to the Brooklyn Navy Yard, the De la Vergne Refrigerating Works, and the Washington Bridge over Harlem river. Luncheon on the boat.

2. By railroad to the Tilly Foster Iron-mine and Electrical Separating-works. Luncheon on the train.

3. By railroad to the Edison laboratory at Llewellyn Park, N. J., where the party were entertained at luncheon by Mr. Edison.

Also, optional visits by individuals or small parties to the following points:

The Spiral Weld Tube Works, East Orange, N. J.; works of the Passaic Zinc Company, New Jersey; Passaic Rolling-mill, Paterson, N. J.; New York and Brooklyn Bridge; Hudson river tunnel; and many other points of interest in and near the city.

In addition to the above list, the ladies accompanying members and guests of the Iron and Steel Institute and the American Institute of Mining Engineers were provided for in special excursions by carriage on Tuesday and Wednesday, to Central Park, Riverside drive, the Metropolitan Museum of Art, and leading jewelry, dry-goods, and other shops of the city. A theater party was also organized for their entertainment on Thursday evening. These and other arrangements for the reception of ladies were in charge of the following special ladies' committee:

Mrs. J. C. Bayles, *President;* Mrs. J. P. Holloway, *Vice-President;* Mrs. Horace See, *Secretary;* Mrs. Abram S. Hewitt, Mrs. Andrew Carnegie, Mrs. James A. Burden, Mrs. Edward Cooper, Mrs. George A. Crocker, Mrs. Henry R. Towne, Mrs. George H. Hull, Mrs. John C. Kafer, Mrs. R. W. Raymond, Mrs. Charles Macdonald, Mrs. J. H. Harris, Mrs. B. S. Church, Mrs. Levi Holbrook, Mrs. John Stanton, Mrs. W. A. Perry, Mrs. F. S. Witherbee, Mrs. F. H. Prentiss, Mrs. John Thomson, Mrs. David Williams, Mrs. J. F. Lewis, Mrs. F. R Hutton, Mrs. John Bogart, Mrs. C. M. Kempton, Mrs. S. W. Baldwin, Mrs. I. P. Pardee, Mrs. H. G. Torrey, Miss Bessie Baldwin, Miss Lily Raymond, Miss Josephine F. Rand, Miss Florence O. Rand, Miss Antoinette Harris, Miss Isabel Harris.

PROCEEDINGS OF THE INTERNATIONAL SESSIONS, PITTSBURGH, PA., OCTOBER, 1890.

Local Committee.

John H. Ricketson, *Chairman General Committee;* William L. Abbott, *Chairman Executive Committee;* Jos. Abel, G. Aertsen, Henry Aiken, Chester B. Albree, Edw. B. Alsop, D. Ashworth, Moses Atwood, Charles Bailey, James M. Bailey, Phineas Barnes, A. J. Barr, Hon. Thomas M. Bayne, G. M. Beach, Max. J. Becker, George A. Berry, Julius Bieler, John Bindley, George P. Black, John W. Black, G. H. Blaxter, A. Bradley, J. A. Brashear, A. F. Brown, H. G. Brown, J. Stuart Brown, J. W. Brown, Jos. S. Brown, J. I. Buchanan, H. Buhl, Jr., A. P. Burchfield, A. M. Byers, H. M. Byllesby, John Campbell, William Campbell, Robert H. Campe, David A. Carter, George A. Chalfant, J. W. Chalfant, James A. Chambers, Harvey B. Chess, A. H. Childs, George H. Clapp, C. J. Clarke, Edw. L. Clark, C. L. Cole, H. E. Collins, James Collord, William H. Conley, E. C. Converse, W. C Cronemeyer, Horace Crosby, H. M. Curry, Hon. John Dalzell, Henry Darlington, James Darsie, Charles Davis, Alex. Dempster, Charles R. Dilworth, Jos. R. Dilworth, Lawrence Dilworth, Charles Donnelly, Edw. T. Dravo, H. G. Dravo, Charles B. Dudley, Herbert DuPuy, William A. Edeburn, Theo. N. Ely, E. M. Ferguson, G. W. G. Ferris, C. L. Fitzhugh, John Flannery, E. A. Ford, Edw. Ford, J. B. Ford, Charles Foster, H. C. Fownes, W. C. Fownes, Aaron French, H. C. Frick, *Chairman Finance Committee;* J. W. Friend, James Gayley, T. H. Given, M. B. Goff, George J. Gorman, Mayor Gourley, Matthew Graff, A. L. Griffin, S. P. Harbison, John Harper, Jr., G. B. Harton, S. W. Hay, F. J. Hearne, H. J. Heinz, James Hemphill, D. C. Herbst, Henry D. Hibbard, I. E. Hirsch, Elliott Holbrook, W. J. Holland, John G. Holmes, C E. Hubbard, Millard Hunsiker, Alfred E. Hunt, *Secretary Executive Committee;* C G. Hussey, F. A. Hutchinson, J. B. Jackson, Benjamin F. Jennings, J. H. Johnson, William G. Johnston, B. F. Jones, W. Larimer Jones, A. F. Keating, Thomas Keenan, Jr., George A. Kelly, George D. Kelly, Hugh Kennedy, Julian Kennedy, John C. Kirkpatrick, W. E. Koch, James Lappan, H. W. Lash, George Lauder, G. M. Laughlin, H. A. Laughlin, James Laughlin, Jr., J. G. A. Leishman, William J. Lewis, W. E. Lincoln, Gustave Lindenthal, Henry Lloyd, Charles Lockhart, Andrew Long, W. B Lupton, Christopher L. Magee, S. S. Marvin, A. W. Mellon, William Metcalf, Charles Meyran, Reuben Miller, J. B. Milholland, John Moorhead, Jr., M. K. Moorhead, W. J. Moorhead, George Morris, L. M. Morris, William Mullins, R. Munroe, David McCargo, George McCleane, S. McClure, William McConway, James McCrea, William McCreery, James H. McCutcheon, John R. McGinley, H. Sellers McKee, W. M. McKelvy, Wharton McKnight, Charles B. McLean, George G. McMurtry, James McNeill, John N. Neeb, Theo. W. Nevin, F. F. Nicola, Alex. Nimick, Eugene M. O'Neill, W. W. O'Neil, Jr., D. B. Oliver, George T. Oliver, H. W. Oliver, James B. Oliver, Charles Paine, A. E. W. Painter, G. Blair Painter, Jacob Painter, Jr., William P. Palmer, Eugene W. Pargny, D. E. Park, James H. Park, William G. Park, Simon Perkins, Henry Phipps, James B. Pierce, J. J. Pierce, Robert Pitcairn, H. K. Porter, John C. Porter, W. C. Quincy, Henry R. Rea, Nelson P. Reed, Joshua Rhodes, William B. Rhodes, John H. Ricketson, O. G. Ricketson, D. C.

Ripley, Thomas B. Riter, T. P. Roberts, George Robinson, Thomas Rodd, William Roseburg, M. Rosenbaum, B. H. Rubie, W. T. Sanger, William J. Sawyer, C. C. Scaife, Marvin F. Scaife, O. P. Scaife, W. Lucien Scaife, William E. Schmertz, Fred. A. Scheffler, J. M. Schoonmaker, S. L. Schoonmaker, A. L. Schultz, C. J. Schultz, J. E. Schwartz, James B. Scott, Jos. S. Seaman, David Shaw, William P. Shinn, William H. Singer, John S. Slagle, F. Slataper, Andrew D. Smith, F. B. Smith, H. D. Smith, Persifor F. Smith, W. P. Snyder, Norman Spang, J. J. Spearman, John Z. Speer, C. P. Stafford, G. A. Steiner, Cephas Taylor, E. B. Taylor, William Thaw, Jr., E. H. Utley, J. J. Vandergrift, John Walker, John W. Walker, Jos. Walton, George Warden, M. W. Watson, Jos. D. Weeks, William Weihe, Calvin Wells, George Westinghouse, Jr., C. Y. Wheeler, E. A. Wheeler, Henry White, George I. Whitney, Frank Wilcox, John F. Wilcox, J. P. Witherow, Allan W. Wood, B. L. Wood, Jr., R. G. Wood, W. Dewees Wood, Mayor Wyman, Charles H. Zug.

Hotel-Headquarters, Monongahela House.

These sessions were held in Carnegie Hall, Allegheny City, by the Iron and Steel Institute, the Verein Deutscher Eisenhüttenleute and the American Institute of Mining Engineers, members of other technical societies being also invited to take part.

The first session was held on Thursday morning, October 9th, Sir James Kitson, Bart., President of the Iron and Steel Institute, in the chair. An eloquent address of welcome was delivered on behalf of the Local Committee of Reception by Mr. John H. Ricketson, of which the following is a condensed report:

After suitable complimentary allusion to the distinguished character of the assembly, and particular mention of Sir James Kitson, Sir Lowthian Bell, Sir William T. Lewis, and Messrs. Richards, Martin, Snelus and Jeans of the Iron and Steel Institute, and Mr. Berkeley, senior vice-president of the Institution of Civil Engineers, and of Messrs. Thielen, Wedding, Daelen and Schrœdter of the Verein Deutscher Eisenhüttenleute, all of whom were present, and an expression of regret at the absence of Sir Henry Bessemer, Hon. A. S. Hewitt and Mr. Andrew Carnegie, Mr. Ricketson gave a graphic description of the industrial situation of Pittsburgh, concluding as follows:

We have here twenty-one blast-furnaces, which in 1889 produced 1,293,435 tons of pig-iron; thirty-three rolling-mills, twenty-seven of which roll steel, and their production in 1889 was 1,105,573 tons of steel and 638,450 tons of rolled iron. Our annual capacity of steel rails is at present 550,000 tons. Our product of wrought-iron pipes this year will, I am informed, not fall short of 350,000 tons, while our output of structural iron and steel will be fully 165,000 tons. We have forty-nine iron foundries, representing a capital of

nearly $10,000,000. The principal electrical industry in Pittsburgh is in apparatus for incandescent lighting. Of the dynamos in the United States, having a capacity for the supply of current for 1,500,000 sixteen candle-power lamps, Pittsburgh alone has furnished 650,000, or nearly 44 per cent. We have fifteen firms or companies making window-glass, thirty-seven making flint and lime glass, and fifteen making green and black glass bottles. The 15,000 coke-ovens in this district consume 9,000,000 tons of coal in making their product of about 6,000,000 tons of coke. In 1889, in round numbers, we shipped to Southern markets by river, 4,000,000 tons; we shipped to Eastern markets by rail, 2,500,000 tons; we shipped by rail for reshipment by lake, 1,000,000; the railroads entering Pittsburgh used 1,000,000; we shipped by rail to North-Eastern markets, 500,000; we used for home and miscellaneous trade, 2,000,000; making a grand total of 20,000,000 tons, or about 60 per cent. of the output of bituminous coal in Pennsylvania.

The railway tonnage of Allegheny county, of business originating here, exclusive of what passes through, is 20,000,000 tons per annum, or a fraction over 3 per cent. of the total railway tonnage of the United States, which amounted in 1889 to 619,137,237 tons.

I am officially informed that in this district 750,000,000 feet of natural gas are delivered each day through 1125 miles of pipe to consumers in our mills and factories, and to upwards of 30,000 private warehouses, stores, hotels, and dwellings. The present annual displacement of coal by natural gas is estimated at about 8,000,000 of tons. There are 46,000 oil-wells in the United States, producing on an average 130,000 barrels per diem, and representing a capital of $120,000,000. There are 10,000 miles of pipe-lines for the transportation of crude petroleum, involving a capital of $50,000,000. The stock in tanks, at present, of crude petroleum is 26,500,000 barrels. The country has a refining capacity of 140,000 barrels per day for illuminating oil, and 15,000 barrels per day are used for fuel purposes.

Gentlemen, when in addition to the 300,000,000 of barrels of petroleum which have flowed from the soil of Pennsylvania between 1876 and 1890, we remember that she is still producing 2,500,000 barrels per month; that her total yearly output of bituminous coal is 30,000,000 of tons, and of anthracite 35,400,000 of tons, and that she produced last year within a small fraction of one half of all the pig-iron and more than half of all the rolled iron and steel made in the country, you can form some conception of the immeasureable and

boundless, though then hidden, resources of the princely domain of 44,986 square miles which King Charles II. ceded to the Penns in satisfaction of a claim against the British Government for £16,000.

More than two thousand years ago a Hebrew sage, in the spirit of prophecy, the fulfilment of which we are witnessing to-day, uttered these words, which in this presence seem like a benediction: "Every carpenter and work-master that laboreth night and day, ... the smith also sitting by the anvil and considering the iron-work; the vapor of the fire wasteth his flesh, and he fighteth with the heat of the furnace; the noise of the hammer and the anvil is ever in his ears, and his eyes look still upon the pattern of the thing that he maketh. He setteth his mind to finish his work, and watcheth to polish it perfectly. So doth the potter, sitting at his work and turning the wheel. . . . All these trust to their hands, and every one is wise in his work. . . . Without these cannot a city be inhabited. They shall not sit on the judge's seat, nor understand the sentence of judgment, and they will not be found where parables are spoken. But they shall maintain the state of the world, and all their desire is in the work of their craft."

Sir James Kitson, after a suitable acknowledgment of this welcome, delivered the opening address.*

The following papers were read and discussed:

On the Probable Future of the Manufacture of Iron, by Sir Lowthian Bell, Middlesbrough, England.

The Protection of Iron and Steel Ships against Foundering from Injury to their Shells, Including the Use of Armor, by Sir Nathaniel Barnaby, K.C.B., London, England.

The Development of the Marine Engine and the Progress made in Marine Engineering during the past Fifteen Years, by A. E. Seaton, Hull, England (read by title only).

The second session was held at the same place on Friday morning, October 10th, Mr. Alexander Thielen, of the Verein Deutscher Eisenhüttenleute, in the chair, succeeded during the latter part of the session by Mr. H. M. Howe, Vice-President of the A. I. M. E.

After brief introductory remarks by Mr. Thielen, Dr. R. W. Raymond, of New York, called attention to the circumstance that the semi-centennial celebration of the famous mining academy of Leoben, founded by Peter von Tunner, an honorary member of all

* See "British Contributions to the Metallurgy of Iron and Steel," in another part of this volume.

three of the societies here represented, was in progress at that time. Without disparagement of Grüner of France, Åkerman of Sweden, or the brilliant Scheerer—too early lost—of Freiberg, and without forgetting the laborious researches of Percy and Bell, he was confident that all would join in declaring the pre-eminent services rendered to the metallurgy of iron by Tunner, the Nestor of their profession. Dr. Raymond concluded by reading the following letter from Dr. T. Sterry Hunt, a past president of the Institute of Mining Engineers, and a friend and colleague of Peter von Tunner:

NEW YORK, October 1, 1890.

DEAR DR. RAYMOND: The mention of the name of Peter Von Tunner of Leoben awakens in me many pleasant and tender memories. More than thirty-five years since, at the great Exposition at Paris in 1855, I was the youngest of a group of jurors appointed to examine and judge the objects therein belonging to the domain of geology, mining and metallurgy. It is sufficient to mention among my colleagues the names of Elie de Beaumont, Dubrensy, de Chancourtois, Callon, de Vaux, Warington W. Smyth and Peter von Tunner. Of that group of jurors, so far as I am aware, von Tunner and myself are to-day the only survivors. How he endeared himself to me and to us all by his assiduity, his earnestness, his simplicity and his profound knowledge of the principles and details of his profession, I still remember with delight.

Again, in 1876, having as colleagues our honored fellow-members, Sir Lowthian Bell and the lamented Alexander Holley, whose monument in bronze we have so lately unveiled in New York, I was a judge at our Centennial Exhibition at Philadelphia. It was then my good fortune at one of the informal evening gatherings of our Institute of Mining Engineers to receive as our guest the Ritter von Tunner, and to bid him welcome to the Institute which he had already honored by his membership. I well remember his pleasant surprise and his emotion at this greeting after twenty-one years from his former colleague.

And now in the fulness of time are gathered the representatives of three societies, one from Great Britain, one from Germany, and a third, their friendly rival, all devoted to that profession which von Tunner has done so much to render illustrious, and each claiming him as an honorary member. And by a happy coincidence they meet in the great center of the American iron industry on the day Leoben celebrates the jubilee of its world-famed metallurgical school and of its founder, Peter von Tunner. What more fitting and proper than that we should send across the sea on this occasion our hearty greetings. I, therefore, venture to propose a telegraphic message from our international session conveying our congratulations to von Tunner and to Leoben.

Yours truly,

T. STERRY HUNT.

DR. RAYMOND moved that a cable-dispatch be sent to Professor Hoefer, the orator of the day at Leoben, conveying the congratulations of this meeting to Ritter von Tunner and the Leoben Academy in these words:

The International Session of German, British, and American Engineers and Metallurgists send greetings to Tunner and Leoben.

Sir Lowthian Bell: Mr. President, Dr. Raymond has wisely abstained from entering into any details with regard to the eminent services which have been rendered by Ritter von Tunner to the science of metallurgy; and he has done this simply because to enumerate them would be to traverse almost the entire field of scientific investigation of the metallurgy of iron. Sir, I have very great pleasure, indeed, in seconding this motion. My acquaintance with Ritter von Tunner is, perhaps, longer than that of any gentleman whom I have the honor of addressing; it dates back just fifty years, or to the very time when Professor von Tunner was engaged in founding the college at Leoben. Circumstances at first prevented me from availing myself of the advantages I might have derived from an intimate acquaintance with so illustrious a man. But some eighteen or twenty years later it was my pleasure and privilege to read of his researches on the action of the blast-furnace, and it was only when I came to fully appreciate the value of these researches that my mind was directed to a further inquiry into the actions of the most important elements in the manufacture of iron. Since that time we have maintained a frequent correspondence, and I do not overrate in the slightest degree the bonds which have united us when I say that a more affectionate friend I never possessed. At the time when the American Institute of Mining Engineers honored me by electing me as an honorary member, I was not aware, Sir, that Professor Tunner was similarly distinguished. I esteem the honor the greater in having my name coupled with one so illustrious; but I am surprised, Sir, that the small services which I may have rendered to the science of metallurgy were not completely eclipsed by the gentleman who received that mark of distinction at the hands of the mining engineers of the United States. I now beg to second in the most cordial manner the motion just submitted to this meeting.

The motion was unanimously carried by a rising vote.

The Chairman (Mr. Thielen) then offered the following resolution, which he called on Mr. Gilchrist to second:

"*Resolved*, That this International Assembly recognizes with gratitude the services rendered to the progress of the metallurgy of steel by the labors of the late Professor Grüner, Alexander L. Holley, C. W. Siemens, Sidney Gilchrist Thomas, John Percy, and Daniel Adamson; and that the Secretary of this session be instructed

to communicate officially to the surviving representatives of these distinguished men the expression of this feeling."

In seconding this resolution, Mr. Percy C. Gilchrist alluded to the gentlemen named in it substantially as follows:

Grüner, that great French metallurgist, whose writings will always be remembered for the great principles they set forth, and who so clearly foresaw how phosphorus could be eliminated when pig-iron was to be converted into steel.

Holley, the great steel-making engineer, the keen observer, and critic of our European practice, who, with the able co-operation of his American colleagues, enabled our cousins on this side the ocean to obtain the maximum output from a given plant without the sacrifice of quality.

Siemens, one of the most versatile inventors the world has seen, whose fame is forever associated, among his other great achievements, with the invention of the regenerative furnace and the establishment of the open-hearth manufacture of steel.

Thomas, who independently discovered the practical method of eliminating phosphorus from phosphoric pig-iron, and inaugurated the basic process which bears his name.

Percy, our grand old English metallurgist, who was the first to codify our knowledge and experience in the manufacture of iron and steel.

Adamson, who added to other claims upon our admiration and affection that of his influential example and argument in favor of the general employment of steel for boiler-plates; and who, as the President of the Iron and Steel Institute, had so much at heart, and did so much, to bring about the International meeting which this session crowns and concludes.

The resolution was then unanimously adopted.

The following papers were then presented and discussed:

The Progress of German Practice in the Metallurgy of Iron and Steel since 1876, with Special Reference to the Basic Process, by Prof. Hermann Wedding, Berlin, Germany.

On the Darby Process of Recarburization, by A. Thielen, Ruhrort, Germany.

International Standards for the Analysis of Iron and Steel: Notes on the Work of the American Committee, by John W. Langley, Pittsburgh, Pa.

Aërial Wire Ropeways, by J. Pohlig, Cologne, Germany.

The Iron-Ores of the United States, by T. Sterry Hunt, New York City (read by title only).

After the passage of suitable resolutions of acknowledgment for courtesies and hospitalities received, the session was adjourned.

Excursions and Entertainments.

An extraordinarily interesting and elaborate programme was provided by the local Reception Committee:

Thursday, October 9th, after the close of the first session.—1. Excursion over the Pittsburgh and Western Railroad to the Wildwood oil-field, where gas and petroleum wells were seen in various stages of evolution, and the operation of "shooting" a well was witnessed.

2. Visit to the Pittsburgh Exposition.

3. Excursion by steamer down the Ohio river to the Davis Island Dam and the new Ohio Connecting Bridge.

4. Excursion over the Pennsylvania Railroad to the new shops of the Westinghouse Air-Brake Company at Wilmerding.

5. Excursion over the Baltimore and Ohio Railroad to the mines of the First Pool Monongahela Gas Coal Company, where the practical application of electricity to mining was witnessed.

6. Excursion over the West Penn Railroad to the open-hearth steel plant of the Spang Steel and Iron Company at Etna.

Also, visits by individuals and small parties to the works of the Westinghouse Electric Company; Sable Iron Works; Mackintosh, Hemphill & Co.; Pittsburgh Steel Casting Company; Park Bro. & Co.; Soho Iron Mills; La Belle Steel Works; Linden Steel Company; Singer, Nimick & Co.; Oliver Iron and Steel Company; Oliver and Roberts Wire Company; American Iron Works; and several other establishments of interest.

In the evening, a brilliant reception at the Duquesne Club was tendered by the Reception Committee.

Friday, October 10th, after the close of the second session.—Excursion up the Monongahela river, on the steamer Mayflower, to the Edgar Thomson Steel Works of Carnegie Bros. & Co., and the Homestead Steel Works of Carnegie, Phipps & Co. A collation was served on board, and on the return trip, at 7.45 P.M., a beautiful pyrotechnic display of natural gas, arranged at Hay's Station by the Philadelphia Company, was witnessed from the boat.

Saturday, October 11th.—A choice of two all-day excursions:

1. By special train *via* the Pennsylvania Railroad to the Connellsville coke region. Luncheon on the train.

2. By special train *via* the West Penn Railroad and the Allegheny Valley Railroad to many of the works along the Allegheny river. These included the Isabella blast-furnaces at Etna; the immense works of the Pittsburgh Plate Glass Company at Tarentum; the Lucy blast-furnaces; Crescent Steel Works; Keystone Bridge Works; Carbon Iron Company's works operating a "direct" process; and the Pittsburgh Reduction Company's plant for the manufacture of pure aluminum. Luncheon was served on the train.

During the day the ladies accompanying the visiting members of the several societies were most pleasantly entertained by the ladies of Pittsburgh and Allegheny, and in the evening enjoyed a fine organ recital in Carnegie Hall.

PROCEEDINGS OF THE FIFTY-EIGHTH (ANNUAL) MEETING, NEW YORK CITY, FEBRUARY, 1891.

In accordance with the previous announcement of the Council, the proceedings of this meeting were confined to the necessary annual election of officers and the presentation of the report of the Council.

Messrs. E. Renshaw Bush, S. A. Reed, and George B. Wilkinson were appointed Scrutineers, and after having examined the ballots received, according to Rule V., reported the election of the following persons:

President, John Birkinbine, Philadelphia, Pa. *Vice-Presidents*, S. B. Christy, Berkeley, Cal.; E. C. Pechin, Roanoke, Va.; J. C. F. Randolph, New York City. *Managers*, J. B. Mackintosh, New York City; J. A. Porter, Durango, Colo.; J. C. Smock, Trenton, N. J. *Treasurer*, Theo. D. Rand, Philadelphia, Pa. *Secretary*, Rossiter W. Raymond, New York City.

The report of the Council was then presented, and the meeting adjourned.

Annual Report of the Council.

In accordance with the decision of the Council, announced in its last annual report, no spring meeting was held in 1890. Two meet-

ings only have been held during the year: the twentieth annual meeting (being the fifty-sixth of the Institute), at Washington, in February, which was made particularly enjoyable by a large attendance, excellent papers, and delightful weather; and a memorable September meeting in New York, immediately preceding the American meeting of the Iron and Steel Institute in this city, as announced in the last annual report, and followed early in October by International Sessions in Pittsburgh, in which the Iron and Steel Institute and the Verein Deutscher Eisenhüttenleute took part. As soon as the preliminaries of the autumn meeting had been agreed upon with the Council of the Iron and Steel Institute, invitations were extended to the members of other foreign societies to visit our shores at the same time, and, to our very great gratification, the acceptances were so numerous, especially from the members of the Verein Deutscher Eisenhüttenleute, that that distinguished body was virtually represented in its full official capacity, and imparted to the international sessions at Pittsburgh a broader character than could have been hoped for, at the same time enriching our *Transactions* with able and valuable contributions.

In the interval between our autumn meeting in New York and the international sessions in Pittsburgh, and for several weeks after the latter, the members of this Institute and the foreign visitors—numbering over 500—were guests of the American Reception Committee, which had been organized by the joint action of this and kindred engineering societies, the American Iron and Steel Association and the American iron and steel trade generally, for the purpose of entertaining our common guests. An account of the extensive excursion under the auspices of the committee will be published in the *Transactions*. The Council desires to express the thanks of the Institute to the General Committee and, through it, to the various local committees whose efficient co-operation contributed so much to the success of the excursion, and to recognize with specially grateful appreciation the indefatigable zeal of that Committee's Secretary, Mr. C. Kirchhoff; the Chairman of its Transportation Committee, Mr. William P. Shinn; and its Treasurer, Mr. George A. Crocker, all well-known members of the Institute.

The financial statement of the Secretary and Treasurer shows receipts for the year from all sources, including the balance at the beginning of the year, of $32,930.46, and expenditures of $32,057.56, leaving a surplus of $872.90. In addition to this the Treasurer holds $7498 in interest-bearing securities belonging to the Institute,

as follows: United States 4 per cent. bonds, par value $2900; certificate of deposit, Real Estate Title Insurance and Trust Company (resulting from the conversion of $4400 of United States 4½ per cent. bonds), $4598, making the total amount of cash on hand and securities at par value, $8370.90.

The detailed statement is as follows:

Statement of the Secretary and Treasurer, of Receipts and Disbursements from February 1, 1890, *to January* 31, 1891.

Balance from statement, February 1, 1890,		$10,509 88
Received for dues from members and associates,	$18,289 87	
Received for life memberships,	1,100 00	
Received for sale of publications,	1,484 74	
Received for binding *Transactions*,	87 54	
Received for author's pamphlets,	411 00	
Received for author's engraving and electrotyping,	178 01	
Received from Holley Memorial committee,	331 55	
		$21,882 71
Received for interest on United States bonds and deposits,		537 87
		$32,930 46
Paid for printing Vol. XVIII. of *Transactions*,	$3,663 35	
Paid for printing pamphlet editions of papers,	3,406 49	
Paid for printing author's editions of papers,	462 75	
Paid for printing list of members,	172 50	
Paid for printing mailing list,	103 00	
Paid for printing circulars and ballots,	204 21	
Paid for binding Vol. XVIII. of *Transactions*,	1,751 33	
Paid for binding other volumes of *Transactions*,	366 80	
Paid for binding of exchanges,	114 25	
Paid for engraving and electrotyping,	4,372 87	
Paid for postage, including P. O. box rent,	1,261 95	
Paid for stationery,	572 65	
Carried forward,	$16,452 15	

Brought forward,	$16,452 15
Paid for rent of offices,	800 00
Paid for express and freight charges,	204 47
Paid for janitor's fees,	102 00
Paid for telegrams and car-fares,	17 52
Paid for insurance,	9 00
Paid for coal, ice, and porters,	76 51
Paid for salaries—Secretary, assistants, and stenographers,	7,891 64
Paid for expenses of Secretary,	362 05
Paid for storage on *Transactions*,	158 77
Paid for Vol. XIX. bank folio paper,	1,306 25
Paid for miscellaneous printing,	225 00
Paid for telephone,	77 70
Paid for gas,	10 62
Paid for miscellaneous expense, mailing pamphlets,	59 40
Paid for U. S. Ex. Co., ex. batch and vol.,	518 30
	$28,271 38

Special Expenses for Iron and Steel Institute Meeting:

Paid for printing pamphlets, 600-edition,	$736 78		
Paid for Bureau of Information,	794 41		
Paid for hire of halls, stenographer, and lantern,	1,266 37		
Paid for extra editorial work, Ex. of Secretary,	715 65		
Paid for stationery and sundries,	97 40		
Paid for express and porters,	122 49		
Paid for telegrams and messengers,	22 58		
Paid for list of excursionists,	30 50		
		3,786 18	
			$32,057 56
Balance,			872 90
			$32,930 46

The total expenditures of the year, it will be observed, were larger by more than $13,000 than those of the preceding year. This apparent difference is in part due to the purchase of large quantities of special plate-paper, costing in the aggregate $1943.75. This paper is manufactured for the Institute, and is ordered in lots costing over $600 each. Last year none was required, but this year

three lots have been purchased, and the supply on hand will probably last through more than the next year.

The expenses of the Bureau of Information, hire of halls and holding of meetings, amounting to $3786.18, are directly chargeable to the recent meeting of the Iron and Steel Institute in this country, and to the International sessions, in which that society and the Verein Deutscher Eisenhüttenleute were represented, at Pittsburgh. But the above sum does not by any means cover the expense of the Institute in this connection. The increase of printing, engraving, postage, etc., in short, nearly all the items constituting the increased outlay of the year, may be traced to the same cause.

It is scarcely necessary to say that no expenses of social receptions or entertainment of the members or the foreign guests of the Institute have been paid out of its treasury. These matters were in the hands of a general reception committee and of various local committees, representing, not the Institute merely, but other technical societies, the iron and steel trades, and our citizens generally. The Institute can claim no credit (except so far as its individual members may have deserved it by their personal labors and contributions) for the hospitalities, organized and carried out at an expenditure of over $150,000, by these general and local committees. Nor, on the other hand, can any member of the Institute who may not have participated in the excursions and entertainments of October last, feel that money contributed in dues for the benefit of all members through the regular operations of the Institute was diverted to the pleasure and advantage of a few. The expenditures of the Institute have been confined to the usual legitimate objects, and no precedent has been set for the employment of its funds outside of that category. This point is emphasized, not by reason of any complaint or criticism. On the contrary, the enthusiasm with which members regarded the proposed visit of our many and distinguished foreign guests was such as to make it probable that any contribution towards their entertainment would have been applauded. But the Council felt the importance of adhering to the established rule, and strengthening the precedents of the past in its support.

Attention should also be called to the circumstance that the items of salaries, rent, etc., show no increase, and that the normal expenses of the Institute have, therefore, not been increased beyond the increase due to larger editions of larger and more expensive publications.

The delivery of Volume XVIII. bound without extra charge to members, involved an extra cost of $1751.33. This practice will be continued during the present year, in the belief that the Institute can afford it.

The amendment to Rule II. authorizing, at the option of the Council, the election of new members through the mail, has been adopted by a vote of the Institute by mail, as provided in Rule VIII.; two elections of members have been conducted under this new provision.

Volume XVIII. was the largest but one ever published by the Institute. In consequence of its size, the list of members has again been omitted, but such a list has been issued in pamphlet form. By the courtesy of the Iron and Steel Institute, all of the papers and debates contributed to its New York meeting will be published in Volume XIX. of the *Transactions*. Three thousand six hundred copies of this volume will be printed; a copy will be presented to each of our late foreign guests, leaving an unusually large surplus for sale. It is believed that this volume will be universally admitted to excel, in value as well as variety, any previous publications of its class, and that it will be received by the members of the Institute as fully justifying the great labor and expense involved in its preparation.

The changes in membership during the year are shown in the following table:

		H. M.	F. M.	M.	A.	Totals.
At date of last report		10	45	1749	164	1968
Gains,	By Election	.....	.....	179	31	210
	" Change of Status	3	.....	5	.....	8
	" Reinstatement	.....	.....	1	.....	1
Losses	" Resignation	.....	.....	13	7	20
	" Dropping	.....	.....	45	6	51
	" Change of Status	.....	3	.....	5	8
	" Deaths	1	.....	21	4	26
Total Gains		3	.....	185	31	219
" Losses		1	3	79	22	105
Present Membership		12	42	1855	173	2082

The list of deaths comprises the names of Bergrath Adolph Patera, Honorary Member; G. Billing, Thomas G. Boyle, Michael Carroll, Thomas P. Conant, Isaac Fegeley, H. H. Fisher, Robert G. Ford, Frank E. Hopke, Thomas H. Halbert, Henry C. Kriete, D. B. Lienau, W. R. Maffet, Charles W. Matthews, O. E. Michaelis,

Reuben Miller, William Oliver, Augustus Pettibone, A. N. Rogers, A. B. Stollmeyer, C. B. Von Tuyll, M. D. Wheeler, Members; and Augustus C. Brown, Cyrus Butler, R. Gatewood, and J. H. Irwin, Associates.

Proceedings of the Meeting of the Iron and Steel Institute, New York, October, 1890.

The following is an outline of the proceedings of this meeting, more fully reported in the *Journal* of the above Society. The papers, addresses and discussions of technical value, are given in full in the present volume.

Wednesday, October 1.—Sir James Kitson, Bart., President of the Institute in the chair. An address of welcome by Mr. Andrew Carnegie, chairman of the American Reception Committee, was followed by a response from the president. Addresses of welcome and invitation were made by Mr. Archibald Blue, Canadian Deputy Minister of Agriculture, and Dr. A. C. Selwyn, Chief of the Canadian Geological Survey, on behalf of the authorities of the Dominion. After the election of Sir Frederick Abel as the next president, and of a large number of new members, a paper on Development of American Blast-Furnaces, by Mr. James Gayley, of Braddock, Pa., was read and discussed.

Thursday, October 2.—After the conclusion of the discussion of Mr. Gayley's paper, the presentation to Hon. Abram S. Hewitt of the Bessemer gold medal was made in an appropriate address by Sir James Kitson, and acknowledged by Mr. Hewitt, who delivered in reply an address describing the far-reaching results of Bessemer's invention. The following paper was then read and discussed:

Notes on the Bessemer Process, by Henry M. Howe, Boston, Mass.

Friday, October 3.—The following papers were read and discussed:

On Aluminum Steel, by R. A. Hadfield, Sheffield.

On Welding by Electricity, by Professor Elihu Thomson, Lynn, Mass.

On the Wear of Metal, as Influenced by its Chemical and Physical Properties, by C. B. Dudley, Altoona, Pa.

The following papers were read by title:

On Fuel-Gas and some of its applications, by Burdett Loomis, Hartford, Conn.

On Spirally-Welded Steel Tubes, by James C. Bayles, East Orange, N. J.

On the Inspection of Materials of Construction in the United States, by G. H. Clapp and Alfred E. Hunt, Pittsburgh, Pa.

On the Iron-Ores of Virginia, and their Development, by E. C. Pechin, Roanoke, Va.

On Massicks and Crooke's American Patent Fire-Brick Hot-Blast Stoves, by Walter Crooke, Millom, Cumberland.

General Excursions Arranged by the American Reception Committee.

The autumn of 1890 was signalized by a remarkable excursion of three weeks' duration, in which several hundred foreign and American engineers, accompanied by many ladies, took part as the guests of a general reception committee. Early in 1890 the Iron and Steel Institute of Great Britain had accepted an invitation of the American Institute of Mining Engineers to hold its next annual meeting in New York. On the receipt of that acceptance the members of other foreign societies had been invited to visit the United States at the same time and partake of the hospitalities which were extended with equal cordiality to all. To this invitation the members of the *Verein Deutscher Eisenhüttenleute* responded in such welcome numbers that the excursion party exceeded in size and representative character the most sanguine expectations of its organizers, and eventually included about five hundred foreign and more than one hundred American members. The initiative taken by the Institute of Mining Engineers was heartly supported by all kindred societies, and by the Iron and Steel Association and the iron trade generally, and resulted in the formation of an American Reception Committee, with the following representative membership:

Andrew Carnegie, New York, *Chairman;* George A. Crocker, New York, *Treasurer;* C. Kirchoff, Jr., New York, *Secretary;* W. P. Shinn, Pittsburgh, Pa., *Chairman of Committee on Transportation.*

Central Committee.—Hon. Abram S. Hewitt, President of the American Institute of Mining Engineers, New York; B. F. Jones, President of the American Iron and Steel Association, Pittsburgh, Pa.; W. P. Shinn, President of the American

Society of Civil Engineers, Pittsburgh, Pa.; Oberlin Smith, President of the American Society of Mechanical Engineers, Bridgeton, N. J.; W. N. McGugin, President of the United States Association of Charcoal Iron Workers, Olive Furnace, Lawrence Co., Ohio; Dr. R. W. Raymond, Secretary of the American Institute of Mining Engineers, New York: James M. Swank, Secretary of the American Iron and Steel Association, Philadelphia, Pa.; John Bogart, Secretary of the American Society of Civil Engineers, Albany, N. Y.; F. R. Hutton, Secretary of the American Society of Mechanical Engineers, New York; John Birkinbine, Secretary of the United States Association of Charcoal Iron Workers, Philadelphia, Pa.; S. W. Baldwin, New York; J. C. Bayles, East Orange, N. J.; H. G. Bond, Birmingham, Ala.; James A. Burden, Troy, N. Y.; H. S. Chamberlain, Chattanooga, Tenn.; Wm. Chisholm, Cleveland, Ohio; R. H. Coleman, Lebanon, Pa.; Eckley B. Coxe, Drifton, Pa.; Walter Crafts, Anniston, Ala.; F. H. DeBardeleben, Bessemer, Ala.; John Fritz, Bethlehem, Pa.; George W. Goetz, Milwaukee, Wis.; Abram S. Hewitt, New York; Charles Himrod, Chicago, Ill.; J. F. Holloway, New York; A. E. Hunt, Pittsburgh, Pa.; R. W. Hunt, Chicago, Ill.; Dr. T. Sterry Hunt, New York; Fitzhugh Lee, Lexington, Va.; James F. Lewis, New York; Charles MacDonald, New York; Samuel Mather, Cleveland, Ohio; Dr. Henry Morton, Hoboken, N. J.; Richard A. Parker, Marquette, Mich.; H. S. Pickands, Chicago, Ill.; O. W. Potter, Chicago, Ill.; John H. Ricketson, Pittsburgh, Pa.; Ferd. Schlesinger, Milwaukee, Wis.; A. M. Shook, Nashville, Tenn.; John Stanton, New York; Joseph D. Weeks, Pittsburgh, Pa.; David Williams, New York; F. S. Witherbee, Port Henry, N. Y.

General Committee.—William L. Abbott, Pittsburgh, Pa.; Matthew Addy, Cincinnati, Ohio; Benjamin Atha, Newark, N. J.; George F. Baer, Reading, Pa.; George T. Barns, Philadelphia, Pa.; Irving M. Bean, Milwaukee, Wis.; R. Bentley, Lowellville, Ohio; A. S. Bigelow, Boston, Mass.; C. R. Boyd, Wytheville, Va.; James Bowron, Nashville, Tenn.; Jerome L. Boyer, Reading, Pa.; J. H. Bramwell, Bramwell, W. Va.; George Brooke, Birdsboro, Pa.; Lee Burt, Detroit, Mich.; J. G. Butler, Jr., Youngstown, Ohio; J. Parke Channing, Bessemer, Mich.; O. Chanute, Chicago, Ill.; C. M. Clark, Philadelphia, Pa.; B. G. Clarke, New York; Thomas C. Clarke, New York; F. G. Coggin, Lake Linden, Mich.; Joseph L. Colby, Milwaukee, Wis.; L. S. Colyar, Chattanooga, Tenn.; Edgar S. Cook, Pottstown, Pa.; Edward Cooper, New York; W. E. C. Coxe, Columbus, Ohio; Charles H. Cramp, Philadelphia, Pa.; S. S. Curry, Milwaukee, Wis.; John Daniel, Opechee, Mich.; H. C. Disston, Philadelphia, Pa.; Charles Donnelly, Pittsburgh, Pa.; Charles B. Dudley, Altoona, Pa.; Alfred Earnshaw, Philadelphia, Pa.; Thomas A. Edison, Orange, N. J.; George A. Ely, Cleveland, Ohio; Theo. N. Ely, Altoona, Pa., Enoch Ensley, Memphis, Tenn.; W. O. Fayerweather, Paterson, N. J.; E. C. Felton, Steelton, Pa.; F. Firmstone, Easton, Pa.; R. Forsyth, Chicago, Ill.; H. C. Frick, Pittsburgh, Pa.; John Fulton, Johnstown, Pa.; Frederick W. Gordon, Philadelphia, Pa.; M. A. Hanna, Cleveland, Ohio; F. J. Hearne, Wheeling, W. Va.; James Hemphill, Pittsburgh, Pa.; T. T. Hillman, Birmingham, Ala.; H. M. Howe, Boston, Mass.; M. R. Hunt, Ashland, Wis.; George Jamme, Dayton, O.; W. H. Jaques, Bethlehem, Pa.; J. E. Johnson, Longdale, Va.; Walter Katte, New York; Walter E. Koch, Sharpsburg, Pa.; L. G. Laureau, New York; J. Tatnall Lea, Philadelphia, Pa.; E. D. Leavitt, Cambridgeport, Mass.; Robert Leckie, Londonderry, N. S.; F. L. Lehman, New York: J. M. Longyear, Marquette, Mich.; Jawood Lukens, Conshohocken, Philadelphia; Andrew S. McCreath, Harrisburg, Pa.; Hon. James McMillen, Washington, D. C.; George McMurty, Pittsburgh, Pa.; George I. Magee, Corning, N. Y.; Alexander Maitland, Negaunee, Mich.;

George W. Maynard, New York; E. D. Meier, St. Louis, Mo.; Lewis Miller, Akron, Ohio; Philip W. Moen, Worcester, Mass.; E. S. Moffat, Scranton, Pa.; Henry G. Morris, Philadelphia, Pa.; William H. Morris, Pottstown, Pa.; A. J. Moxham, Johnstown, Pa.; Hon. F. J. Niedringhaus, Washington, D. C.; Charles Paine, Pittsburgh, Pa.; A. E. W. Painter, Pittsburgh, Pa.; J. P. Pardee, Secaucus, N. J., William G. Park, Pittsburgh, Pa.; Richard Pearce, Denver, Col.; Edmund Pechin, Roanoke, Va.; J. J. Pierce, Sharpsville, Pa.; Joseph C. Platt, New York; T. C. Platt, Waterford, N. Y.; O. W. Potter, Chicago, Ill.; William B. Potter, St. Louis, Mo.; J. Thorpe Potts, Philadelphia, Pa.; John R. Proctor, Frankfort, Ky.; A. C. Rand, New York; David Reeves, Philadelphia, Pa.; R. H. Richards, Boston, Mass.; Charles Ridgely, Springfield, Ill.; Percival Roberts, Jr., Philadelphia, Pa.; F. W. Roebling, Trenton, N. J.; C. M. Roepper, Alliance, Ohio; H. H. Rogers, New York; R. P. Rothwell, New York; W. S. Russel, Detroit, Mich.; D. M. Sabin, Stillwater, Minn.; W. L. Scaife, Pittsburgh, Pa.; Thomas Seddon, Birmingham, Ala.; William Sellers, Philadelphia, Pa.; Frederick Sloss, Birmingham, Ala.; T. Guilford Smith, Buffalo, N. Y.; E. G. Spilsbury, Trenton, N. J.; Louis Lee Stanton, New York; Homer R. Stoughton, Shelby, Ala.; W. J. Taylor, Philadelphia, Pa.; William Thaw, Jr., Pittsburgh, Pa.; John Thomas, Hokendauqua, Pa.; Samuel Thomas, Birmingham, Ala.; R. H. Thurston, Ithaca, N. Y.; A. Tower, Poughkeepsie, N. Y.; Henry R. Towne, Stamford, Conn.; Johnson Vivian, Calumet, Mich.; Theo. Voorhees, New York; William H. Wallace, New York; Edward Walsh, Jr., St. Louis, Mo.; Willard P. Ward, New York; Smith M. Weed, Plattsburgh, N. Y.; S. T. Wellman, Cleveland, Ohio; Austin A. Wheelock, Boston, Mass.; Jerome Wheelock, Worcester, Mass.; D. J. Whittemore, Milwaukee, Wis.; Ralph J. Wick, Lancaster, Ohio; W. H. Wiley, New York; Oliver Williams, Catasauqua, Pa.; Jones Wister, Philadelphia, Pa.; J. P. Witherow, Pittsburgh, Pa.; F. W. Wood, Steelton, Pa.; Walter Wood, Philadelphia, Pa.; Jacob Yost, Iron Gate, Va.

After a series of entertainments and excursions in and about New York, an account of which will be found in the proceedings of the New York meeting of the Institute of Mining Engineers, the tour was begun on Saturday morning, October 4th, by the departure of the entire party for Philadelphia in a special train via the Pennsylvania Railroad. Arriving at Tacony in the outskirts of the city, the extensive file and saw works of Henry Disston & Sons was first visited, and then a delightful excursion and luncheon were enjoyed on the Delaware river, at the invitation of the Executive Committee of the Engineers of Philadelphia and Eastern Pennsylvania. This committee, consisting of Joseph D. Potts, *Chairman;* John Birkinbine, *Secretary;* J. Tatnall Lea, Eckley B. Coxe, James M. Hibbs, and Jones Wister, had charge of all the arrangements for the entertainment of the party from the 4th to the 7th of October. Monday, October 6th, was devoted to visiting the Baldwin Locomotive Works, the machine shops of Wm. Sellers & Co., and many other manufacturing establishments of Philadelphia. A brilliant evening

reception at the Academy of Fine Arts brought the stay in that city to a pleasant close; on Tuesday morning the party boarded three special trains, comprising in all 27 Pullman cars that were destined to convey most of the travellers over the 3000 miles of their excursion route. The day was spent in interesting visits to the Cornwall iron-ore banks, the Colebrook blast-furnaces and Mount Gretna. During the night the party was conveyed to Altoona, Pa., where on Wednesday morning, October 8th, under the guidance of Theo. N. Ely, and Dr. C. B. Dudley of the Local Committee, three hours were given to visiting the mammoth shops of the Pennsylvania Railroad. At noon the journey was continued, beautiful weather favoring the travellers in their ride across the Alleghenies, which were arrayed in the richly tinted hues of early autumn. A stop was made at Johnstown, and although only a short time was at the disposal of the visitors, the efficient arangements of the Local Committee allowed them to enjoy a trip up and down the Conemaugh valley on platform-cars, and visit the iron and steel plants of the Cambria Iron Company, the Haws silica and fire-brick works and the rolling-mill and switch-works of the Johnson Company. The members of the committee were J. H. Geer, *Chairman;* H. H. Weaver, *Secretary;* J. J. Fronheiser, John Fulton, Alex. Hamilton, A. J. Haws, W. McLain, Jas. McMillen, M. G. Moore, Jos. Morgan, Jr., T. T. Morrell, A. J. Moxham, C. S. Price, Geo. Thackary. On the same evening the party arrived in Pittsburgh, and for the next three days was busily engaged in carrying out an extremely interesting programme, which will be found in the proceedings of the Pittsburgh International sessions (p. xxiv).

Resuming their journey on Sunday evening, October 12th, the visitors reached Chicago on the morning of the 13th, and were received with a cordial welcome by representatives of the city government and by the Chicago committee. The officers of the latter, and the chairmen of the various sub-committees composing it were O. W. Potter, *General Chairman;* R. W. Hunt, *Secretary and Treasurer;* H. S. Pickands, W. R. Stirling, O. Chanute, Hosea Webster, G. W. Cope, Morris Sellers, G. F. Brown, C. H. Ferry, C. B. Holmes, A. Onderdonk, W. T. Baker, D. H. Burnham, D. Adler, C. H. McCormick, W. J. Chalmers and J. E. Deering. Thanks to the excellent arrangements of the committee, a marvellously short time sufficed to accomplish the task of comfortably quartering the large numbers who had debarked from the trains. After a formal reception by Mayor Cregier at the Palmer House, the visitors were

driven through the parks and boulevards, to the Washington Park club-house, where they were regaled with a bountiful repast, while comfortably seated and enjoying fine music and several brilliant speeches. During the day various excursions were undertaken by separate groups, the programme allowing a selection from a large number, which included visits to the Chicago Tire and Spring Works, the McCormick and the Deering Reaper Works, Fraser & Chalmers' great Machine Works, the new water tunnel under the lake, etc. In the evening a concert and reception were held in the Auditorium building, where an opportunity was afforded for inspecting the magnificent appointments of this latest of Chicago architectural achievements.

Tuesday, October 14th, was ushered in by clear skies and bracing winds, in refreshing contrast to the rains which had marred a portion of the preceding day. The event on the programme was an excursion to the South Chicago works of the Illinois Steel Company, stopping *en route* at the Fowler Pressed Steel works, and to the various enterprises located at Pullman. There were also a number of special excursions to points of interest, including the stock-yards, meat-packing establishments, grain-elevators, etc. It was a day of thorough enjoyment, and at its close the universal appreciation of Chicago's munificent hospitality was sobered only by a sense of regret that the stay in Lake city was unavoidably of such short duration.

On Tuesday evening, the excursion-party was divided, five-eighths of its members electing a tour through the mining and metallurgical regions of the New South, while the remainder preferred a trip to Lake Superior. After an interval of ten days, both branches of the party joined forces again in Washington, and made their last excursion on schedule-time.

The Northern Tour.

In a new and superbly equipped train of Wagner vestibuled-coaches, the northern tourists were taken, over-night, *via* the Chicago and Northwestern Railroad to the Menominee iron-range, where they spent Wednesday, October 15th, under the guidance of F. Schlesinger and G. W. Goetz, in visiting the mines and surface-works of the Chapin Mining Company, at Iron Mountain, and the celebrated hydraulic power-plant at Quinnessec Falls, which furnishes the mines with compressed air. The party was very hospitably entertained at luncheon, by Mr. J. L. Buel, on behalf of the

Chapin Company, and proceeded, in the afternoon, to Hurley, Wisconsin, in the Gogebic range. The Norrie, Colby, and Aurora mines were there visited on the 16th, interest being particularly directed to the operation of a novel steam-shovel which is employed to load cars from the stock-piles at the rate of twenty tons in four minutes. Late in the afternoon, the train departed for Houghton, in the copper-country of Michigan.

A diversified and highly interesting itinerary had been prepared for October 17th and 18th, by the indefatigable local committee of the copper-region, composed of:

Jacob W. Alt, Jacob Baer, Henry Brett, D. Cavan, T. L. Chadbourne, Will A. Childs, F. G. Coggin, R. G. Collins, J. R. Cooper, J. N. Cox, Joseph Croze, Capt. John Daniell, A. David, James R. Dee, F. W. Denton, Henry Drittler, Sr., Capt. John Duncan, Wm. A. Dunn, James Dunstan, Hon. T. B. Dunstan, A. D. Edwards, R. M. Edwards ,R. R. Goodell, A. R. Gray, Joseph Gregory, Capt. S. B. Harris, A. E. Haynes, Joseph Hennes, R. M. Hoar, C. J. Hodge, Arno Jaehnig, B. Judkins, B. T. Judkins, Harry F. Keller, Edgar Kidwell, Frank Klepetko, D. Kloeckner, A. C. Lane, William Lapp, Alex. Loranger, J. P. Mason, J. H. Moyle, Capt. W. E. Parnall, M. B. Patch, H. B. Patton, Graham Pope, A. F. Rees, Edward Ryan, Philip Scheuermann, J. H. Seager, F. F. Sharpless, George C. Shelden, Frank McM. Stanton, John Stanton, Hon. J. W. Stone, J. B. Sturgis, Capt. Wm. Tonkin, William Veal, Capt. Johnson Vivian, Prof. M. E. Wadsworth, C. A. Wright, J. N. Wright.

The first day was spent in visiting the Calumet and Hecla smelting-works and stamp-mills, and the imposing surface-plants of the Calumet and Hecla, and the Tamarack mines. A train of platform-cars conveyed the visitors from point to point throughout the day; by noon, their ride in the crisp, invigorating air of Lake Superior made the vision of long tables loaded with inviting edibles, and seats for everybody, especially welcome, as the party was ushered into a large banquet-hall. The luncheon was enjoyed with a rare relish, even to the Lake Superior "water," with which the *menu* ended. The evening was given up to a thoroughly enjoyable reception by the ladies of Houghton county, their invitations to the visitors having been issued on plates of polished copper, attached by silk ribbon to daintily printed musical programmes.

On the second day, a choice of several excursions was offered, including visits to the Quincy, Franklin, Atlantic, and Tamarack mines and stamp-mills, and the Tamarack-Osceola wire- and plate-

mills. A number of the party took advantage of this opportunity to spend the whole day underground, while the majority devoted a part of the time to an inspection of the Michigan School of Mines, at Houghton.

Sunday was observed as a day of rest. In the evening the excursionists took their leave, and proceeded to the Marquette iron region, where, under the energetic and carefully systematized management of R. A. Parker, *Chairman*, Geo. A. Newett, *Secretary*, and C. H. Hall, Peter Wright, A. Maitland, F. P. Mills, S. Mitchell, J. Parke Channing, Alfred Kidder, J. M. Longyear, E. C. Anthony, and Peter Pascoe, of the local committee, two days were very agreeably and profitably spent in visiting the most important producing mines of the range. Monday, October 20th, was devoted to the Michigamme, Champion, and Republican mines—all the mining engineers of the party availing themselves of the facilities for going underground. In the evening a very pleasant reception was given at the Nelson House, in Ishpeming, the parlors being tastefully decorated with evergreens, enlivened by the scarlet berries of the mountain-ash, and intertwined with the flags of Great Britain, Germany, and the United States. The Lake Superior, Lake Angeline, Cleveland, and Jackson mines of the Ishpeming and Negaunee basins claimed the visitors' attention on Tuesday. During the afternoon they inspected the ore-docks of Marquette, and one of the great saw-mills of the district, and, as guests of the citizens of Marquette, enjoyed a drive in perfect weather to Presque Isle park, a beautiful promontory jutting out into Lake Superior. A charming evening entertainment at the residence of Peter White concluded a particularly pleasant day. During the night, the train left Marquette for the nickel mines of Sudbury, Ont., stopping *en route* to permit a thorough inspection of the government locks at Sault Ste. Marie.

Thursday, October 24th was devoted to the nickel-mines and smelting-works of the Canadian Copper Co. and the Old Dominion Copper Co., most of the visitors being escorted underground by Messrs. Evans, F. L. Sperry, R. Bell and others. In the evening the train departed for Niagara Falls, where the party was most hospitably received and entertained by Erastus Wiman, and, thanks to his assiduous attentions, enjoyed the sight-seeing in perfect comfort. On Thursday night the journey was continued to Washington, where the northern and southern sections of the excursion were reunited.

The Southern Tour.

Departing from Chicago in three Pullman trains on the evening of October 14th, the south-bound section of the excursion party reached Birmingham, Ala., after a long railroad journey, on Thursday morning, October 16th, and began at once two days of busy sight-seeing by embarking on special trains for a round of the principal points of interest on the programme. Extraordinary efforts had been made by the local committee for the entertainment of their guests, and the result of their labors added greatly to the enjoyment and interest of the visit. The committee-at-large consisted of H. F. DeBardeleben, Edwin Thomas, H. G. Bond, J. H. Aldrich, Thomas Seddon, J. H. Woodward, C. P. Williamson, A. Griggs and W. M. Newbold, who were ably assisted by many other leading men of the district.

The limestone quarries of the Birmingham Mining and Manufacturing Co. were first visited, and then the trains were carried along Red Mountain, and the whole valley in which the city lies was spread before the visitors. The first iron-ore bank to be reached was the Eureka, which afforded an excellent opportunity to observe at its best the Red Mountain vein. Then followed the Blue Creek coal-mines, at Adger, and the mines at Blocton, where considerable coke is made. In the evening a reception was tendered to the guests by the Alabama Club, a feature of the evening being a series of appropriate speeches.

On Friday, the 17th, the furnace-plants, coke-ovens and rolling-mills at Bessemer were the first objective point. Next a brief stop was made at the Woodward furnaces, and then the great plant of the Tennessee Coal, Iron and Railway Co., at Ensley, was seen in full blast. The Pratt coal-mines and coke-ovens of the same company, and the Pioneer Co.'s two furnaces at Thomas, concluded the day's programme.

During the night the party was conveyed to Shelby, Ala., where the visitors were welcomed early on Saturday morning, October 18th, by a local committee composed of R. P. Huger, *Chairman;* T. G. Bush, J. C. Sproul and H. R. Stoughton. The Shelby Co.'s blast-furnace plant and the famous ore-banks, the largest thus far developed in the South, were the points of greatest interest. On leaving Shelby, delays occurred which allowed only a brief stop at the coke-furnace of the Talladega Co., and the charcoal furnaces of the Clifton Co. at Ironaton. Anniston, the gem of that section of the

South, was finally reached, but at so late an hour that very little opportunity was afforded to appreciate its resources and industries.

During the night the trains made the run from Anniston to Chattanooga, Tenn., where the guests were met on their arrival by a large number of the local reception committee, among them being N. Sanders, *Chairman;* L. S. Colyar, W. M. Bowron, N. S. Chamberlain, G. Bittel, J. M. Duncan, J. Dowling, S. B. Lowe, G. W. Oehs, T. Talbot, W. M. Duncan, A. M. Shook, W. T. McGruder, J. Lodge, W. C. Harriman, W. H. Mullins, W. Warner, E. O. Nathurst and G. Jamme.

The day was wonderfully clear, and those who went direct by the broad-gauge railroad to Lookout Mountain had spread before them in all its beauty the grand panorama which has made that mountain famous. The great Lookout Inn had been reopened for the day through the efforts of the Chattanooga committee, and Sunday was thus spent quietly under pleasant auspices. In the afternoon Gen. J. T. Wilder delivered an address at Point Lookout, explanatory of the topography of the country and descriptive of the great events of which it was the scene. The evening passed with singing by ladies and gentlemen of the party, and by the Pullman Singing Society, organized by the German guests, who had held a number of rehearsals during the previous evenings. This was followed by an address by General Wilder, on the geology and resources of Tennessee and adjoining States, with the aid of a great map compiled for the Chattanooga Chamber of Commerce.

On Monday morning, October 20th, the greater number left the mountain on the Lookout Narrow Gauge and Incline, to visit the Chattanooga Foundry and Pipe Works, which are chiefly remarkable for having casting-pits with revolving tables, and the basic steel plant of the Southern Iron Co. The party then embarked on a boat, passing up the Tennessee river to the Citico furnace and the City Water Co.'s pumping and filtering stations.

The afternoon was spent in visiting the battle-fields in the vicinity and attending an informal reception tendered by the citizens at the rooms of the Chamber of Commerce.

The early risers on Tuesday morning, October 21st, found themselves in the mountains of Kentucky, passing along the Cumberland River through Barbourville, famous in vendetta days, and Pineville, one of the many aspirants to fame as a coke and iron-producing center. They had the experience, too, of seeing streets being graded in a field without a house in sight. Then the trains rolled into the

depot at Middlesborough, Ky., the creation of a year, where the party was received by the members of the Local Reception Committee, composed of Alexander A. Arthur, *Chairman;* John M. Brooks, John B. Cary, Edgar Watts and F. J. Hoyle.

The alternative excursions of the forenoon were: (1) to the coal mines and coke ovens *via* the Belt Railroad; (2) to Arthur Cumberland Gap, park and cave; (3) to the furnaces and manufacturing plants of Middlesburgh; (4) to Pinnacle. In the afternoon Prof. J. R. Proctor spoke on the geology and resources of the district, and Sir James Kitson expressed the good wishes of the visitors in well-chosen words. Then the trip was resumed, and on Wednesday morning, October 22d, the party found a special train in waiting at Pulaski, Va., to convey them to the famous Gossan lode, under the escort of E. C. Pechin, Fitzhugh Lee, J. E. Johnson, Jacob Yost and J. E. Bramwell, the Virginia Local Committee. The zinc-works and Pulaski furnace first claimed attention, and then the Great Outburst mine, on the lode 48 miles from Pulaski, where the character of the deposit, which is a decomposed pyrrhotite of remarkable extent, could be clearly seen.

In the evening the trains left Pulaski, travelling during the night over the Norfolk and Western Railroad to Pocahontas, on the edge of the Flat Top coal-field. It had been raining heavily during the day, and there was some danger that land-slides might seriously interfere with the movements of the train. Extraordinary efforts were made by E. W. Eddy, general manager of the road, to prevent mishap. The whole line was patrolled during the night by an extra force, and, thanks to the precautions taken, the trip was safely accomplished.

Thursday morning, October 23d, was given over to the coal-mines and coke-plants at Pocahontas, conspicuous among those visited being the collieries of the Southern Virginia Improvement Company. During the afternoon the party proceeded to Roanoke, Va., and on Friday visited Luray Cave, en route to Washington.

On Saturday, October 25th, the southern section of the excursionists joined their northern associates and visited the government departments, the National Museum and various points of interest, under arrangements provided by the local committee, consisting of A. W. Greely, *Chairman;* David T. Day, *Secretary;* T. C. Mendenhall, Prof. Langley, Commander Barber and Major Powell.

In the afternoon the President and Mrs. Harrison, Secretaries Windom, Noble and Rusk received the foreign guests at the White

House. In the evening the citizens of Washington, under the leadership of Major J. W. Powell, Director of the Geological Survey, tendered a reception at the Arlington Hotel, which was counted a great success by all participants.

On Monday morning, October 27th, the party left Washington for Baltimore, Md., where they enjoyed the bountiful hospitality of Major L. S. Bent and F. W. Wood, on behalf of the Pennsylvania Steel Company, and spent a very interesting forenoon in visiting the new and extensive tide-water plant of that company at Sparrow's Point. Many of the party then pursued their journey, arriving in New York in the evening, but a large number, especially of the Southern tourists, accepted the invitation of the citizens of Baltimore, who showed them their harbor facilities and entertained them on a steamer, finally landing them at a point where their Pullman coaches were in readiness to convey them to Niagara Falls, and then return them to New York.

This ended the official programme of the American Reception Committee. About 50 or 60 of the visitors then accepted the invitation of the Thomson-Houston Electric Company, leaving New York on the steamer Pilgrim, on Tuesday evening, October 28th, visiting the plant at Lynn, where they were entertained at luncheon. In the afternoon they enjoyed a ride through the city and suburbs, and in the evening were tendered a banquet at the Vendôme Hotel. Another party of about 150 accepted the hospitality of the Canadian Government and its leading cities, Hamilton, Sudbury, Ottawa, Montreal and Quebec; and some of them stopped on their return route to visit the well-known Lake Champlain iron-ore mines of Witherbee, Sherman & Co., at Port Henry, N. Y.

PAPERS.

THE IRON-ORES OF THE UNITED STATES.

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(International Session, Pittsburgh, October, 1890.)

HAVING been honored by a request to prepare for this occasion some account of the iron-ores on which a great industry has been built up in the United States, and moreover, to make such a record of our present knowledge as may be of value alike to our distinguished guests from abroad and to my colleagues of the American Institute of Mining Engineers, I had at first some doubts how best to approach the task. That among my hearers are many better fitted to discuss the question of our iron-ores from commercial and economic points of view, I am well aware; but I have thought that my studies, chemical, mineralogical and geological, continued during a period of more than forty years, and covering a wider field than those of any other observer, would enable me to present the theme under a somewhat novel aspect, and moreover, to connect it with geological and geographical facts which will give to the description a permanent value. To this end, it is evident that the great coal-fields and their relations to the iron-ores of the more ancient rocks have to be carefully kept in view.

Over nine and a half million tons of pig-iron were produced in the United States during the year ending June 30, 1890, including about half a million tons from imported ores. Of the nine millions, in round numbers, ninety-nine per cent. were made from ores mined to the east of the Mississippi, and the remainder, with insignificant exceptions, in the State of Missouri, near the western bank of that great river. The supply of native ores is, therefore, essentially from the eastern portion of our country. From the Mississippi stretching westward to the frontier range of the Rocky Mountains, and in that great mountain-range itself, are treasures alike of iron and coal which await development; while still farther west the vast American basin, and its Pacific mountain-belt, have as yet attracted attention chiefly for their rich stores of gold, silver, copper and mercury.

Up to the present time, then, it may be said that the iron-industry of

the United States is confined to the great region drained by the Mississippi and Ohio and their tributary rivers southward toward the Gulf, by the northern chain of lakes discharged by the St. Lawrence, and to the Atlantic mountain-belt, whose drainage-waters find their way by more direct channels to the Atlantic. The great physical features of this eastern half of the North American continent are remarkable. Between the eastern front of the Cordilleras in Colorado and the mountains of the Atlantic belt and the Appalachians, which stretch from New England and eastern New York to Alabama, spreads a vast region, traversed by navigable rivers, from Lake Superior southward to the Gulf of Mexico and eastward to the Gulf of St. Lawrence. In one place alone is the great Atlantic barrier broken. This is where the tidal and navigable waters of the Hudson river pass through the Highlands, and, permitted by the sinking away of the Appalachians in southern New York, connect the waters of the Hudson, through the Erie canal, with those of Lake Erie, and through Lake Champlain with the St. Lawrence, affording a system of internal navigation which must always give to the city of New York a commercial supremacy on our eastern shores.

Geologically speaking, the structure of this great interior region is as simple as it is geographically. Nowhere, perhaps, in the old world, are the areas of the principal geological groups so widely extended, nowhere else are displayed such large and accessible coal-fields. The ancient crystalline rocks which, while they constitute the Atlantic belt, skirt the northern borders of the great lakes and the St. Lawrence valley, and to the westward form the range of the Cordilleras, are rich in valuable iron-ores. Small areas of these older iron-bearing rocks rise like islets among the younger strata in Missouri, and in Texas; while the great Appalachian valley, itself abounding in ores of iron, stretches along the western base of the Atlantic belt, which it divides from the parallel Appalachian or Alleghany mountain range. The coal-measures themselves in various parts of their distribution, moreover, abound in ores of iron.

Without here entering into details, it should be said that the important coal-basins of the region in question are all of Paleozoic age; the small basins of newer coals in eastern Virginia and North Carolina, and farther westward, being of little account in the present connection. In like manner, the iron-ores of the region may be said to be Paleozoic or still more ancient. The extensive areas of Mesozoic and Cenozoic rocks in our territory, so far as known, furnish little or no iron-ore; the small portions of tertiary strata with lignite, ochres

and hydrous iron ores, at intervals from Vermont southward along the Appalachian valley, being of geological rather than of economic interest. In the early days of our history small deposits of bog-ore were wrought in many places, but have long since lost their importance. Mention, however, should here be made of the magnetic iron sands, which have a certain historic interest. Found along the shores of Connecticut, the sands early attracted attention in England under the name of the American sand-iron, and in 1763 were noticed alike by the Royal Society of London and the Society of Arts. The sand was even manufactured into malleable iron in a furnace on the Connecticut coast, which was probably a German bloomery. Attempts to work similar sands in the Gulf of St. Lawrence and on Long Island, New York, have been made in our time without pecuniary success.*

Passing over these preliminaries we may now proceed to describe the principal iron-ore horizons of the United States in geological succession, beginning with those of the ancient crystalline rocks which, as they underlie the Paleozoic, have been variously designated as Eozoic, Archæan and Primary. These rocks, as the result of life-long studies, we have been led to divide into several distinct groups or terranes, which appear to have a definite order in the earth's crust, and, moreover, to sustain constant relations to their contained iron-ores; thus giving to these divisions not only universal application but great economic importance. Without entering at length into questions of theoretical geology, it may be said in a few words that we reject for these ancient crystalline rocks alike the so-called metamorphic and the igneous hypotheses of their origin, believing them to have been formed at definite and distinct periods in the world's earlier history, through the intervention of water, but under conditions widely different from those of later sediments.† We, moreover, reject as untenable the notion of the igneous origin of the iron-ores themselves, which appear to be in all cases deposited from water, generally contemporaneous with the enclosing rocks, but more rarely by subsequent processes in fissures, after the manner of mineral veins.

* Those who are curious to follow the details of this history will find it told at length by the writer in the Report Geol. Survey of Canada for 1866–69, pp. 261–269, 280–284. See further, J. M. Swank, *Iron in All Ages*, p. 97, for the early trials in Connecticut.

† An extended statement of the writer's views on this subject will be found in an "Essay on the Origin of Crystalline Rocks and the Crenitic Hypothesis," in his volume entitled *Mineral Physiology and Physiography*, pp. 68–189. Eds. 1886 and 1890.

The great divisions of these stratiform crystalline rocks are designated as follows in ascending order: Laurentian, Norian, Arvonian, Huronian, Montalban and Taconian.* The succession is, however, frequently incomplete from the local absence of one or more of the intermediate divisions. The first of these divisions, named from the Laurentide mountains north of the St. Lawrence, includes also the greater part of the Adirondacks, the Highlands of the Hudson, and the continuation of these along the so-called South Mountain or Blue Ridge, into the Carolinas. We have predominant in this terrane a peculiar type of granitoid gneiss, with highly-inclined stratification, sometimes associated with crystalline limestones in certain portions, and including in beds and lenticular masses conformable to the stratification great deposits of iron-ore, generally magnetic, which are extensively mined in the Adirondacks to the west of Lake Champlain, and farther southward at intervals along the Atlantic belt at least into North Carolina. To this most ancient horizon also belong the similar rocks and ores in Canada, to the north of the St. Lawrence and of Lake Ontario.

These Laurentian magnetites are sometimes exceedingly pure, but at other times they include phosphate of lime, which, in the form of crystalline apatite, marks, in one locality on Lake Champlain, the planes of stratification of the ore. This impurity, however, is of rare occurrence. Iron-pyrites is not unfrequently so abundant as to render a preliminary roasting of the ore necessary, and combined titanium-oxide is sometimes found, though not, so far as known in this terrane, in such amount as to unfit the ores for use in the blast-furnace. The occasional occurrence of disseminated scales of graphite in these Laurentian magnetites is a fact of mineralogical interest, and the presence of intermingled calcite, quartz, orthoclase, pyroxene, and other silicates, notably a chloritic mineral, may be noted. Crystalline specular hematite sometimes replaces the magnetite in these Laurentian rocks. Mention may here be made of franklinite, a mineral resembling magnetite, but including besides the oxide of iron those of manganese and zinc. It is, moreover, generally accompanied by crystalline red zinc-oxide. This ore, elsewhere a mineralogical rarity, is mined to some extent in the Laurentian in New Jersey, and after yielding its zinc by distillation is smelted, yielding a manganesian alloy or spiegeleisen.

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* *Mineral Physiology and Physiography*, pp. 402–425; also Report E, Second Geological Survey of Pennsylvania, *passim*.

Next in ascending order among the Eozoic terranes we have placed the Norian series, so called from the predominance therein of great masses of stratiform norite, a rock essentially composed of labradorite or some closely-related feldspar. These norites are, however, not unaccompanied with gneisses and limestones like those of the older Laurentian, on which the Norian is found to rest unconformably in many localities. So far as known, the iron-ores of this series are highly titanic, and though sometimes holding an admixture of magnetite, are often destitute of magnetism, being a pure ilmenite or menaccanite, and in some cases containing an excess of titanic oxide in the form of crystalline grains, apparently of rutile. These ores, though sometimes abundant, are thus, in the present state of our iron-industry, of little value. They abound in areas of Norian rock in the Adirondack region in Essex county, New York, and have been found under similar conditions in the Black Hills of Wyoming and in various parts of Canada, notably at the Bay of Seven Islands, on Lake St. John, and at Bay St. Paul near Quebec, where an unprofitable attempt was made to smelt them a few years since. Similar ores are found in the Norian rocks in the Isle of Skye, and in Norway, in which country, and in England, they have been smelted to some extent, when not too titaniferous, in admixture with other ores, or else with addition of great amounts of flux. The magnetic ores of Scandinavia occur, so far as known, in a gneissic series apparently identical with the Laurentian.

Third in the ascending series of crystalline terranes we have placed the Arvonian, which may be described as a stratified petrosilex, either jaspery or porphyritic in character, apparently the same with what is called hälleflinta in Sweden, and associated with quartzites, certain argillites, greenish fine-grained schistose rocks, and more rarely with crystalline limestones. To this horizon belong the remarkable varieties of magnetite and red hematite mined at the Iron Mountain, Shepherd Mountain and Pilot Knob in Missouri. I have examined areas of these same rocks in Wisconsin, along the north shore of Lake Superior, in the crystalline belt known as Green Ridge to the west of Gettysburg in Pennsylvania, and at many points along the New England coast from near Boston to the Bay of Fundy, and further east in the Cobequid Hills; noting occasionally the occurrence of iron-ores like those of Missouri, which at points in Maine and in Nova Scotia may assume an economic importance. This series, which was by C. H. Hitchcock included in

the base of the Huronian, is also well developed in Sweden and in North Wales, where it was designated by Hicks as Arvonian.

Passing over the Huronian proper for a moment, only to return to it again, we notice the great terrane which includes the younger gneisses, and the fine-grained granulites and mica-schists of the White Mountain series, which we have designated Montalban. Widely developed in New England, it underlies the city of New York, there resting directly upon the Laurentian gneiss of the Highlands, which is itself seen in the northern parts of the city. The Montalban is well displayed in and around Philadelphia, and forms a large portion of the Blue Ridge throughout its southern extension. Remarkable for granitic veins sometimes carrying tin and other rarer elements, the Montalban abounds in many parts with deposits of iron-pyrites, often cupriferous, but is not rich in iron oxides, though including, in some localities, beds of magnetite.

The Huronian series itself demands special notice because great deposits of certain iron-ores now believed to belong to the distinct Taconian or Taconic series (about to be considered at length) were for a long time referred by geologists to this horizon. The Huronian, a great terrane marked by characteristic serpentinic, dioritic, chloritic and epidotic rocks, and including important deposits of sulphuretted copper-ores, contains, it is true, more or less hematite and magnetite, in some cases at least with considerable admixtures of titanium, and also carries chromic iron-ores, which in various parts of its distribution are of economic importance. The immense deposits of magnetic and specular iron-ores in the vicinity of the great lakes in rocks formerly called Huronian are, however, to the best of the writer's judgment, to be referred to the Taconian.

The study of this series brings us back to the eastern part of the United States, and to the region already defined as the Appalachian valley, lying along the southwestern border of the great Atlantic belt, and traceable from western Vermont as far southward as Alabama. From an early time in the history of American iron-working, the importance as a source of iron of the brown ores of that region was recognized, and they were smelted more than a century since in western Connecticut and in Pennsylvania. These hydrous ores, for the most part limonite, and generally known as brown hematite, are found imbedded in soft deposits more or less clay-like in character, and consisting of certain crystalline schists altered and greatly softened *in situ* by chemical changes, often to a depth of 100 feet or more. These folded and inclined strata,

which have been sheltered from erosion by the belt of older crystalline rocks along their border, merit an especial study alike for themselves and for the ore-deposits which they include. The similarity in form, distribution and mode of occurrence of these hydrous ores is well known to those engaged in the mining of them, and may be advantageously studied in parts of western New England and in Dutchess County, N. Y., in portions of New Jersey, and throughout the great Appalachian valley in Pennsylvania and farther southwards, where they are also associated in many places with anhydrous oxydized ores—magnetite and red hematite. The nature of all these deposits, but especially of the hydrous ores and their decayed strata, as well as their geological age, has been the subject of much vague speculation and unfortunately of great misconceptions on the part of many geologists, nor is it too much to say that an adequate solution of the problems thus raised involves many difficult questions alike of chemistry, of mineralogy and of geological stratigraphy, which go far to explain the confusion still existing, and which we shall endeavor briefly to explain.

Resting in unconformable stratification indifferently upon the Laurentian, the Montalban, or some intermediate terrane, is a series of rocks not less than 5000 feet in thickness (and in many parts greatly exceeding it), consisting in their lower part of quartzites and peculiar soft crystalline schists, and higher in the series including great beds of crystalline limestone, often magnesian, and becoming in many parts a banded or a white marble; the whole with more or less alternation of the soft schists, passing into argillites. This great series was, by the earlier investigators, Maclure, Eaton and Emmons, supposed to be distinct from the Paleozoic rocks above the First or Transition Graywacke, and, with the exception of some intervening argillites, was included, under the names of Granular Quartz rock and Granular Lime rock, in the Primary or Eozoic, with the gneisses, mica-slates, etc. Above the Transition Graywacke, in the Champlain and Mohawk valleys, the researches of certain New York geologists soon after made known the existence of a series of quartzites and limestones directly overlaid by shales and sandstones, constituting the Second Graywacke, not unlike the First, with which these observers confounded it. These divisions, then and since designated Potsdam, Calciferous, Trenton, Utica and Loraine, are highly fossiliferous, and very much less in aggregate thickness than the series elsewhere interposed between the older Primary and the First Graywacke. This series was, moreover, though not without certain

traces of organic life, destitute of the fauna found in the fossiliferous strata above named, but contained many characteristic crystalline minerals. The metamorphic hypothesis was at that time—fifty years since—greatly in vogue, and led many geologists to imagine the Primary or Eozoic rocks to be wholly or in great part the result of a mysterious and unexplained alteration of Paleozoic strata, portions of which were supposed to be changed into the well-known adjacent gneisses and mica-schists. In like manner, it was assumed that the Granular Quartz rock and Granular Lime rock, with their interposed and immediately-overlying schists, below the First Graywacke, were the metamorphosed representatives of the wholly distinct Potsdam-Trenton-Loraine divisions. These have an aggregate thickness of not more than 2000 feet, while the great and very unlike crystalline series, included by H. D. Rogers in Pennsylvania under the names of Primal Slate, Primal Quartzite, Auroral Limestone and Matinal Shale, and having an aggregate thickness of not less than 5000 or 6000 feet, was declared to be their metamorphosed equivalents, extending throughout the great Appalachian valley.

This peculiar and distinct crystalline series constitutes what was subsequently called by Emmons Lower Taconic, and by the writer Taconian; the Upper Taconic of Emmons being the First Graywacke of the earlier writers. The Taconian rocks include mineral species which give to it a distinctive character as well as a great economic importance. Apart from the ores of zinc and of copper, must be mentioned, in the first place, those of iron. Of these, the hydrous ores, limonites or brown hematites, found in the soft decayed strata, are of epigenic origin; that is to say, they result from the alteration *in situ*, through the action of water holding oxygen in solution, either of iron-carbonate (siderite), or of iron-pyrites, large deposits of both of which are met with in the upper part of the series, the carbonate, however, predominating. The percolation of atmospheric waters charged with oxygen, converts by well-known reactions the iron-carbonate into hydrous peroxide; any lime and magnesia present being removed as dissolved carbonate, while the iron is left as a hydrous peroxide, generally limonite. As I have elsewhere pointed out, the conversion of pure iron-carbonate into limonite by this process involves a contraction of nearly twenty per cent., and as this change takes place from without inwards, and is generally accompanied by a rearrangement of the insoluble compound, it happens that the masses of limonite are cavernous and often concentric in structure. In some cases these hollow masses contain nuclei of carbonate, and

large quantities of this unchanged mineral are met with in deep or in protected workings. The presence of manganese-carbonate often gives an admixture of manganese-oxides, which have been found in considerable quantities with these limonites in more than one locality.

Lenticular beds of iron-carbonate which are but superficially changed to limonite are mined at Burden, Columbia Co., N. Y., where they appear a little east of the Hudson in gray white-weathering argillites with some calcareous and arenaceous beds, highly contorted, with eastern dips and often overturned. They have lately been well described by Kimball, who, following the tradition of Mather, refers them to the Hudson-River group, which, as will be shown by a note farther on, was made by Mather to include alike Taconian, Cambrian and Ordovician strata.

Less frequently in these strata are found beds of iron-pyrites, which, under favorable circumstances, have also suffered a process of oxidation resulting in the production of limonite. Several examples of this have been described in mines in Pennsylvania, where waters impregnated with iron-sulphate appear, and unaltered pyrites is found in depth. Their structure will serve to distinguish the pyritous from the sideritic limonites, which latter are by far the more abundant in this series. While the accumulation alike of iron-carbonate or iron-sulphide has, in many cases, been effected by a process of segregation, all evidence in the epigenic processes here noticed points to the conversion of masses already *in situ*.

Crystalline magnetite and compact or scaly red hematite are alike frequently disseminated in the schists and quartzose layers of the lower part of this series, sometimes giving rise to lean siliceous iron-ores, but at other times to deposits of great richness and economic value. To this horizon belong the mines of Cornwall, Boyerstown, Wheatland, Reading and Dillsburg, appearing in the north side, and the Warwick and Jones mines on the south side of the belt of Mesozoic sandstone in Pennsylvania.* Protected by this rock, and its accompanying dykes of dolerite, these remarkable deposits of magnetite have been by certain geologists regarded as of Mesozoic age, but as I have long since shown, careful studies leave no doubt of the correctness of the later view of H. D. Rogers, who referred these ores to his so-called Primal Slate. The mineralogical characters of

* See the author's *Mineral Physiology and Physiography*, pp. 261–268; also p. 538; also his further description of the Cornwall mine, *Trans. Amer. Inst. Mining Engineers*, iv., pp. 319–325; and Report E, Second Geological Survey of Pennsylvania.

these ores are very distinct, and unlike those found in the old gneisses. Softer, more finely granular, and less coherent, they generally include small portions of iron-sulphide, with a little manganese, cobalt and copper; the latter sometimes in such proportions as to be separated with advantage in the process of mining. Considerable deposits of zinc-ore (blende or calamine) are also met with in more than one locality in the limestones of this series.

These ore-bearing Taconian rocks are, however, by no means confined to the Appalachian valley. Even in their southward extension, while the chief belt is traced through eastern Tennessee and into Alabama (where the Taconian asssumes a great thickness along the western base of the older crystalline rocks), several areas of it are found farther eastward on the Atlantic slope, which it probably once covered. One such subordinate belt is traced continuously from the main area in Pennsylvania into North Carolina, where it is one of four Taconian areas east of the Appalachian valley which have there been alternately described as Huronian and Taconian. That of King's Mountain, which lies partly in North and partly in South Carolina, having a thickness of about 5000 feet, is noticeable alike for its marbles, its quartzites, its limonites, and its magnetic and specular iron-ores. It has been independently described by Oscar Lieber as an Itacolumitic group. Farther northward also, Taconian areas occur in New England, along the Bay of Fundy, and in the Cobequid Hills in Nova Scotia, where they include important deposits of iron-ores.*

The great iron-bearing horizon of northern Michigan, which extends into Minnesota and Wisconsin, is in crystalline schists, which were by Murray, and after him by Kimball and Credner, regarded as Huronian.† That they are distinct therefrom and belong to an

* For a detailed account of this matter see in the author's *Mineral Physiology and Physiography*, The Taconic Question in Geology, pp. 517–686; also *Ibid*, 413–415 and "The Taconic Question Restated," *American Naturalist* for February, March and April, 1887. The Itacolumite group of Brazil, to which Lieber has with great probability compared the Taconian, is there one of much importance alike stratigraphically and economically. It is, like the Taconian, auriferous and diamond-bearing, and its itabirite and specular schist represents the similar Taconian ores. The official reports of Lieber are very rare, but a pretty complete analysis of them, as regards this group, will be found in *Mineral Physiology*, etc., pp. 564–569. The Taconian limestone of King's Mountain has long been mined for gold. The series here rests upon Montalban gneisses and mica-schists, including granitic veins and masses which I find to be endogenous, carrying so much cassiterite as to have lately attracted considerable attention.

† The name of the Animikie group was in 1873 applied by the present writer to

overlying series, including what was first distinguished by the writer in 1873 as the Animikie group, and that they coincide alike in mineralogical character and in stratigraphical position with the Taconian, had long been suspected. The independent researches of Rominger and of Irving leave, we think, no doubt that this iron-bearing series, provisionally referred to the Huronian, and including the writer's Animikie, is a great development, in the northwest, of the Taconian series. To this also the writer has referred smaller areas of similar iron-bearing rocks in Hastings county, Ontario, and in St. Lawrence county, New York. The conditions now presented around Lake Superior where erosion has swept away the evidences of sub-aerial decay, while, however, sparing the harder magnetic and specular iron-ores, and in many localities the epigenic limonites, which were there mined in the earlier times, are necessarily very unlike those of the Appalachian valley.

The space which we have thus given to the Taconian ore-deposits is more than justified by their great economic importance, not less than by the curiosities of their geological history. These ores from Lake Superior furnished in 1887 over 44 per cent., and with those from the Cornwall mine, and the brown hematites of the Appalachian valley, considerably over one-half of the total iron-product of the United States.

The evidence drawn from the supposed contiguity of portions of fossiliferous rocks with the Taconian in certain localities within the great valley, is discredited by not less clear evidence of the stratigraphical inferiority of the ore-bearing Taconian; while the absence of such ore-deposits in the same fossiliferous rocks throughout the regions in question may suffice to close the discussion.

The decay of crystalline rocks which has been noticed in connection with the limonite ores of the Taconian, was a process universal in its application (as pointed out by the writer in 1870) and has been the source of all clays, sands and boulders, the subsequent erosion and rearrangement of these by aqueous action being a very important

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the slates and sandstones making the lower divison of the Upper Copper-bearing series; the Huronian, upon which the Animikie was seen to rest unconformably, being the Lower Copper-bearing series of Logan. The sandstone and amygdaloids, which in their turn overlie unconformably the Animikie, were then also first claimed by the writer as a distinct series, named the Keweenaw or Keweenian. Irving's subsequent statement that I regarded the Animikie as newer than the Keweenaw (Fifth An. Rep. U. S. Geol. Survey, 1885, p. 203) is totally at variance alike with the views and the language of the present writer. See *Mineral Physiology*, etc., pp. 414–415, and 578–583.

factor in surface geology.* As we proceed southward through the United States we enter regions where portions of the decayed material still remain undisturbed, not only in the sheltered Appalachian valley but on higher levels. Hence it is that we find great deposits of iron pyrites in Virginia, and farther southward, still enclosed in their walls of decayed gneiss and mica-schist, though already changed into limonite by oxidation. In many cases these superficial masses of oxide have attracted attention as iron-ores, the working of which has not unfrequently disclosed rearranged local accumulations of copper; while at depths of 100 feet or more an unchanged, and often more or less cupriferous pyrites, is met with in the unaltered gneiss. These deposits, though not of great or permanent value as iron-ores, deserve mention the more as they often attract attention disproportionate to their importance. Space may here be found to describe a peculiar occurrence of limonite on Staten Island, New York. Here an area of several square miles has a surface of serpentine, which by decay *in situ* has been converted to a considerable depth into a brownish earthy matter, including concretionary masses and grains of a limonite, which like the underlying serpentine is chromiferous, and has in past times been the seat of a considerable mining industry; the whole presenting a very instructive example of the results of subaerial decay in an area which, probably from Mesozoic time, has been protected from erosion.

Having set forth the chief facts in the geological history of the iron-ores of the great divisions of the crystalline rocks, it remains to notice those ores which are included in the deposits of Paleozoic time. Recognizing in these the now generally accepted divisions into Cambrian, Ordovician, Silurian, Devonian and Carboniferous (the latter including the generally recognized coal-measures but excluding those of Mesozoic age) we note that the First or Transition Graywacke, the Upper Taconic, also designated Hudson-River and Quebec group,† while distinct from the Taconian proper, is limited to certain areas within the Appalachian valley, and with the exception of occasional portions of overlying and included Ordovician and Silurian, is Cambrian in age. It is uncrystalline in character, and is not known to contain any workable deposits of iron-ores whatever.

* See, for the whole question, an essay on "The Decay of Crystalline Rocks," *Mineral Physiology and Physiography*, pp. 246–278.

† See, for a concise statement, "The Geological History of the Quebec Group," *American Geologist* for April, 1890. Also the preface to the second edition of *Mineral Physiology*, etc.

The same may be affirmed of the succeeding Ordovician, including the Trenton limestone and its overlying Utica and Loraine shales. The notion of the supposed identity of this group with the Taconian has already been discussed and rejected in these pages. This Ordovician group passes upward into the true Silurian, consisting in its lower portion of the sandstones which are known in the State of New York as Oneida and Medina, and in conjunction with the underlying Ordovician shales make up what has been called the Second Graywacke; the confounding of which with the First Graywacke has been the cause of many errors in American geology.*

Resting upon the sandstone just named, and between it and the great overlying Niagara limestone of the Silurian, is a subordinate limestone division locally known as the Clinton, which is of interest, as including the first iron-ore of our Paleozoic series. This is a stratum a few feet in thickness in a marine fossiliferous limestone, the organic remains in which are cemented together by red oxide of iron or hematite. This ore-horizon, variously designated as the Clinton, the dyestone and the fossil ore, is traced from the vicinity of the great lakes in the north, eastward beneath and around the Appalachian coal-basin as far as Alabama, and is mined and smelted in many localities. As a fact of geological interest it may be mentioned that the great movements of the strata which in Alabama have subdivided the coal-measures and brought to the surface between the Warrior and the Cahaba coal-fields the brown hematites of the Primal or Taconian, as elsewhere in the great valley, with their enclosing clays, have exposed considerable outcrops of the Clinton ore, which it thus brought into close proximity at once with these higher grade ores, and with the valuable coking-coals of the region. The Clinton or dyestone ore is, however, of inferior quality, and in the deeper workings, where solution has not removed the included calcareous fossils, is often no longer rich in iron. A somewhat

* It was W. W. Mather who fifty years since, by his denial of the existence of the First Graywacke of Eaton, and his attempt to show that this ancient uncrystalline group, together with the whole underlying Taconian, the Trenton limestone, and the Second Graywacke itself, were modifications of the Potsdam-Trenton-Loraine series, or the so-called Champlain division of New York Paleozoic rocks, in various stages of alteration, caused this confusion. From him, and from Vanuxem, came the use of the term Hudson-River group (afterwards changed by Logan to Quebec group) for these schistose rocks of three distinct periods, which were all assumed to be equivalent to the Loraine shales.

similar ore-horizon occurs with the Oriskany sandstone at the base of the Devonian. It is often a brown hematite, very siliceous, but is mined in parts of Virginia, where it yields a valuable iron-ore.

Passing upward into the coal-measures of the Appalachian basin, we meet, at various horizons, iron-ores which may be described, mineralogically, as belonging to two distinct classes. Of these, the most abundant are those consisting essentially of carbonate of iron, either as so-called black-band ore, or as massive or concretionary clay-ironstone, which, near its outcrop, is transformed by peroxidation into limonite, with the loss of any combined lime or magnesia, and the hydration of the iron-oxide. The second type is the anhydrous peroxide or red hematite, which, though more rare, is found in many localities in concretionary masses of such great hardness and compactness as to have attracted the notice of the aborigines, who fashioned them into arrowheads and other weapons.

The concentration of iron-carbonate in rocks by the intervention of organic matter reducing and rendering soluble the ferric oxide diffused through the sediments, has been studied by many chemists, notably by W. B. Rogers, while Shaler has called attention to the conversion of layers of carbonate of lime in the coal-measures by percolation from above; a replacement explained by the chemical reactions observed between carbonate of lime and dissolved bi-carbonate of iron. The subsequent conversion of such iron-carbonates into limonite is familiar, but the generation instead of this of a half-hydrated species, as turgite, or an anhydrous form, as in the compact red hematite found in these Paleozoic strata, is not well understood. The interstratified ores of the coal-measures form a not inconsiderable source of iron in various parts of the great Appalachian coal-fields; while similar ores abound, and were at one time smelted to some extent, in the western coal-field in Kentucky.

It may be said, in general terms, that the abundant supplies of richer and easily-mined ores, in localities not too remote from the coal, have greatly contributed to their development. It remains to be noted that, with the exception of the deposits in the Atlantic belt in New York and New Jersey, the ores of the United States are not favorably situated for the production of iron for foreign markets. This consideration has given an importance to the trade in foreign iron-ores which, notwithstanding the present high duty imposed, has in the last few years assumed considerable proportions. High-grade ores from Spain, Algiers, Italy, Greece, from Cuba, and to a limited

extent from Canada, must be reckoned as factors in our iron-industry.*

The Atlantic belt in New England, and in the province of Quebec, has afforded little iron-ore, though there are undeveloped possibilities in several localities, while there is reason to believe that in the maritime provinces of Nova Scotia and New Brunswick, alike in the Paleozoic basin, with its included coal-measures, and in the older crystalline rocks, there are iron-deposits which, from their proximity to coal, and from their accessibility by sea, may one day assume considerable economic importance.

In bringing to a close this review of the iron-ores of the country, necessarily very meagre and incomplete, no apology is attempted for the manner in which it has been presented, and this for the reason that I believe the end which I have had in view will be best subserved by the geological method here adopted. The geographical distribution of the various ores, their relation to the supplies of coal, and of the centers of production and manufacture alike to home and foreign markets, are obvious to the general student, without any special reference to the geological distribution of the ores. The consideration of these, and of the subordination of the special types of ores to certain geological horizons, are, however, less familiar to our iron-masters, and at the same time will always possess a great significance. Such a study as the present must necessarily be supplemented by an elaborate chemical investigation of these different classes of ores considered alike in their relations to blast-furnace practice and to their respective adaptability to the various branches of iron and steel-industry. I need not remind you of the great amount of information already acquired by the skill of the manufacturers, and the scientific knowledge of our metallurgical chemists. Another hand more trained than my own is needed to attempt the task of discussing it.

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* The following figures will suffice to give a notion of the absolute and relative importance of the importations of iron-ore to the United States for the fiscal years ending June 30, 1887, and June 30, 1888, which amounted respectively to 1,441,774 tons and 917,644 of ores carrying from 50 to 60 per cent. of iron:

	1887.	1888.
Spain,	45.8	45.0
Algeria,	18.8	17 7
Italy,	10 0	10 7
Greece,	4.0	3.7
England,	6.3	6.2
Cuba,	9.6	16.0
British North America,	2.1	1 5

The imports from Cuba for 1888-89 had risen to 34.5 p. c. of 652,032 tons.

EXPLOSIONS FROM UNKNOWN CAUSES.

BY J. C. BAYLES, EAST ORANGE, N. J.

(New York Meeting, September, 1890.)

THE most unsatisfactory occurrences in the experience of a manufacturer are those from which he suffers damage and learns nothing useful. That there are such incidents, and that they occur with annoying frequency, is unfortunately true. An accident which can be understood and explained always carries some consolation with it. However bad the consequences, one finds comfort in reflecting that they might have been worse, and that the knowledge of how to avert a more disastrous calamity from the same cause is worth what it cost. But when an accident occurs, which remains unexplained after anxious days of investigation and sleepless nights of reflection, and which is as liable to occur twice or twenty times, as once, very little satisfaction of any kind can be extracted from it by the most philosophical victim. Three such incidents have come under my notice in one establishment. Fortunately none of them were attended with very serious consequences, as no one was hurt, and the damage to property was slight; but in each instance, loss of life and great destruction were escaped by so narrow a margin as to make them extremely disquieting. I have recorded them in the hope, that from the experience of others may be gained what my own careful investigations have failed to reach—satisfactory explanations.

The first of these curious occurrences was the bursting of a 16-inch pipe carrying air under a compression of about 1 pound. The pipe was made of light galvanized iron with soldered seams. Into it a rotary fan-blower delivered air, and from it smaller pipes were carried to the furnaces. The blower was run continuously. Neither the main pipe nor its branches had any connection with the gas conduits. Both air- and gas-pipes delivered into the furnaces; but although the gas was under much higher compression than the air, there appeared to be no good reason why, having free escape in case of leakage, it should ever make its way back into the air-pipe. One warm afternoon in June the main air-pipe exploded with great vio-

lence. Every window in the mill was blown out, a considerable section of the roof was raised an inch or two, and in several places it was broken through. The pipe was torn into a thousand pieces, and a wagon-load of fragments not larger than my hand were scattered all over the mill. Several of these fragments were driven edgewise into the roof-timbers. The disk closing the end of the pipe was projected against a brick wall with such violence that it remained fastened in place, and is there yet, a mural tablet commemorating the event.

I promptly investigated the accident and learned the following facts: The pipe in which the explosion occurred extended the whole length of the mill. The machines then in use were placed together near the end connected with the blower, leaving some 80 feet of what may be called dead end. It was in this dead end that the explosion occurred. The portion of the pipe from which outlets were taken was substantially uninjured, but 75 feet of the 80 feet beyond the farthest outlet were utterly destroyed. The fact that, with very little mending, the part of the pipe which the explosion had not reached continued for some months to supply the machines with air, shows how local the explosion was; and the damage to the mill building gave sufficient evidence of its violence.

The natural explanation of this explosion is that gas found its way into the air-pipe and was packed away in the dead end, and that when mixed with air in explosive proportions, it reached a furnace and exploded. I can only say that the most rigid investigation failed to explain how the gas got into the air-pipe against the pressure it carried, and why an explosion beginning at a furnace should have restricted its effects to the dead end of the air-pipe. It was undoubtedly a gas- or vapor-explosion, but I can find no other explanation of the presence of gas or vapor than that it was formed by the volatilization of the oil consumed in lubricating the trunnions of the blower. It is conceivable that the large amount of oil consumed by the blower is volatilized, and that it becomes a hydro-carbon gas which would behave like any other gas of similar composition. This gas, being lighter than air, would occupy the upper part of the pipe, and remain undisturbed, while air was drawn from outlets taken from its underside. This light gas may have worked along and accumulated in the dead end of the air-pipe until it reached, in admixture with air, the explosive condition. But whence the spark? And why, if fired by a furnace, was the destructive force of the explosion exerted so far from the point of ignition? This hypothesis assumes that the vola-

tilized or gasified oil of many days' running would remain undiffused for as many nights, until its accumulated volume was great enough to explain the phenomena of the subsequent explosion. The best that can be said of it is that, perhaps, it is better than no theory at all.

Nothing similar has occurred since. We replaced the galvanized iron pipe with a 16-inch steel tube, 400 feet long, to meet the increased requirements of the establishment. All the other conditions remain the same, except that a small opening was left in the end of the pipe which cannot be wholly closed. Whether this is necessary we do not know. The accident taught us nothing whatever; and, so far as we are aware, the same causes are now at work, and may at any time produce like results. The fact that no great damage was done is due to the frail character of the tube in which the explosion occurred. If the 16-inch steel tube should ever be destroyed with equal thoroughness by such an explosion as I have described, I hope I shall be in another State.

The second of the curious actions I shall mention was the explosion of a No. 6 Sturtevant blower. I was a witness of this amusing, though somewhat alarming, occurrence, and can speak of it from personal knowledge. The blower was inside the mill, and was driven by two belts from pulleys on the main line of shafting. It was used to furnish blast for the gas-generators. Some trouble with the main driving-belt necessitated a stoppage of the mill-engine, and the blower stopped. In a few minutes the engine started again, and with it the blower. It had been long in other use, but as this was its first day of service in that position I was naturally curious to see how it worked. So I stood watching it. Suddenly it disappeared. One side passed close to me and lodged against a post. Fragments weighing twenty to fifty pounds were distributed in all directions. The explosion was accompanied by a violent report and succeeded by a dense cloud of yellow-brown, offensive-smelling smoke, which rose to the roof, rolled right and left, and finally escaped at the monitor.

Again I investigated, until there remained no questions to ask. That it was not a centrifugal rupture I know without being told. The conclusion was that during the stoppage of the engine some air-gas from the producers had worked back through the pipe into the blower. When the blast was resumed these products of imperfect combustion were carried with the air-current into the producers, and being mingled in explosive proportions had been fired by contact

with the incandescent fuel and exploded. This explanation was never quite satisfactory to me. An explosion which began in the producer could only reach the blower through two branches of a tee, six feet of vertical pipe, an elbow, twenty-five feet of horizontal pipe under ground, another elbow, six or eight feet of vertical pipe, another elbow, and four feet, more or less, of pipe connected with the outlet of the blower. Some of these pipes were light and some heavy, and the section underground was much larger then the section at either end of the run. If an explosion violent enough to wreck the blower completely had occurred through the whole length of this very circuitous pipe, I should have expected to find some evidence of it in the pipe itself. It was intact. Not a joint was started. Furthermore, as the blower had been running at least four minutes immediately before the explosion, what could have remained in it to explode? The fact was, however, that the blower was shattered, while the pipe was undisturbed, even the delivery-nozzle of the blower remaining coupled to the length of pipe on the mill-floor, which was not thrown out of line. As in the first instance, this explosion taught us nothing.

The third of the series of unexplained accidents consisted of two explosions following one another so closely and under conditions so nearly identical that they may be considered as one episode. In the purification of gas we use purifying boxes of the usual pattern. We have four boxes so connected by the center-seal that we can throw any one of the four out of use when it is necessary to clean it. The gas always passes through three boxes before reaching the gasometer, and one is always kept ready to be filled with fresh iron and brought into use when needed. When the gas shows the presence of impurities or diluents it is time for a change. To make the procedure clear let us suppose the boxes to be numbered 1, 2, 3 and 4, and the gas to be passing through 1, 2 and 3 in the order stated. No. 1 would, of course, become foul first, as it first receives the gas. If a test of gas which has passed No. 3 shows that it is not completely purified, No. 1 is cut out and No. 4 brought into use. The gas would then go through Nos. 2, 3 and 4 in the order stated, and No. 1 would be emptied and refilled in readiness to become the third of the series when the fouling of No. 2 made it necessary to pass the gas through Nos. 3, 4 and 1. In reality, the box to be brought into use is not refilled until it is needed, but otherwise the procedure is as I have described.

One day, the superintendent and the manager had occasion to go into the purifier-house together, and while there the superintendent tried the gas. Getting a reaction indicating the presence of impurities, and finding the fourth, or idle box, ready, he turned the center-seal, cutting out the box which had been the first to receive the gas, and making the clean box the last of the series. The cap of the outlet was left off for the escape of the air, and not screwed on until there was a strong smell of gas, indicating that the air had been expelled. The same thing had been done in the same way hundreds of times. In two or three minutes the third box exploded with great violence. The cover was wrenched loose from the four clamps holding it down; carried up through timbers and roof and dropped again, badly wrecked. The center-seal was canted to one side, allowing a copious escape of gas. The building took fire, and a second explosion in the basement blew out about half the foundations. The second explosion was easily understood. Fortunately, fire-extinguishers and hydraulic jacks saved the building, and except the need of repairing the broken box, the damage was slight. I at once began an investigation, which has lasted ever since. The explosion was undoubtedly due to the ignition of a mixture of gas and air in the box; but how was it ignited? The gas, before reaching the box in which the explosion occurred, had passed through the hydraulic main, two scrubbers, more than 500 feet of unjacketed pipe, and two purifying boxes, each containing three layers of wet sesquioxide of iron. It requires a violent stretch of the imagination to believe that a spark could travel so far under conditions so adverse. The pipe which delivers gas to the boxes is rarely quite cold, but I have never found it more than warm. The tops of the boxes are always cold, and the gas enters the gasometer at atmospheric temperature. While we were speculating as to the cause of this accident, and congratulating ourselves that it was never likely to happen again, another box, the third of the series in use, exploded under exactly similar conditions. A detailed account of one explosion describes the other perfectly.

Matters were getting serious. No one had been hurt; but it was not impossible that the thing would happen under conditions which could not fail to kill somebody. I must find out what was wrong and correct it. So I called in all the experts I could reach. Some were honest enough, after looking the plant over, to confess that they had no explanation to offer. Others gave reasons which would have been satisfactory had they not been at variance with the facts. For

the information of those who may be disposed to speculate as to why these boxes exploded, I may say:

1. The hydraulic main is modelled after the best gas works practice.

2. The scrubbers are adequately supplied with water.

3. The iron in the boxes which exploded was found, on analysis, to contain less than nine per cent of free sulphur, and is still in use.

4. The iron was adequately revived before being replaced, and did not heat in the boxes. After the explosions it was found to be cold.

5. The iron was sufficiently wet.

6. There was no fire in the purifier-house and "no smoking."

Since these two explosions, which occurred in April last, we have had no trouble. There has been no change in the arrangement of the gas plant, for we can discover no way to improve it.

It would have been a great satisfaction to have been able in these instances to follow the sage advice of Hotspur, and out of the nettle danger pluck the flower of safety. But our nettle crop does not seem to be of the flowering variety. If I had investigated less closely I might have reached satisfactory conclusions—perhaps of no more value, however, than the honest doubts I am now willing to confess.

NOTES ON COAL-MINING IN OREGON.

BY R. HENRY NORTON, M.E., PORTLAND, OREGON.

(New York Meeting, September, 1890.)

THE State of Oregon, although admitted into the Union February 14, 1859, is to-day, so far as any practical knowledge of its mineral resources is concerned, almost *terra incognita*, as compared with the other Western States and Territories.

This is partly to be accounted for by the class of emigrants who first settled in the country; they seem to have been, as they still are, a pastoral community, to whom the terms "mines" and "mining" are synonymous with bankruptcy and disaster, and who discourage all attempts to develop the mineral resources of the State. This statement is emphasized by the fact that Oregon has no State Geolo-

gist, while her sister State, Washington (admitted into the Union only last year), has her State Geologist, with a fully-equipped office and laboratory, where those interested in mining may obtain trustworthy information. The only official information concerning Oregon is derived from the United States Geological Reports (to which I am indebted for the analysis of the Coos Bay coal given in this paper), and a pamphlet published by the State in 1888 on the "Resources of the State of Oregon," which, while showing careful research and making conservative statements of the advantages of the State, indicates also that, in the minds of the present State officials, mining continues to occupy, as it has done in the past, a secondary rank.

One quotation will suffice to prove this. There are thirty counties in Oregon, and in nineteen of them outcrops of coal have been discovered, yet in only one county is coal mined for market, and the remaining eighteen are dismissed by this pamphlet with the remark: "Coal has been discovered in this county, but capital and enterprise are needed to develop it."

The question naturally suggests itself, Why should the mining industries of Washington be advancing with such rapid strides? For the year ending June 30, 1889, the output of coal in Oregon was 44,850 tons; that of Washington for the same period was 918,344 tons, more than twenty times as much; and the statistics for the current year will undoubtedly show a much greater difference. Yet the coal-fields of Oregon are not less extensive, and probably of greater intrinsic value, than those of her youthful rival.

The writer has been recently engaged in the examination of coal-lands contiguous to the proposed route of the Oregon Pacific railroad, and the results of his investigations are submitted to the Institute, in the hope that others may be led to continue the inquiry. Since millions of tons of coal are imported annually into San Francisco and Portland, at high cost in ocean-freights and duties, a market for all that can be mined at home is evidently assured. Good bituminous coal readily brings $12 to $14 per ton, and, to the writer's knowledge, has been sold as high as $20 per ton; and the average price of Welsh coke is $22 per ton at these ports. The margin of profit to operators and the commercial benefit to the State which may be expected from the development of the Oregon coal-fields can be readily inferred.

The coals of Oregon, like those of Washington, vary from the crudest lignites, retaining their woody structure, up to a high grade

of bituminous coal, making coke of excellent quality. Up to the present time no anthracite has been discovered in the State.

Commencing at the Pacific Coast, the writer examined first, in accordance with instructions, the country tributary to Yaquina bay, the western terminus of the Oregon Pacific railroad. For the past fifteen years coal has been known to exist in this neighborhood, and has been supposed to be the connecting link between the Nehalem coal-field on the north and the Siuslaw and Coos Bay fields on the south. Yet, although large sums of money had been expended in exploration both by the railroad company and by private individuals, no results of definite value had been reached. The railroad, when completed to an eastern connection, will be some 400 miles long. At present 150 miles east from the coast are in operation. The company maintains, in addition, a line of steamers between Yaquina and San Francisco. Hence it was highly important to determine whether steam-coal of even tolerable quality could be found within easy reach of the coast. The coal for the company's steamers is purchased at San Francisco, and the locomotives all burn wood. Indeed, incredible as it may appear, all the locomotives in Oregon (except those on the Union Pacific railroad), and all the steamers on the rivers in Oregon, numbering several hundreds, are wood-burners.

After three months' steady search, under considerable climatic difficulties (it being the rainy season), a vein of excellent coal was uncovered and traced for nearly two miles up-stream. The country, like most of the Oregon and Washington coal-fields, was a difficult one to prospect in, being rough and broken, with a dense growth of timber on the hillsides, and a deposit in the valley of silt, from 50 to 200 feet deep. The location of this seam of coal was for weeks a puzzling problem. The left bank of the stream is a precipitous bluff, fully 1000 feet high, while the right bank does not exceed 15 feet in height, the summit of the latter presenting an acclivity, gently sloping for about a mile and a half to the foot of the mountains on the other side of the valley. Embedded in the right bank was a block of coal weighing two tons or more, an examination of which soon revealed the fact that it was an isolated mass, and the question was: Where did it come from? It was naturally believed to have fallen from the high cliff on the opposite side of the creek, and many a weary climb was undertaken in search of its original location. As it had been intimated that this was the rock upon which all previous engineers and experts had split, and that it was supposed the present party would also fail in solving the

problem, every member of the party felt that it was incumbent upon him to find that coal *in situ*. Finally it was decided that the coal must be on the right bank, and by sluicing out a small tributary to the main creek and removing several thousand cubic yards of the superincumbent alluvial silt, a seam of excellent coal was exposed nearly four feet thick, and running diagonally across the creek. Unfortunately the financial condition of the railroad company did not permit the immediate further development of this seam or the continued search for others; and the primary object of the expedition having been achieved by the solution of the fuel-problem both for railroad and steamboat service, the party was disbanded.

The following is a section of the coal where it was uncovered:

Coal,	8 inches.
Black shale,	2 "
Coal,	3 feet.
Blue clay,	5 "

An analysis of the Yaquina Bay coal will be found in the table at the conclusion of this paper.

Samples of coal from (1) the Nehalem valley; (2) from a point 18 miles south of Scio, in Linn county; and (3) from the head-waters of the Clackamas river, all west of the Cascade range, have been submitted for the writer's examination at various times. Analyses are not at hand, but judging from physical characters, these coals are closely similar to those of Coos Bay and Yaquina Bay, and have in all probability equal commercial value.

In May last, samples of coal and coke were shown to the writer, which were much superior in appearance to anything he had previously seen in the State. An examination of the property was promptly made, and the result exceeded all expectations. The coal-field in question is situated on the John Day river, east of the Cascade mountains, and seems to be confined to a limited area (some 2000 acres), although systematic exploration may disclose further deposits. The country is a high rolling prairie, frequently broken by deep ravines. Viewed from the summit of the coal-field, it has the appearance of a vast ocean, bearing away to south and west, and covered for hundreds of square miles with deposits of glacial drift, sometimes aggregating several hundreds of feet in thickness. This drift covers the hills to their very summits, thus indicating that it has been deposited since the upheaval of the underlying strata. The

heavy winter rains had washed this drift away on the south side of the claim, and exposed the upper seam, which has since been called the "discovery seam." Developments on this seam proved it to be continuous and regular. It has been traced for more than a mile along the hillside, and shows a uniform thickness of about 20 feet, 14 feet of which is excellent coking coal. The bed has a uniform dip of about 18 degrees to the southeast. This discovery was kept secret by the locators while efforts were made to discover other seams —not without success. About half-way down the mountain, a seam seven feet thick was discovered, and about 300 feet from the base of the mountains, another seam, 11 feet thick, was exposed. At the time of the writer's visit, this latter seam had been opened by a tunnel about 60 feet long, to supply a small local demand.

The exploitation of this coal-field will undoubtedly be of great benefit to the manufacturing industries of the State. It lies about midway between the Oregon Railway and Navigation Co.'s railroad on the north and the Oregon Pacific system on the south, being about sixty miles from each, and about eighty-five miles west of Baker City, a circumstance which is considered as assuring the success of smelting and reduction-works at the latter place. Several lines of railroad are already projected to tap this coal-field.

In considering the analyses given below, it must be borne in mind that they are all from outcrop-samples, except No. 7, which was taken from the face of the tunnel about 60 feet from the surface. While on the ground, the writer started the workmen on another tunnel, which will tap the lower seam about 1000 feet lower on the dip; and it is reasonable to suppose that when the coal is reached by this tunnel, a still more favorable analysis will be obtained. The coals are all clean and bright, and remarkably free from sulphur and other impurities. Time did not permit an extended examination of the sandstones and shales; but the indications are that the coals are of Cretaceous age.

In a conversation with the writer, at the time of the excursion to drive the golden spike on the completion of the Northern Pacific Railroad, Professor Geikie, of the Edinburgh University, admitted that his ideas and opinions on geological questions had certainly been expanded by his visit to the great West. A remarkable instance of the geological anomalies so frequently encountered by the mining engineer, in this western country, is furnished by this coal-field, in the entire absence of fire-clay with the upper seams, while *on top* of the lower seam is a layer of excellent fire-clay,

nearly a foot thick, and immediately above the fire-clay is a hard, compact sandstone.

The following table gives the analyses of the various coals mentioned in this paper, and also, for comparison, the official analysis of the Coos Bay coals:

Analyses of Oregon Coals.

Locality.	Ash.	Fixed Carbon.	Volatile Matter.	Sulphur.	Water.	Coke.	Analyst.
Coos Bay. Upper Bench	8.05	34.95	41.55	2.53	15.45	None.	U. S. Geo. Reports.
Coos Bay. Lower Bench	6.18	32.40	44.15	1.37	17.27	None.	U. S. Geo. Reports.
Yaquina Bay	7.10	32.60	46.20	1.07	13.03	None.	Private assay.
John Day River, Discovery Seam	7.20	51.00	37.20	.60	4.00	Good.	J. H. Fisk, Portland, Oregon.
John Day River, Lower Vein	7.26	48.19	40.00	.60	4.55	Good.	Geo. S. Miller, Oswego, Oregon.
John Day River, Lower Vein	5.76	50.67	36.80	.79	6.77	Good.	Made at Omaha, Neb. Analyst's signature illegible.
John Day River, Tunnel Lower Vein	5.95	52.41	34.45	.65	6.54	*Excel'nt.	Thomas O. Cairns, Portland, Oregon.
Coke from Lower Vein, John Day River	6.18	87.71			6.11	†	Thomas O. Cairns, Portland, Oregon.

SOME ONTARIO MAGNETITES.

BY T. D. LEDYARD, TORONTO, CANADA.

(New York Meeting, September, 1890.)

Sir William Logan, our great geologist, predicted that Canada would become eventually one of the greatest iron-producing countries of the world. Although possessed of numberless deposits of iron-ore, we have not yet done much to verify this prediction. Causes can be found for our backwardness, the principal one being

* Yield of coal in coke, 58.36 per cent.

† The coke was immersed in water on being turned out of the crucible; hence the excessive moisture.

the want of a market. A glance at the map shows that the small population of Canada is too scattered to support extensive iron manufactures. Six good-sized iron-mines would supply the ore necessary to make all the iron and steel used in Canada, even if we used nothing but our own make instead of importing the greater portion, as we do at present. And distances in Canada are too great in proportion to the small consumption to encourage home manufactures on a large scale.

But while Canada is suffering for want of a market, many parts of the United States would be benefited by the free admission of Canadian ores. The Eastern States could get cheap ores from Quebec and Nova Scotia, and many Pennsylvania furnaces could obtain Bessemer ores from Ontario at much lower rates than they are now paying for imported foreign ores.

Geology of the Ontario Iron-Districts.

Dr. Robert Bell, Assistant Director of the Geological Survey of Canada, in his "Geology of Ontario,"* says:

"The name Huronian (derived from Lake Huron) was first given by the Geological Survey of Canada more than forty years ago, and has been adopted by geologists in other countries as universally as the term Laurentian, and is made to include all the rocks lying between the Laurentian below it and the Cambrian or earliest fossiliferous rocks above. Some of the rocks of the district stretching between the counties of Lanark and Hastings in eastern Ontario, which were formerly included in the Laurentian series, are now believed by some to belong rather to the Huronian. An attempt has been made quite lately, among some American geologists, to restrict the name Huronian to rocks like some of those north of Lake Huron, although Sir William Logan and his colleagues, in introducing the term originally, described it as applying equally to the dark grayish and greenish schists, conglomerates, diorites, etc. The more extended investigations which have since been made in Canada and other parts of the world have confirmed the propriety and convenience of including under this name all the rocks which had been originally described as Huronian.

"The Huronian, as above defined, is the great metalliferous system of Ontario, as well as of northern Michigan and Minnesota, and hence its great importance in the economic geology of the country. The whole series is more or less metalliferous, but the various ores are not uniformly distributed, some occurring in one region or in some special stratum, while others may prevail in another section of country or in a different horizon in the series. Iron appears to occur most frequently in the lower or schistose portions of the Huronian system."

Some Magnetite-Deposits.

This paper will be confined to a short description of certain ores found in the counties of Peterboro and Haliburton, and in the more

* *Mineral Resources of Ontario*, pages 17 to 22.

easterly portions of the Province of Ontario. Those which will be mentioned are probably among the most important, yet comprise only a few of the ores which have come under the writer's notice. Magnetites only will be here taken into account, although hematites are found in these districts in deposits of considerable importance and sometimes of great purity.

Prof. E. J. Chapman, Ph.D., of the School of Science and University College, Toronto, says:

"This area of Ontario is traversed in a general northeast and southwest direction by belts of elevated rocky land, consisting of unstratified, probably eruptive, syenites or syenitic granites of a prevailing red color. Between these belts lie rugged tracts of essentially synclinal structure, occupied in ascending order by strata of red and gray gneiss and by some crystalline graphitic limestones, succeeded more or less irregularly by dark green amphibolic and pyroxenic rocks, with which the iron-ores are chiefly associated."

The Belmont Iron-Mine.

About 110 miles east of Toronto, 6 miles north of the branch of the Canadian Pacific Railway, between Toronto and Montreal, on Lot 19 in 1st Concession of the township of Belmont, county of Peterboro, is found a large deposit of rich magnetic ore, closely resembling in mode of occurrence some of the Lake Superior mines. Experts estimate that this ore-bed contains at least 1,000,000 tons of ore within 100 feet of the surface, from which there should be no difficulty in producing 400 or 500 tons a day.* The ore is very high in metallic iron, and by no means refractory in the blast-furnace. Samples taken from the pits possess a fine texture, and show but small traces of phosphorus and sulphur, with a high percentage of iron, constituting an almost ideal Bessemer ore.

The stripping is so light that the ore should be mined, by the use of compressed-air drills, at a cost of 50 cents per ton, being so free from rock-matter that little culling will be required.† No trouble with water is to be anticipated, the ground possessing a natural drainage to the west, which can be improved and utilized at slight expense.

A special feature of this deposit is its proximity to the surface, necessitating but little expense for stripping, as mining can be done in pay rock almost from the commencement.‡ Some hematite is found on the east side in conjunction with soapstone.

* Prof. Heys, Report, 1885, and Prof. C. Gordon Richardson, Report, 1890.

† Report of F. D. Taylor, M.E., 1886.

‡ Prof. C. Gordon Richardson, Report, 1890.

Prof. Chapman says: "This is an exceedingly good ore, not too dense in texture, rich in metal, quite free from titanium and practically free from phosphorus and sulphur, and is well adapted for the Bessemer process."

The following analyses have been made, the first being from surface samples.

No.	Metallic Iron.	Phosphorus.	Sulphur.	Silica.	Manganese.
1	64.26	Faint trace.	0.04	.....	.....
2	65.36	0.002	Trace.	4.5	.....
3	66.29	0.024	Trace.	3.19	0.42
4	68.88	0.006	Trace.	3.18	.....
5	68.85	0.008	Trace.	1.96	.....
6	69.99	0.012	Trace.	3.10	.....

No. 1 was made May, 1884, by Prof. E. J. Chapman; No. 2. July, 1886, by W. F. Brugman, chemist of the Scranton Steel Co.; No. 3, November, 1887, by the chemist of the Joliet Steel Co.; No. 4, same date, by Prof. Davenport Fisher, Milwaukee; Nos. 5 and 6, August, 1888, by the chemist of the Cambria Iron Co.

Much of the rock-matter surrounding this ore-deposit is composed of epidotic diorite.

The Blairton Iron-Mines.

Four miles south of the Belmont mine are the Blairton iron-mines, at the village of that name, on Lot 8, in 1st Concession of Belmont, about 2 miles from the Canadian Pacific railway.

At this place, originally known as "the big ore-bed," there are large deposits of magnetic iron-ore, averaging probably 50 to 55 per cent. metallic iron, and low in phosphorus, with no titanium. Analyses have shown 0.035 phosphorus.

A considerable amount of ore has been taken out of these mines, as much as 300 or 400 tons daily having been produced here at one time. A large and deep excavation has been made, 150 feet wide by 250 feet long, and a second opening 160 feet deep lies south of the first.*

On the east side the deposit is capped with Silurian limestone, while the west wall is a jaspery quartzite. The strike is west of

* Report of the Royal Commission on the "Mineral Resources of Ontario," 1890, p. 127.

north. In the wall, quartz and serpentine predominate. This ore is much mixed with rock, necessitating a good deal of culling, which renders mining expensive; and there has been sulphur in some parts of the deposit from the beginning.

This is one of the many iron-mines in Canada which cannot be worked at a profit for export under the present United States duty of 75 cents per ton. If there were no duty this mine would doubtless be worked; it is capable of producing largely. At present the only use that can be made of such ore is to smelt it in a local charcoal furnace, there being plenty of hard-wood in the neighborhood from which charcoal can be cheaply made, and limestone for flux not being far distant.

Central Ontario Railway Mines.

There are several large deposits of magnetic ore on or close by the Central Ontario railway in the north part of the county of Hastings. Some of these ores are of Bessemer quality (after roasting out the sulphur), but unfortunately the mines are now lying idle, owing to serious dissensions among the owners. The statement has been made that these ores are worthless, and that the mines are not worked on account of the sulphur in the ore; but this is surely not the true reason. Nearly 800,000 tons of magnetic iron-ore are annually taken out of the Cornwall hills in Pennsylvania and used as Bessemer ore, although containing an average of 2 per cent. sulphur and only about 48 per cent metallic iron; and nearly 700,000 tons of other sulphurous ores, some of them being hard magnetites from the States of New York and New Jersey very similar to Central Ontario ores, are annually used in the United States. The sulphur entails an additional cost in roasting; but where other injurious elements are absent, a sulphurous ore, after being properly roasted, often can be used in the manufacture of Bessemer iron.

The roasting should be carefully done in Davis-Colby or other suitable roasters, where plenty of oxygen is supplied during the process. The ore should be broken up fine enough and plenty of time given to make the roasting effectual.* This should be done where fuel is cheap, at a cost not exceeding 25 cents per ton. The cheapness of mining and freight would much more than counterbalance this expense on some Ontario ores.

Many of the Central Ontario ores could be conveniently shipped

* See Dr. Valentine's paper of October, 1889, *Trans.*, xviii., 303.

to furnaces in Pennsylvania by returning American coal-cars, which bring coal into Canada, but now usually return empty.

Snowdon Iron District.

The townships of Snowdon, Galway and Glamorgan in the county of Haliburton, about 40 miles northeast of Lindsay, contain several valuable deposits of iron-ore.

Lots 25, 26 and 27 in the 4th Concession, Snowdon, on the south side of Burnt river, contain several outcrops of magnetite extending about three-quarters of a mile. This ore is on high ground overlooking the river and railway-track, and most conveniently situated for mining by drifts or tunnels run into the hillside, the mine-cars running by gravity to the shipping platforms or pockets at the railroad.

The following analyses show these ores to be very low in phosphorus:

No.	Metallic Iron.	Phosphorus.	Sulphur.	Silica.	Titanium.
1	61.48	0.01	0.16	.....	None.
2	62.	Trace.	0.025	1.7	"
3	62.57	0.025	Trace.	.....	"
4	63.	Trace.	0.025	3.1	"

No. 1 was made by Prof. E. J. Chapman; Nos. 2, 3 and 4 by chemists of different iron-works in Pittsburgh, whose names are not in my possession.

Concerning analysis No. 1, Dr. Chapman says: "A fine-grained, magnetic ore from the Howland mine, Lot 26 in 4th Concession, Snowdon. The ore near the surface is very pyritous, but becomes almost free from pyrites in descending. The sample was taken from a depth of 81 feet from the surface. A second shaft has been sunk on another part of the deposit to a depth of about 30 feet. The deposit is apparently an elongated stock-formed mass of large extent. It has been opened over a length of about 80 feet."

The railway passes through Lot 20 in 1st Concession, Snowdon, on which are several outcrops of magnetite, one of them crossing the Monck road, where it was first discovered. Prof. Chapman has analyzed a surface-sample and found it to contain metallic iron, 60.18; phosphorus, 0.08; sulphur, 0.04; titanic acid, 0.73. He describes it as a black magnetic, fine grained ore, somewhat porous in texture. The deposit is apparently a stock-formed mass of large

size, as shown by recent borings and excavations. Another analysis is given, showing metallic iron, 61.02; phosphorus, 0.052; and still others have shown metallic iron as high as 65 per cent.

This district is about 200 miles from Buffalo, N. Y., to which point the ores could be delivered very cheaply in returning coal-cars. Moreover, these deposits are not much more than 100 miles from Midland, Ontario, whence ore could be shipped by boat to Chicago in returning grain-vessels. Limestone suitable for flux is found in close proximity to the ores, and there is abundance of hard-wood suitable for charcoal. There is also brown hematite in the neighborhood, yielding 45 to 48 per cent. metallic iron which would mix well with the magnetic ores in a charcoal-furnace.

The Paxton mine, in the adjoining township of Lutterworth, has produced several hundred tons of ore yielding 55 to 60 per cent. iron with no titanium and traces only of sulphur and phosphorus.

Ores from different parts of Galway Township, south of Snowdon, have given the following analyses:

No.	Metallic Iron.	Mang. oxide.	Phosphorus.	Sulphur.	Titanium.
1	62.37	Trace.	0.012	Slight trace.	None.
2	62.60	1.27	0.008	Slight trace.	"
3	65.	.....	0.04	0.02	"
4	70.40	.....	Trace.	Trace.	"

Nos. 1 and 2 were made by Prof. Chapman; Nos. 3 and 4 by Prof. Thos. Heys, formerly lecturer on chemistry in the Toronto School of Medicine.

Various Ore-Deposits.

Professor Chapman has also made the following analyses of magnetites from this district which are free from titanium, viz.: No. 1,* from township of Glamorgan: a black crystalline, highly cleavable ore, carrying 70.38 metallic iron, with merely traces of phosphorus and sulphur. There are a few ore-exposures and needle-attractions showing a strong body of ore extending over a length of about 400 feet by 40 feet in width. No. 2 is an ore corresponding almost exactly in its cleavable structure and composition with No. 1, although it comes from another place several miles distant. Samples taken from a small trial-pit show 70.5 per cent metallic

* These five analyses are numbered in Professor Chapman's list 9 to 13 inclusive.

iron with rock-matter under 3 per cent., but the ground is practically undeveloped.

No. 3 is a black, fine granular ore from the vicinity of Apsley village, in the northern part of the township of Burleigh. Metallic iron, 63.68 per cent.; phosphorus, a trace only; and sulphur, 0.03. The intermixed rock-matter is essentially pyroxenic. Ground undeveloped.

No. 4 is a very rich magnetic ore from the township of Madoc, the dried ore yielding 69.16 metallic iron; phosphorus, 0.04; and sulphur, 0.03; with siliceous rock-matter, 3.32.

No. 5 is a magnetic ore mixed with specks of actinolite from Madoc township. The sample contained nearly 1 per cent. pyrites and yielded iron, 60.32; phosphorus a trace; and sulphur, 0.52. Another sample from the same property gave iron, 62.54; phosphorus, 0.03; and sulphur, 0.38.

Kingston and Pembroke District.

The following analyses (page 36) have been made of magnetic ores from the country tributary to the Kingston and Pembroke railway.

Titaniferous Ores.

In Minden, the township lying north of Snowdon, there are large quantities of magnetic ores containing a considerable amount of titanium. A black, granular, strongly magnetic ore from near Lake Kushog showed by Dr. Chapman's analysis* the following results: Metallic iron, 51.56; titanium, 15.64; sulphur, 0.17; phosphoric acid, trace; siliceous rock-matter, 4.08.

Similar ore is found on several adjoining lots, and there appears to be a good deal of titaniferous ore in this and neighboring townships.

There is an enormous deposit of black, strongly magnetic ore in the township of Glamorgan, about half a mile south of Burnt river. The deposit rises abruptly in the form of a succession of ledges to a height of from 80 to 100 feet above the general level of the ground, and is exposed in an easterly and westerly direction for at least 1800 feet, with an average width of 140 feet. The analyzed sample contained 52.04 metallic iron; 8.11 titanium; 0.005 phosphorus; and 0.06 sulphur.

Another deposit of similar ore has been found some distance from the one just described. This ore carries from 8 to 10 per cent. titanic

* All the analyses of titaniferous ores here given were made by Prof. Chapman.

acid, which appears to be persistent throughout the whole ore-bed, the yield of metallic iron being nearly 60 per cent.; phosphorus 0.007; and sulphur 0.08.

It may be remarked that titaniferous ores in these districts are generally very low in phosphorus, and if some treatment could be found whereby the injurious effects of the titanium could be got rid of they might be used in making steel.

Dr. Chapman remarks that "the presence of these titaniferous ores in close proximity to deposits in which no trace of titanium can be detected, is a fact of much interest hitherto apparently unrecognized in the iron-districts of Europe."

No.	Metallic Iron.	Phosphorus	Sulphur.	Silica.	Titanium.
1	65.33	0.017	.....	.....	None.
2	66.	Trace.	Trace.	2.14	"
3	62.	0.006	Trace.	4.12	"
4	60.	0.009	.....	6.31	"
5	52.	0.055	0.10	6.53	"
6	65.07	0.07	0.05	6.66	"
7	66.34	0.14	Trace.	1.04	Trace.

No. 1 was made, June, 1883, by Ledoux and Ricketts; Nos. 2, 3, 4 and 5 by a chemist employed by the Kingston and Pembroke Iron and Mining Co.; No. 6, November, 1871, by J. Blodgett Britton; No. 7 by a chemist unknown to me. This analysis, with Nos. 2, 3, 4 and 5, was published by the Minister of Agriculture in connection with the mineral exhibit of Ontario at the Cincinnati Exhibition of 1888.

The writer feels warranted in protesting against the habit, which has prevailed in some quarters, of characterizing all Canadian ores as titaniferous. Nothing could be more unfair. Canada is a country of great extent, and it is just as absurd to class all Canadian ores as titaniferous because those objectionable ores are found in some places, as it would be to condemn Lake Superior ores because titanium is found in some parts of northern Minnesota. Ontario contains, as I have shown, many ores perfectly free from titanium, although there are certain ranges or belts of territory in which titanium prevails to a considerable degree.

Many of the magnetite deposits in eastern Ontario are rich in iron, and carry, too, very low percentages of phosphorus and sulphur with no titanium at all. Others are high in sulphur but still

free from other impurities, so that they require roasting only, in order to make them available as exceedingly pure 60 per cent. ores. Finally, the rates of wages in these districts generally range from $1.00 per day for laborers to $1.25 for miners; and, as has been observed, the commercial conditions of transportation are exceedingly favorable. Nothing seems to be lacking but access to the market of the United States.

THE WALKER ANTI-VIBRATION REGULATING SHUTTER OR SLIDE FOR GUIBAL AND OTHER ENCLOSED FANS.

BY EDWIN R. WALKER, MEM. INST. MIN. AND MECH. ENGR'S OF SOUTH WALES, PAGEFIELD IRON WORKS, WIGAN, ENGLAND.

(New York Meeting, September, 1890.)

THE old adage, "Necessity is the mother of invention," is fully borne out in the case of the device described in this paper. About four years ago the firm of the writer received instructions from Sir John Hawkshaw, C. E., and I. Wolf Barry, Esq., C. E., to construct and erect three large Guibal fans for ventilating a portion of the Metropolitan and District Underground Railway, of London. On account of the cost of coal and the very limited space for boilers, it was arranged that the fans should be driven by gas-engines.

Those who have had experience with Guibal fans will be aware that as each blade or vane passes the lower edge of the shutter a pulsatory action takes place. When the fans above mentioned were set to work, the pulsation or vibration of the immense volumes of air discharged was so great that it caused a corresponding vibration of the window-sashes, doors, etc., of the surrounding buildings; and this was so unpleasant that the shopkeepers and others near obtained an injunction against the railroad company, compelling it to stop one of the fans. The work was by this time out of our hands, and we only casually heard what had taken place, learning also that, in addition to this injunction, another was threatened in the case of the second fan, and that the railroad company had thus far escaped the enforcement of this threat by agreeing to run the second fan at a few short stated intervals only during each day.

In considering the case, it occurred to us to incline the lower edge of the shutter, instead of making it parallel with the axis of the fan, by which means we thought that the waves of air might be broken

up, or at least reduced. It was, however, ultimately decided to make the shutter like an inverted M—thus: W.

Having a small working-model of a Guibal fan 46 feet in diameter —made to a scale of ½ inch to 1 foot, and running at about 500 revolutions per minute, we attached the shutter to it, and were gratified to find that, whereas with an ordinary Guibal shutter the vibration could be heard about a hundred yards away, with the new shutter vibration had ceased, and only the sound of the rushing air could be heard even by a person standing quite close to the fan.

By way of parenthesis I may add that subsequently we made further experiments with the fan, and had boxes and pipes, representing the roads of a mine, attached to it. From previous experience we had found that in the Guibal fans the air was thrown violently against the outer walls of the *évasée* or chimney, but that at the inner wall there was a distinct downward suction. To test the action of the new shutter as to this, we held slates smeared with a colorless mucilage, a few inches above the chimney, and inserted white flour into the aforesaid boxes. The flour was drawn through the fan, discharged against the slate, and was found to be evenly distributed over the mucilage—the downward suction had disappeared. But when the Guibal shutter was attached, the main body of the flour was found sticking to that part of the slate held directly over the outer edge of the chimney.

No time was lost in attaching suitable shutters to the London fans, and the results were in every way successful; the fans being then able to run regularly 15 to 18 hours per day, to the entire satisfaction of both the railway company and the surrounding neighborhood.

It was only necessary to make known this invention to the many colliery companies using Guibal fans, in order to receive orders for the shutters; and in about two years the majority of the Guibal fans in Great Britain were provided with them. At the present time there are in use upwards of 120 Guibal and Walker fans, varying in size from 15 to 50 feet in diameter, in addition to a considerable number of smaller fans, such as are used for smelting-fires, etc., to which this shutter is attached.

The whole plan of ventilation, as well as the construction and erection of the four fans (two of 40 and two of 30 feet in diameter, with their engines) for the railway tunnel under the River Mersey, connecting the important cities of Liverpool and Birkenhead, was carried out by our firm. These fans are likewise provided with the Walker shutter.

Mr. Francis Fox, C. E., who was largely concerned with the designing and execution of the works, says in a paper read before the Institute of Civil Engineers:

"The fans were somewhat similar to the well-known Guibal fans, excepting that in the shutters, to which Mr. Guibal attached the chief value of his patent, an important alteration has been made."

The effect of the alteration he thus describes:

"The action of this patent regulating shutter has an important bearing upon the ventilating-fans in their consequently increased durability and efficiency. In towns like Liverpool and Birkenhead any pulsatory action would be readily felt by the inhabitants. It is difficult to detect any sound whatever when standing close to the buildings containing the fans."

The fan (40 feet in diameter) and engines for ventilating the tunnel of the Great Western Railway Company, under the River Severn, which is probably the longest sub-aqueous tunnel in the world—its length being about 4¼ miles—were erected by our firm under instructions from Sir John Hawkshaw, C. E., who has expressed his approval therewith. This fan is also provided with our regulating shutter, and is exhausting about 500,000 cubic feet of air per minute.

Having thus explained the history of the invention and its utility in removing vibrations and consequent noise (for without it, pulsation of the blades can be heard at a considerable distance from the fans), I now come to a still more important advantage to be gained by its application. As most persons who have had Guibal fans in operation for a considerable number of years will be aware, fan-shafts, even though made unusually strong in proportion to the horse-power transmitted through them, frequently break. About four years ago the writer learned in Belgium, the birthplace of the Guibal fan, that the average life of the shafts was from 12 to 15 years. The breakage of fan-shafts and the loosening of bolts and rivets in fans generally are to be attributed to the excessive vibration hitherto inseparable from their operation. This vibration is caused by the too abrupt cessation of the delivery of the air from the fan-blades or vanes, as they pass the opening to the chimney; and for this the shape of the regulating shutter or slide is responsible. The upper part of the opening formed by the lower edge of the shutter or casing, as hitherto constructed, is parallel to the tips or edges of vanes, and as there is but little clearance or space between the shutter and the blades when their edges approach and concur, the delivery of the air is abruptly

terminated. While discharging, the air presses against the front or face of the blade, but the instant the edge of the blades passes the edge of the shutter, not only is the weight of the air suddenly removed, but there is also a partial vacuum formed above this point, so that the blade bounds or jerks forward.

Some time ago the writer took readings from a water-gauge which he attached to a wrought-iron fan-casing at intervals from the center to the periphery, above the shaft. At the center the water-gauge indicated 3 inches; but near the outer edge or periphery it was $\frac{1}{2}$ inch. The fan was 24 feet in diameter, and ran at 80 revolutions per minute. It had eight blades, each about 8 feet square, or 64 square feet in area. Taking the average water-gauge over the surface of blades as $1\frac{1}{2}$ inches, it would represent a pressure of 7.8 pounds per square foot, or a total pressure on each blade of 499.2, say, 500 pounds. Assuming the center of the blade to be the center of the load, the distance from center of load to center of crank-shaft would be, approximately, 9 feet. One can readily understand that a dangerous strain is put upon the whole structure by the removal of such a weight alone, every time one of the blades passes the shutter. But to emphasize this more fully, let us take the work of one day of 24 hours, the fan running throughout at 80 revolutions per minute. Eight blades x 80 revolutions x 60 minutes x 24 hours = 921,600. This product, therefore, represents the number of times in a single day that a weight of at least 500 pounds, is, as it were, instantaneously removed from the blades, and the shock or strain resulting from the removal, is communicated to the shaft. There is, however, not only what we may term the primary shock, but there must necessarily be, at least, one recoil of the arms and blades. This shock or vibration is transmitted through the whole of the fan, and as each vane represents such a powerful lever, the momentum of the blow or jerk becomes serious on reaching the shaft, a dangerous tremor being set up, which, sooner or later, destroys the fiber of the metal. The consequent injury to the general structure of the fan also is obvious.

The Walker regulating-shutter removes this evil by effecting a perfectly gradual change in the pressure referred to, and so governing the discharge of air as to cause it to pass, without objectionable eddying, in a continuous, instead of intermittent, stream from the fan-blades into the chimney, without the pulsatory action described. The working of the fan is thus rendered noiseless, and there is an entire freedom from vibration. Noise and vibration are the evidence

of mischief. In machinery, they may be regarded as synonymous terms.

So effective is this shutter, that even with the largest-sized fans, say, those of 46 or 50 feet in diameter, hardly the faintest beat can be heard, when running at from 50 to 60 revolutions per minute.

The invention of the shutter has enabled us to design fans of smaller dimensions than would be required for Guibal fans, to perform the same amount of work, and we now construct fans, 20 or 24 feet in diameter, driven on second motion, to meet requirements for which formerly we should have made Guibal fans, 40 or 46 feet in diameter.

It is generally considered that the best test of the efficiency of a fan is the water-gauge reading which can be obtained by a given periphery velocity. In this respect, we have obtained with large fans results equal to those given by the Guibal fans, and far exceeding any data we have been able to ascertain or have seen published concerning any other known fans.

As previously explained, the cutting edge of the shutter is changed from the horizontal to an elongated, inverted V or M. The invention, however, as described in our patent, does not confine itself to this shape, but includes other forms which permit the gradual discharge of the air from the fan-vanes into the chimney. As the shutters are of wrought-iron or steel plates, made very strong and stiffened where necessary with angle- and tee-irons, they are practically indestructible. They can also be adjusted in position, to suit the varying circumstances of each ventilator. The shutters can be very easily adapted to all closed fans, and can be erected *in situ* by the ordinary colliery engine-wright.

In every case where the shutter has been applied, it has given very satisfactory results, not so much perhaps in increased ventilating-power for the same velocity, as in the greater smoothness of working and in reduction of repairs to the fan and casing. We do, however, obtain the very important advantage, that the fans can be run safely at much higher velocities, and consequently with a proportionate increase of exhausting-capacity.

We have concluded arrangements for the employment of our patent shutter with the Lehigh Valley Coal Company, Wilkesbarre, Pa., Eckley B. Coxe, Esq., of Drifton, Pa., and the Philadelphia and Reading Coal and Iron Company, Pottsville, Pa.

In conclusion, I may add that, at the International and Colonial Mining, Engineering and Industrial Exhibition at Newcastle-on-Tyne, England, in 1887—probably the most important of its kind

ever held—out of several hundred exhibits in the Mining and Colliery Machinery section, the only gold medal awarded was given to us for this invention. In view of the circumstance that the majority of the members of the Council were more or less connected with mining interests, greater weight is added to their decision.

A SUSPENDED FEED-TABLE FOR ROLLING-MILLS.

BY JAMES MORGAN, PITTSBURGH, PA.

(New York Meeting, September, 1890.)

THE convenience of mechanical arrangements for handling ingots, blooms, billets, bars, beams, etc., and feeding them to the rolls, is so universally recognized as to require no demonstration.

In the rolling of heavy beams, such as are now coming into general use, such arrangements are practically essential, by reason of the difficulty and expense of the manual handling of the large masses of metal treated. I submit herewith the drawings of a rolling-mill feed-table, designed and patented by myself, which I think will be found simple in construction and operation, not likely to get out of order, and efficient in performing the work for which it is intended.

Fig. 1 is a sectional plan, taken on the line I I of Figs. 2 and 3. Fig. 2 is a vertical longitudinal section on the line II II of Figs. 1 and 3. Fig. 3 is a vertical cross-section on line III III of Figs. 1 and 2.

As the details of the drawings are numbered throughout, and the same number always indicates the same part, I cannot more clearly describe the mechanism than by giving the following key to these numbers:

Nos. 2 and 3 are the sets of rolls (here shown as three-high rolls, though that is not absolutely necessary).

Nos. 4, 4 are the two feed-tables, one on each side of the rolls. The side frames of the tables are made of parallel channel-bars.

Nos. 5, 5 are the feed-rollers, journaled between the channel-bars and provided at their ends with beveled gear-wheels, 6, 6, which mesh with beveled pinions, 7, 7, on shafts, 8, 8, which latter are driven by elevated power-connections, to be mentioned presently.

The tracks, 9, 9, on which the supporting-carriages are mounted,

are upheld by elevated supports in the building, as shown in Figs. 2 and 3, and extend transversely to the line of the feed-tables. They consist of parallel flanged beams, separated so as to permit the passage between them of the suspending mechanism of the feed-tables, but connected firmly at intervals by strong yokes, 10, 10, the ends of which underlie and support the tracks, while the middle portions are arched high enough to permit the passage of the trolleys or carriages. These yokes are connected by links, 11, 11, with the bell-crank levers, 12, 12, which serve to lift them.

Fig. 3 shows the yokes suspended from the bell-crank levers, the latter being fixed to cross-shafts 17, 17, resting with their journals in pillow-blocks 13, 13, on the beams, 14, 14, which extend above the track and parallel therewith, and are suitably supported by the uprights of the building-frame.

All the supporting-levers on each of the beams, 14, are connected by means of connecting-rods, 15, and each is provided with a counterweight, 16, which opposes the weight of the supported feed-tables and partially balances the same. One or more of the cross-shafts, 17, extends across all of the beams, 14, and forms a common axis or torsion-shaft for the laterally-adjacent bell-crank levers, 12, so as to connect operatively the whole system of levers and to cause them all to respond to the motion of any one. One of the end levers of the set is connected with the plunger of a hydraulic motor, 18, by a flexible connection, 19, a quadrant-lever, 48, and a connecting-rod, 49. The motor thus serves to actuate all the levers, as will be readily understood, so that, by the movement of the plunger up or down, all the tracks may be simultaneously and uniformly elevated or lowered.

Each of the tracks is provided with a trolley or carriage, 20, mounted on wheels, 21, on the track directly above the feed-tables, and each carriage is connected to the feed-table by means of a frame, 23, composed of strong upright bars attached to the carriage at the upper end and provided at the lower end with a stirrup or hanger, 22, on which the side rails of the feed-table are fixed. The stirrups, 22, are cast-iron frames open at the side to permit removal of the metal from the feed-table. On one of the frames, 23, of each of the feed-tables is journaled an upright rotary-shaft, 24, provided at its lower end with a bevel-pinion gearing with a corresponding bevel gear-wheel on the shaft, 8, and at its upper end, at the level of the carriage, provided with a gear-wheel meshing with a pinion on a sprocket-wheel shaft, 25, which is journaled in the frame of the carriage. Obviously this sprocket-wheel shaft is constantly in gear

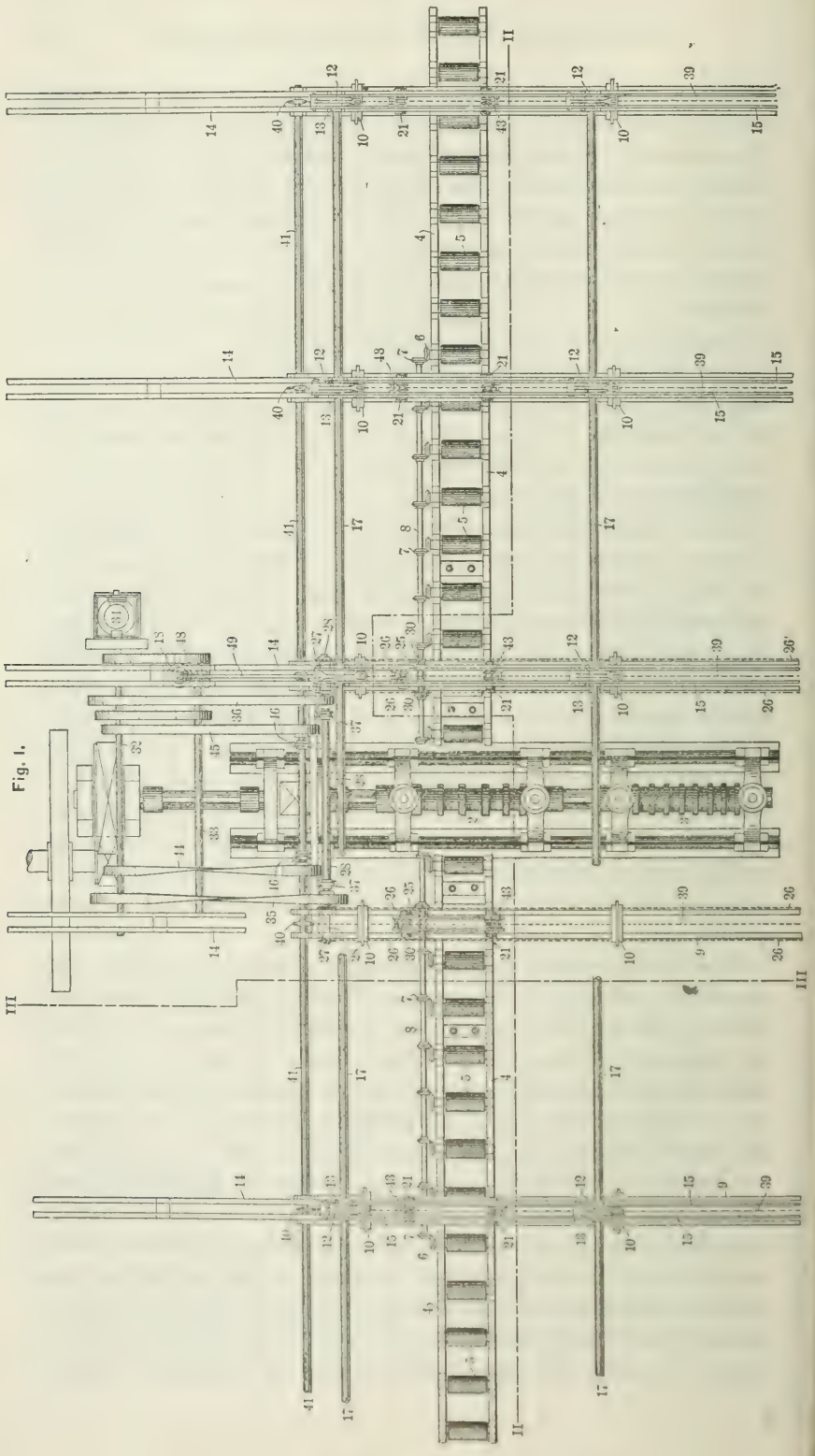


Fig. 1.

with the driving-shaft of the feed-rollers, whatever be the position of the feed-table and carriage.

The sprocket-wheel shaft is driven by means of endless sprocket-chains, 26, which pass around sprocket-wheels, 27, on a driven counter-shaft, 28, around sprocket-wheels, 29, at the other end of the track, and in contact with sprocket-wheels, 30, on the shaft, 25, and with idler wheels, 31, on the carriage. This arrangement is such that the carriages may be moved along the track without disengaging the sprocket-wheels, 30, from the chains, and that in every position of the carriage the motion of the chains will rotate the shaft, 24, and through it the rollers of the feed-table. The counter-shaft, 28, is adapted to be driven in either direction by means of power-connections between it and a shaft, 32, which is connected, through suitable power-connections and an intermediate shaft, 33, with the driving-shaft of an engine, 34.

The power-connections between the shafts, 28 and 32, consist of two belts, 35 and 36, one of which is crossed, and the belt-pulleys on the shaft, 28, are provided with clutches, 37, operated by a lever, 38, by which either pulley may be put into gear with the shaft. These pulleys effect the driving of the shaft in opposite directions, and thus enable the feed-rollers of each table to which the shaft is indirectly connected to be driven in either direction.

Each of the carriages is moved backward and forward on its track by means of chains, 39, which pass around sprocket-wheels, 40, on a shaft, 41, and around sprocket-wheels, 42. The ends of these chains are fastened to the carriages, as is shown in Fig. 3.

The idler-wheels, 43, on the carriages support and guide the moving parts of the chains. The shaft, 41, is driven by belts, 44 and 45 (one of which is crossed), from the shaft, 32, and the belt-pulleys on the shaft, 41, are provided with clutches, 46, which are operated by means of a clutch-lever, 47, in the manner before described with reference to the clutches, 37, and lever, 38, so that by means of these clutches the shaft, 41, may be driven in either direction and the carriages caused thereby to move backward or forward on their tracks, as may be desired.

The operation of the feed-tables is as follows: In order to move the tables laterally from one pass of the rolls to another, or from one set of rolls to another, the carriages, 20, are caused to move on their tracks in the manner already described. By this motion the feed-rollers will not be disconnected from gear with the driving mechanism on the carriages, which may be caused to operate in either direction, as I have explained. If the feed-tables be used in

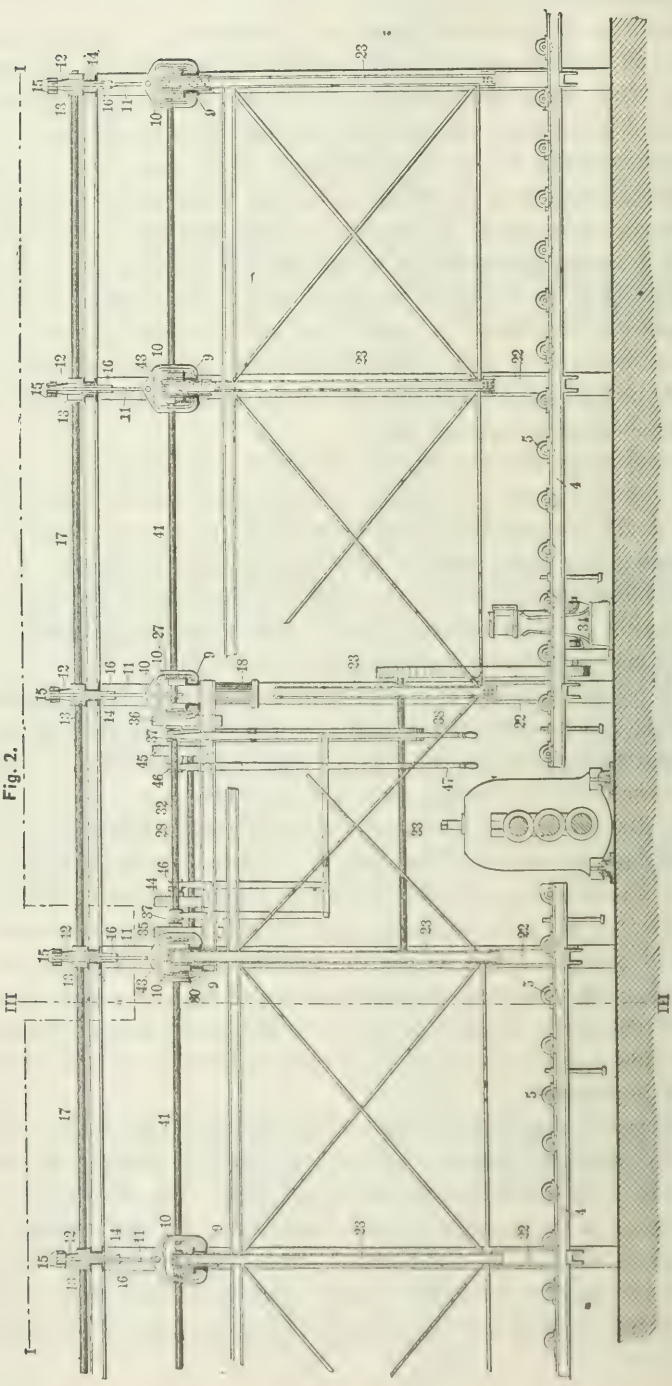


Fig. 2.

Suspended Feed-Table.—Vertical Longitudinal Section on Line II–II, of Figs. 1 and 3.

connection with three-high rolls, they may be raised or lowered to shift the metal from one pass of the set of rolls to the next, either above or below, as the case may be, by action of the hydraulic motor, 18, by means of which the tracks, 9, and the supporting-carriages may be raised or lowered. The driving of the feed-rollers and the lateral motion of the feed-table are effected by means of the levers, 38 and 47, and the hydraulic cylinder may be actuated or controlled by means of a valve situated in proximity to the levers, so that the whole system may be under the control of a single person.

The means for turning the metal piece over on its side on the feed-tables, as shown in Figs. 1 and 3, comprise a suitable number of rods, 50 and 51, set vertically and loosely in cross-pieces, 52, of the feed-table, so that they shall be movable freely. When the feed-table is at its normal position in front of the passes of the rolls, these rods hang down below the tables, the tops of the rods being below the level of the feed-rollers, as shown by full lines in Figs. 2 and 3. If it be desired to turn the metal piece that is on the table over the rods, 50, the table is lowered until the lower ends of the rods, 50 and 51, strike the floor of the mill. This causes a relative upward motion of the rods, as shown by dotted lines in Fig. 3, and the rods, 50, engaging the metal, raise it and tilt it over on its side against the now elevated rods, 51, which serve as stops. By providing the ends of the rods, 50, with steps, so as to enable them to engage at more than one place with the metal, the metal may be turned several times on the table.

I need add but little to the foregoing description. As will be seen at once by those who understand the requirements of rolling-mill practice, the leading peculiarity of this system is, that the feed-table is suspended from above, instead of being supported from below. The advantages of such a suspension are easily appreciated; and attempts, more or less elaborate, have been made to secure them. But the fact that the general practice, especially for heavy work, is to support the feed-tables from below, indicates that these advantages have thus far been outweighed by the difficulties attending the overhead construction, particularly when lateral as well as vertical motion is to be effected and controlled, and the turning of heavy masses is to be combined with their elevation and propulsion. I venture to think that this somewhat complicated problem has been successfully solved in the accompanying design. But that is an opinion which would carry greater weight if it should be confirmed by the verdict of the competent rolling-mill engineers of the Institute, to whose judgment it is respectfully submitted.

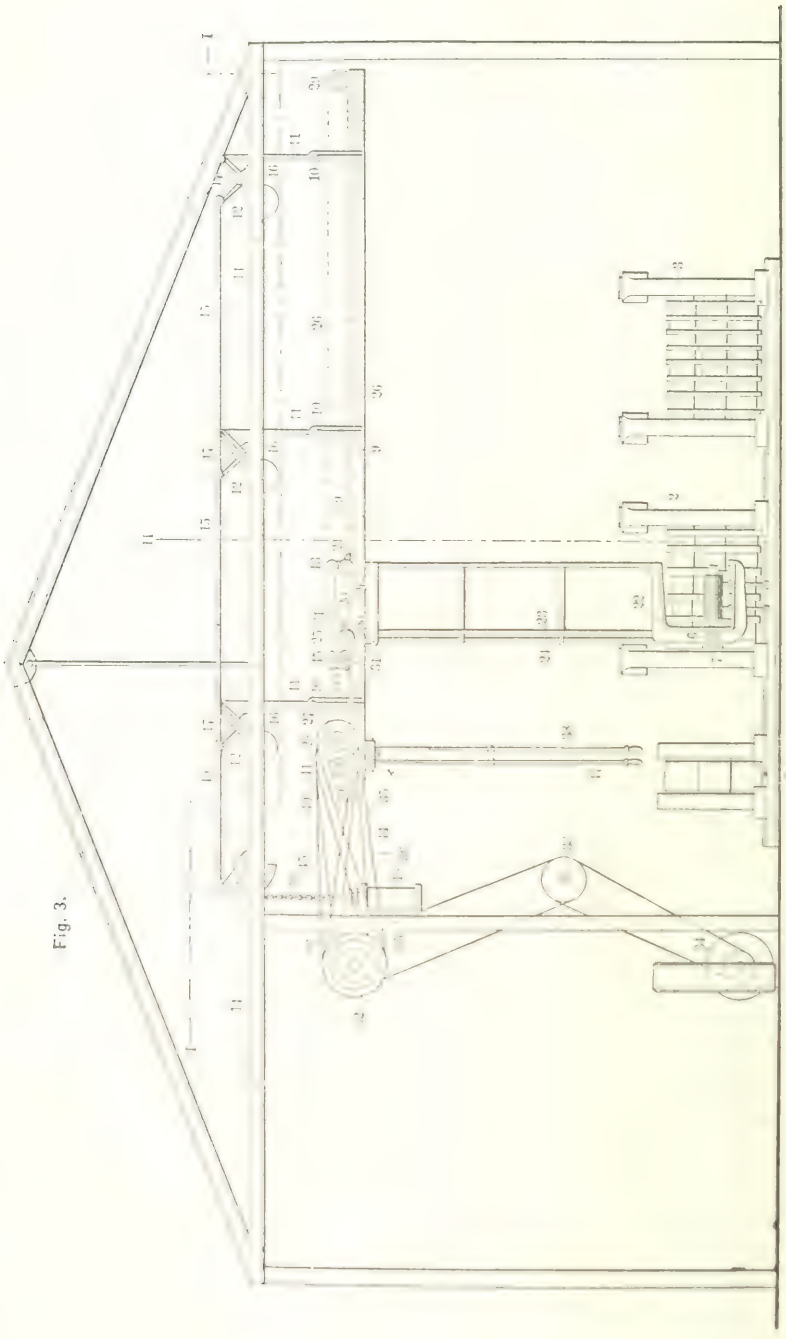


Fig. 3.

Suspended Feed Table.—Vertical Cross-Section on Line III–III, of Figs. 1 and 2.

THE RESOURCES OF THE BLACK HILLS AND BIG HORN COUNTRY, WYOMING.

BY H. M. CHANCE, PHILADELPHIA, PA.

(New York Meeting, September, 1890.)

By courtesy of the officials of the Chicago, Burlington and Quincy railroad, I am permitted to publish the results of an examination made in 1887 and 1888 of the country west of the Black Hills, and lying between the Black Hills and the foot-hills of the Big Horn Mountains in the northeastern part of Wyoming.

With the exception of the country constituting the foot-hills, or the encircling ridge or rim of the Black Hills proper, all of this area is occupied by rocks of Cretaceous age.

In the central part of the Black Hills are found the tin-bearing granites, the mica-schists, and gold-bearing slates, already so well known. Surrounding this central area of older rocks (which should be considered as the "Hills" proper) are the attenuated representatives of the Palæozoic, dipping gently away from the Hills in every direction, overlaid by the gypsum-bearing Mesozoic, which here forms a bold high ridge or rim, completely encircling the Black Hills, and known as the inner ridge. Overlying, and apparently conformable to this, we find the Dakota sandstone, which likewise forms a prominent ridge or rim, known as the first or outer ridge.

This is the formation which contains the coal now worked at the mines near Newcastle. It is also the formation in which the best oil-wells have been obtained in the "Stockdale," or "Beaver," and the "Belle Fourche" oil-districts.

Overlying it are the Colorado shales, forming the great plains drained by Beaver creek and the south fork of the Cheyenne river, and extending west to the Belle Fourche, near Donkey creek, where the overlying Fox Hill formation (thin) and the Laramie group, the great lignite-formation, are first seen. The Laramie group occupies the surface from the Belle Fourche at Donkey creek west to the foot-hills of the Big Horn mountains at Buffalo, an air-line distance of some seventy or eighty miles.

It is impossible to travel through the Black Hills, especially through their southern portion, without being charmed by the beauty of the country. It is not a rugged, inaccessible mountain region, although, perhaps, by reason of its elevation of 5000 to 7000 feet above sea-level, it should be termed mountainous. The summits are rounded, and the traveller seems to be passing through a rolling or hilly country, partly wooded and partly prairie, of great natural beauty. Here and there a bare granite peak rises above the general level, reminding one of the more rugged features of Rocky mountain scenery.

The fertility of the valleys is surprising; but when we consider that the valley soils originate in the decomposition of granites and metamorphic slates, containing a high percentage of all the elements necessary to fertility, it becomes evident that their present productiveness may be expected to continue for a long period. Similarly, we find the soils of the valleys and low lands (as well as a large portion of the hill-country) immediately surrounding the Hills rich and fertile, large areas consisting of limestone land from which good crops can be expected for centuries.

In the Central portion of the Hills heavy dew falls during the summer season, and showers are sufficiently frequent to insure the growth of crops without irrigation; but in the country bordering the Hills profitable farming can be assured only by the aid of irrigation. Every stream that issues from the Hills will be used for this purpose.

Aside from its agricultural resources and its scenic beauty, the central portion of the Black Hills is remarkable for its varied mineral resources. The precious metals, copper, iron and tin, are found widely scattered, and we cannot doubt that the output of metals and ores from this region will grow steadily and surely.

The ore-deposits of the Central Hills have been well described by Prof. F. B. Carpenter, of Rapid City, South Dakota, in a paper* read before the Institute at the New York meeting, February, 1889. Accompanying that paper is an excellent geological map of that portion of the Black Hills lying east of the Wyoming line. The "free-milling" gold-ores of the Homestake class, the refractory gold-ores and the tin-deposits are elaborately described by Prof. Carpenter, with many valuable original suggestions as to the best methods of treatment. Prof. Carpenter states that cement and fire-clay exist in large quantities.

* See *Trans.*, xvii., 570; also papers by Mr. Devereux, *Trans.*, x., 465, and by Prof. W. P. Blake, *Trans.*, xiii., 691.

Lack of proper railway facilities, the high cost of all kinds of materials and supplies, the high price of labor, and the difficulty and cost of transporting mining machinery, all combined to check the mineral development of the Hills until their cost was reduced by the advent of a railroad.

The tin-deposits have frequently been described by articles appearing from time to time in various technical journals, and the public has been led to expect a large output of that metal. The mining, concentration and reduction of tin-ores cannot be carried on successfully by small operators; large capital, extensive and thoroughly well-planned mills, concentrating- and reduction-works, cheap supplies and reasonably low-priced labor are necessary. These will be rendered possible by the building of the railroads now penetrating the Hills, and I think we may soon expect a considerable output of tin from the southern hills, especially from the neighborhood of Custer City, which, by the way, is one of the most beautifully located places I have ever seen.

With the cheapening of the chlorination processes, the successful working of the pyritic smelting process, and the introduction of coke from the Newcastle mines, it becomes possible to operate profitably a great number of mines, scattered throughout the Hills from north to south, that yield refractory ore, which, until the present time, it has not been possible to treat with commercial success.

On the western borders of the Hills the Stockade or Beaver creek and Belle Fourche oil-fields, now being developed, will become an important factor in the prosperity of the region. Here will doubtless be developed a great petroleum industry. Lubricating-oil in promising quantity and of superior quality has already been obtained, and the prospect of finding the lighter illuminating-oils at greater depth is good.

The oil is found in the Dakota sandstone, which can be reached by wells of moderate depth throughout a belt bordering the Black Hills and surrounding their outlying spurs, stretching north from the Cheyenne river to the Montana line, a distance of some 90 miles.

A few miles west of this belt, and parallel to it, is the Fox Hill sandstone, which underlies the Laramie formation and which may also prove oil-bearing.

The existence of oil in quantity, under the geological conditions prevailing in this field, may be taken as a guaranty of the presence of natural gas. Just what peculiarities these oil and gas-fields will exhibit, is as yet conjectural. Whether they will equal or exceed

those of the Eastern States, cannot be foretold, but we have every reason to believe that some large and valuable productive areas will be developed. The geological structure and the character of the formation are favorable to the existence of oil and gas in large quantity, and intelligently planned development by properly located wells will probably result in the opening of some large producing areas.

The Newcastle coal, now mined on a large scale, will furnish coke of excellent quality. I was recently called upon to determine the coking qualities of this coal and the methods requisite to the production of good coke, if that were possible. For this purpose I had a car-load of coal shipped to Connellsville, Pa., where I subjected it to a series of experiments which finally resulted successfully in the production of a hard, well-fused coke, of good color and uniform grain, showing a compressive strength, as determined by Mr. John Fulton, of 263 pounds, which (while not up to the Connellsville average of 301 pounds) is more than sufficient to meet the requirements of modern furnace-practice. Ovens are now being built and arrangements are in progress to place the coke upon the market. The presence of coking coal at this point will be of inestimable value to mines of nearly all classes of smelting-ore in the Hills. This coke will also, doubtless, find a ready market and large use in the smelting-districts of Montana, and possibly, also, in the, as yet, practically undeveloped mineral country of the Big Horn mountains; and I do not see any reason why it may not lead to the establishment of the manufacture of pig-iron and the train of industries which naturally follow. Iron-ores can doubtless be found sufficient to warrant the building of blast-furnaces, and that the manufacture of iron and steel should be profitable is almost self-evident. Should the expectation of finding natural gas in quantity be realized, the establishment of iron, steel and general metallurgical and industrial works will quickly and surely follow. Large markets are at hand; Dakota, Nebraska, Montana, Idaho, Wyoming, Utah and Colorado can be readily reached by routes much shorter than those from any existing iron- or steel-producing center.

With bituminous coal and coke of good quality, with lubricating-oil and illuminating-oil and the possibility of natural-gas, with ores of the precious metals, of iron, copper and tin in the central Hills, with a fertile soil, a beautiful country and healthy climate, the Black Hills are certainly capable of supporting a dense and prosperous population.

Northwest of the Black Hills, a number of uplifts have occurred,

similar in some respects to the Black Hills uplift; but in some cases at least, as at the Inyan Kara mountains, the older rocks have been abruptly forced up through the Mesozoic and the Dakota, leaving the latter abutting against the older formations in comparatively unchanged position.

The most prominent of these are:

1. The Inyan Kara mountains, rising abruptly one thousand or fifteen hundred feet above the surrounding country, and being, perhaps, half a mile wide by one mile long.

2. The Sun Dance mountains, very similar to the Inyan Kara.

3. The Bear Lodge mountains, a narrow range ending at the Sun Dance mountains.

4. The Missouri Buttes, three isolated dome-shaped mountains, nearly in line.

5. And lastly, the Devil's Tower, a columnar mountain rising, perhaps, eight hundred feet above the surrounding country, having a nearly flat top with an area of three or four acres, and almost or quite vertical sides about five hundred feet high, mounted upon a rocky base two or three hundred feet high.

I was not close enough to this mountain to examine it thoroughly. When viewed at a distance it appears to consist of jointed prisms similar to those of the Giants' Causeway, and resembles the core of an extinct volcano. If this is a volcanic core, then the three Missouri Buttes are probably of similar origin.

The Devil's Tower is one of the great natural curiosities of Wyoming, and will attract many tourists. Its vertical sides will effectually frustrate attempts to scale it by ordinary means.

These mountains all rise abruptly through the Dakota sandstone which forms the country immediately surrounding them.

Coal has been found in this formation at a number of places, the first discovery of importance being at the Hay Creek mines, located northwest of Deadwood and northeast of Sun Dance. The bed was quite variable in thickness and quality, and, while superior to the ordinary lignites of the Laramie group, was not pure enough to encourage active developments.

Some fairly good prospects were found between Sun Dance and Newcastle, on the west side of the Black Hills. The best area developed at this date is at Newcastle, the present terminus of the Black Hills branch of the Chicago, Burlington and Quincy system, on which the rails have just been laid.

The coal occurs, apparently, in the Dakota sandstone group,

which here consists of three massive sandstones, ranging from fifty to one hundred feet in thickness, and separated by soft shales and slates, the whole having a combined thickness of perhaps something more than three hundred feet. The principal coal-bed is found above the lower sandstone which directly overlies the red beds of the Jura-Triassic.

In the absence of proper palæontological evidence, we cannot positively assert the Dakota age of this coal-horizon, but the apparent integrity of this sandstone group as a whole, warrants the assumption that the coal is of Cretaceous age; for the upper member of the group seems certainly Dakota, and to whatever the top-rock belongs, the lower portion must also be assigned.

Shortly after explorations were commenced, I found that the outcrop of the coal had been burnt, and the search for coal became at once a search for ashes. The area in which the explorations were made was a plateau or table ranging from six hundred up to twelve hundred feet above the plains to the west (from which it gradually rises) and cut by frequent sharp cañons, two hundred to five hundred feet deep, along the sides of which the outcrop of the coal was usually above water-level.

In some localities the outcrop was readily found, and the full thickness of coal was exposed by drifting a few feet; but where the outcrop had been burnt the coal was rarely found until the ashes and burnt rock had been drifted upon from fifty to one hundred feet.

The coal is one of the varieties difficult to classify, bearing intimate relations to coals both of bituminous and of lignitic composition. It is black and lustrous, with a somewhat cubical fracture, and good coking-properties. From its appearance and coking-properties one might be justified in classing it as a true bituminous coal, but in chemical composition it is also closely allied to the lignites.

This coal will doubtless find a large market in Wyoming, Nebraska, Dakota and Montana. For domestic purposes, and for railroad-and steaming-use generally, it is far superior to any of the true lignites. The percentage of ash and water is moderate (averaging less than eight per cent.), being somewhat higher than Rock Springs coal. The samples analyzed contained more sulphur than Rock Springs coal, but less than many of the Iowa and Missouri coals on which Nebraska is now largely dependent; and the ash-percentage was very much lower than in most of these Missouri and Iowa coals. It is a strong steam-coal, is very hard,

does not slack upon exposure, and will bear transportation and handling well. Its coking-properties have already been described.

The Dakota sandstone is not generally known as a coal-bearing formation. In a paper read before the Institute, in October, 1888 (*Trans.* xvii., 375), Mr. G. C. Hewett notes the occurrence of coal in the Dakota sandstone between the White and Yampa rivers, in northwestern Colorado, which may correspond to the horizon of the Wyoming bed.

The developments made up to date indicate clearly that while in certain districts coal-beds of workable thickness and quality exist in this Dakota sandstone formation, the workable coal is not co-extensive with the formation; in other words, the presence of this formation in any locality is merely an indication that workable coal may possibly exist.

At the time my examinations were completed, the bed developed in the Newcastle field showed from 4 to 7½ feet of workable coal with no important slate partings. Subsequent explorations, I am informed, have shown the presence of a considerable area, ranging from 8 to 9 feet in thickness.

From the Belle Fourche west to the Big Horn mountains, the typical lignites of the Laramie group are finely exposed by repeated natural outcrops in cañons and cliffs; more especially on the head-waters of Donkey creek, Horse creek, Wild Horse creek, and the numerous affluents of the Powder river. These lignite beds range from 3 or 4 feet up to 22 feet in thickness, and are admirably situated for mining, being above water-level, and lying almost flat, so as to present many points at which self-draining mines may be opened and cheaply operated.

Averaging about 20 per cent. in water, these coals have low calorific value; and "slacking" upon exposure to the air (by drying), they will not bear transportation. Hence, while useful as a local fuel-supply, they are not likely to become commercially valuable until some cheap method is devised for artificially evaporating the water and manufacturing a "briquette," or artificial fuel. Their low per-centage of sulphur and ash would assure the production of a most excellent fuel, and the problem seems to be purely one of cost and of mastering the details of manufacture.

The area of this lignite-field is very great; certainly more than 7000 square miles; and that some method of utilizing this enormous body of fuel will ultimately be devised, I cannot doubt. I believe a fuel equal to the best bituminous coal could be produced

from these lignites artificially at a cost of about $2 per ton, and such a fuel would have one great advantage over ordinary bituminous coal, as it would reach the consumer in blocks (or eggs or spheres) of uniform shape, size and composition.

Looking west from the Black Hills, across this great intervening valley of the Belle Fourche, Cheyenne and Powder rivers to the Big Horn range, we see a country of great natural fertility, admirably adapted to the raising of stock and general farming. Along the principal water-courses are bottom-lands which can be successfully cultivated without artificial irrigation. Farmers located along these bottoms have great stretches of hilly country and higher ground along the divides and at the headwaters of all the streams, where excellent grazing can be had. These grazing-lands are not capable of cultivation without irrigation, but it is entirely evident that a large portion can be reclaimed by water taken from the larger streams.

West of the Powder river, and between the Powder river and the Big Horn range, is a stretch of country extending north and south a long distance parallel with the Big Horn range, which is now ready for close cultivation and will soon be producing enormous quantities of all kinds of farm-products. A large part of it has already been brought under irrigation by ditches taking water from tributaries of the Powder and the Tongue river, near where they emerge from the mountains, and carrying it far out over the foothills east of the range.

From Buffalo to Sheridan, along the waters of Clear creek, the Tongue and the Goose, north to the Crow reservation at the Montana line, the whole country is susceptible of high cultivation; and this area is destined to support a dense and prosperous community. As yet, the lack of railway facilities has prevented its more rapid settlement. It was impossible for the settlers already located there to dispose of their products, since it did not pay to transport them 150 miles by wagon to the railroad. When passing through the country in 1888, I found the prices of all kinds of farm-products ridiculously low, the supply being far in excess of the home-demand, which was limited to a few cattle-ranches in the vicinity, and the military post at Buffalo. With the advent of a railroad, all this will of course be changed, and the large yield of the irrigated lands will bring wealth and comfort, and encourage rapid settlement of the country.

No conditions more favorable to successful irrigation could be

conceived than those which exist in this region. On the snow-capped summits of the Big Horn range the rain-fall of seven or eight months of the year is stored, to be melted as the heat of the summer increases, thus giving a supply of water directly proportioned to the needs of the thirsty country below; for the warmer the weather the greater the supply.

The Big Horn range is as yet practically undeveloped so far as its mineral resources are concerned. I believe that important mineral discoveries will be made in these mountains, and that ore-deposits similar to those of the Black Hills, or possibly to those of Montana or of Colorado, will be found. Little systematic prospecting has yet been done. The search for minerals has been confined principally, so far, to prospecting with the pan for placer-deposits. No placers of extraordinary richness have yet been found, but good prospects are frequently reported, and reports have constantly been made of the discovery of tin, of large veins of iron-ore, of free milling gold-ore, and, I believe, also of copper.

The water of all the streams heading in the Big Horn mountains is beautiful, clear, pure mountain water, very rarely muddy; similarly, the water of the streams rising and flowing through the central portions of the Black Hills, is good, clear mountain spring water. In the plains country drained by Powder river and the Belle Fourche and Cheyenne, the water of the streams commonly contains a considerable percentage of the alkalies, but here and there are springs practically free from alkali. These springs are usually found issuing at the base of red buttes. The red color of these buttes has been caused by heat from the burning of thick beds of lignite which underlie all of the country bordering on the Powder river and its tributaries. At a few places, notably at the Burning Bank on Donkey creek, the lignite-beds may still be seen on fire, slowly burning and baking the slates and clay overlying the bed into natural brick and scoriae resembling volcanic tufa.

One of the most beautiful sights it has ever been my fortune to see was that of a ranch located along the banks of Clear creek, where for several miles the land on both sides had been brought under irrigation and a high state of cultivation. Enormous fields of oats, alfalfa, timothy and the wild blue-lance grass were being harvested. I believe sufficient fodder was raised at this ranch to feed something like three thousand head of cattle during the winter season, and I was told that the crop of blue-lance grass (which had never been plowed, but was simply irrigated in its natural condition) would

yield an average of about two tons of hay per acre. The yield of oats was quite large; I do not recall the exact amount, but I think in the neighborhood of one hundred bushels per acre.

In travelling between Buffalo and Sheridan I learned by conversation with farmers at whose places I stopped, that they were successful in raising all kinds of vegetables and all kinds of grasses and grain, excepting perhaps corn, for which they think the season rather short; but the yield of oats, wheat, rye and hay is certainly extraordinary.

At Buffalo one of the lignite-beds underlying the town was mined, and the coal was sold in the town and surrounding country for domestic use. It answers very well for all ordinary purposes, and at the time of my visit, was sold at $3.00 per ton, delivered.

Both the Black Hills and the Big Horn range are timbered more or less densely by the species of mountain pine so common throughout the mountainous portions of the west, and in some localities this timber grows of sufficient size to become valuable for framing and building purposes. In the northern part of the Black Hills it has been cut on a large scale for use at the Homestake and other mines. While perhaps it may not be profitable to ship this timber to a distance, it will always be valuable in furnishing a local supply for building-purposes, and for fire-wood. Its presence in both the Black Hills and Big Horn country is of inestimable value in the development of their mineral resources, furnishing a good, cheap mine-timber which frequently may be cut within the limits of the claim on which the mine is situated.

Northwest of the Hills in the neighborhood of Sun Dance, a little oak timber is found. It is small and scrubby, and is used to some extent for fence-posts. Aside from this, the only other timber of importance found in this part of Wyoming, is the cottonwood growing along the main water-courses, which from a commercial standpoint is practically worthless, but is of considerable value to the settler for fencing and fire-wood, and has been used to some extent in building cabins and stables.

ANALYSES OF LAKE SUPERIOR IRON-ORES.

BY GEO. W. GOETZ, M.E., MILWAUKEE, WIS.

(New York Meeting, September, 1890.)

THE following tables show the analyses of ores from nearly all of the mines which will produce over 10,000 tons of ore each during the season of 1890.

There are many small mines now in the course of development, some of which may become large producers. Many of the mines produce several different grades of ore, but the analyses given in the tables are fair averages of ore such as the respective mines can deliver.

Most of these analyses were obtained from the agents of the respective ores, and are averages of the season of 1889.

One can state positively that over 75 per cent. of all the iron-ores of the Lake Superior district will average over 60 per cent. of metallic iron.

During the past 34 years this region has produced about 48,137,-421 tons of ore, the different districts contributing about as follows:

Marquette District,	29,754,057
Menominee "	10,415,859
Gogebic "	5,622,194
Vermilion "	2,342,991
Miscellaneous,	2,320

In the year 1889, the product of all the districts was nearly 7,300,000 tons. The product of the entire region for the season of 1890 may exceed 9,000,000 tons. This large amount of ore is shipped to the lower lake ports, from Escanaba, Marquette, Two Harbors and Gladstone.

I. Marquette District.

Name of Ore.	Iron.	Phosphorus.	Silica.	Manganese.	Alumina.	Magnesia.	Lime.	Sulphur.
American	65.00	0.045	4.50					
Angeline—Hard	66.50	0.009						
" —Hematite	66.38	0.032						
Barnum	65.00	0.098	4.70	0.20	0.20	0.20	0.35	Trace.
Buffalo	62.33	0.120	6.80	0.37	1.61	0.06		
Cambria	60.00	0.064	7.26	0.37	3.81	0.33	0.62	0.042
Champion, No. 1	67.00	0.040	3.00					
Cleveland, No. 1	65.60	0.104	3.06	0.21	0.99	0.18	0.37	0.037
" Hematite	60.00	0.050	6.74	0.63	2.35	0.12	0.12	0.011
Cliff Shaft	62.70	0.109	3.45	0.52	1.55	1.15	1.61	0.025
Comrade	60.50	0.100						
Detroit	58.60	0.056	8.00					
East New York	60.10	0.055	11.64	0.41	1.11	0.31	0.11	0.029
Humboldt	65.00	0.120	5.00	0.12	0.72	0.26	0.64	0.020
Imperial	57.50	0.280	5.30	0.03	1.71	0.41	1.10	0.045
Jackson	62.00	0.090						
Lake Superior, No. 1A	64.00	0.110						
" " Slate	66.00	0.080	3.00	0.03	1.22	0.12	0.48	0.045
Lillie	59.00	0.084	8.69					
Lucy	53.50	0.045	12.05	4.25				
Michigamme	65.50	0.114	4.57	0.01	0.51	0.11	0.07	0.030
" No. 1	62.00	0.055	7.00	0.60	0.80	0.30	1.35	0.030
Milwaukee	62.05	0.105	4.32	0.45	0.60	0.90	1.10	0.080
Mitchell	60.95	0.131	6.85	0.62	1.35	0.30	0.42	0.020
Negaunee	60.00							
Prince of Wales	62.56	0.106	5.51	0.37	1.70	0.17	0.59	0.053
Prout	53.02	0.041	17.65					
Queen	62.60	0.112	4.65	0.41	1.11	0.31	0.12	0.029
Republic, Magnetic	69.77	0.051	1.78					
" Specular	67.50	0.043	3.01	0.04	0.85	0.07	0.41	0.012
Salisbury Bess	62.00	0.045	4.00	0.22	0.50	0.60	1.00	0.012
Sharon	61.00	0.090	8.00	0.12	0.72	0.26	0.84	0.012
Sheffield	61.00	0.015						
South Buffalo	63.80	0.100	4.10	0.31	1.49	0.34	0.82	0.049
Volunteer	60.10	0.070						
Winthrop	63.40	0.109	5.37	0.85	0.81	0.11	0.24	0.028

II. Menominee District.

Aragon	65.20	0.038	4.68	0.02	0.33	0.01	0.36	0.034
Armenia	60.80	0.188	8.69	0.11	1.33	0.39	0.66	0.022
Chapin	62.13	0.068	4.20	0.26	1.08	2.71	0.81	0.001
Commonwealth	57.70							
Corbett	58.20	0.675	4.37	0.31	3.10	0.62	0.78	0.041
Crescent	64.55	0.044	3.35	0.15	0.67	1.64	0.50	
Cyclops	61.00							
Dunn	60.60	0.344	4.81	0.22	2.98	0.81	0.97	0.036
Eagle	55.20	0.547	5.03	0.02		0.81	5.91	0.061
Florence	60.50	0.257	4.05	0.22	0.81	0.64	1.18	0.041
Great Western								
Iron River	59.80	0.356	5.29	0.37	3.41	0.37	0.69	0.028

II. MENOMINEE DISTRICT—*Continued.*

NAME OF ORE.	Iron.	Phosphorus.	Silica.	Manganese.	Alumina.	Magnesia.	Lime.	Sulphur.
Ludington	64.59	0.070						
Manganate	54.00	0.642	6.00	3.50				
Mansfield	63.40	0.025	3.56	0.55	2.28	8.00	4.00	0.033
Mastodon	58.36							
Millie	64.06	0.027	3.00	0.08	1.30	0.95	1.10	0.035
Norway	58.00	0.120	12.00					
Paint River	56.40	0.502	3.48	0.37	6.12	0.72	5.58	0.068
Pewabic	65.00	0.010						
Rex	55.78	0.080	6.85		1.67			
Shafer	59.00							
Vulcan	62.00							
Youngstown	52.18	0.642	6.00	3.50				

III. GOGEBIC DISTRICT.

Anvil	61.54	0.053						
Ashland	63.82	0.038						
Aurora	63.36	0.034	4.00	0.03	1.98	0.12	0 04	0.033
Brotherton	62.50	0.037	6.13	0.49	2.98	0.39	0.16	0 070
Cary	58.84	0.062	5.20			3.66		
Eureka	62.00	0.076	6.64	0.48	3.69	0.16	0.15	0.015
Germania	59.19	0.058						
Hennepin	65.45	0.048	3.53	0.02	0.31	0.02	0.02	0.022
Iron Belt	62.75	0.042	6.33		2.44			0.037
Ironton	60.55	0.058						
Montreal	64.22	0.053						
Mount Hope	62.32	0.029						
Norrie	63.72	0.043	3.40					
Pabst	62.34	0.037	4.50					
Palms	62.43	0 049						
Ruby	62.50	0.045	4.20	0.49		0.16	0.45	
Section 33, North	64.82	0.045						
Sunday Lake	62.00	0.036						
Trezona	62.55	0.061	4.44	0.02	0.97	0.03	0.02	0.031
West Cary	54.95	0.054	5.35	7.41				
Windsor	62.00	0.051	6.30	0.81	3.45	0.31	0.40	0.025

IV. VERMILION DISTRICT.

Braddock	65.17	0.052	4.50	0.13	0.48	0.16	0.56	0.020
Chandler	64.15	0.044	4.30	0.14	2.25	0.19	0 25	0.040
Long Lake	60.20	0.053	7.00	0 54	2.75	0.19	0.25	0.047
Minnesota	67.54	0.051	1.97	0.15	0.40	0.14	0.53	0.020
Nipigon	63.01	0.053	6.75	0.14	0.65	0.20	0.60	Tr'ce
Red Lake	62.86	0.099	6.95	Trace.	0.65	0 20	0.60	"
Soudan	65.77	0.102	4.25	0.16	0.38	0.17	0 56	0.020
Vermilion	67.31	0.091	2.15	0.15	0.35	0.15	0.58	0.020

MAGNETIC CONCENTRATION AT THE MICHIGAMME IRON-MINE, LAKE SUPERIOR.

BY JOHN C. FOWLE, MICHIGAMME, MICH.

(New York Meeting, September, 1890.)

HAVING had for many years the management of magnetite mines, and having noted the various admixtures, such as jasper, "green rock," actinolite, etc., that occur so frequently in the deposits and make the fine ore so difficult as well as expensive to sort by hand, I studied the different ways of mechanical sorting, but found none satisfactory—the universal objections being the expense, the small quantity that could be handled per hour, and the low grade of ore produced after all. Feeling that some means would be discovered ultimately for concentrating the small ore, I began at the Michigamme mine, at Michigamme, Lake Superior, Michigan, to separate it from the larger and richer lumps, by placing in all the shaft-houses screens of round iron bars set $\frac{1}{4}$-inch apart, on which the skips were dumped as they were hoisted; the coarse ore going into one stock-pile or into cars, and the small ore, from 2 or 3 inches diameter down to powder, going into a special stock-pile, to be treated as soon as some proper means was found to separate the ore from the rock and other impurities.

Careful investigations showed that neither dry nor wet concentration was the right process for us; and I finally studied the magnetic process. Not knowing of any magnetic separator in this country that would handle such large pieces of ore as I wished to treat, I looked abroad and found in Sweden the Wenström separator. One of these machines was brought to the United States and tested in New Jersey and other places. (See the papers of Messrs. Cook and Birkinbine, *Trans.*, xvii., 599 and 728.) This original separator, having 15-inch face of drum, I brought to the Michigamme mines, and, no one having confidence in it, placed it in operation at my own expense. Finding that in this way large ore could be treated much better by magnetic separators than by hand, and with much greater enrichment of the product, I put up a small, crude mill with several series of screens, so as to size the material and

treat on the separator the different sizes by themselves. This gave a still better product. None but the 15-inch separator was used. We shipped from this mill 11,000 tons of concentrates in 1889, a part of which, produced from crude 50 per cent. ore, contained 65 per cent. of iron, while the remainder, produced from crude ore running 52 to 54 per cent., carried 60 per cent. of iron.

In view of this satisfactory result in the separation of large and small pieces of ore, my attention was turned to the large number of old dumps about the mine containing a large amount of iron. Finally, crushers were put in to break up this mixed ore and rock; but soft chlorite and hornblende, which were present in large quantity, used to pack the crushers so that they were continually breaking down, and it proved impossible with crushers alone to reduce any considerable quantity of this mixed ore to pass a $\frac{5}{8}$-inch screen—the size which had been found in practice most advantageous for securing the maximum of product consistent with sufficient coarseness to suit the conditions of blast-furnace use. We bring the ore screened to this size up to 61 per cent. in iron, and can raise the iron a few points by carrying less current on the dynamo, which makes the tailings richer.

The experience at this mill shows that the fine powdered ore and the ore going through a $\frac{1}{4}$-inch screen, should never be allowed to fall directly on the separator, but should be carried near the separator by a belt on an inclined plane and attracted to the drum of the separator by the electro-magnetic force. It is almost impossible to feed this fine ore directly on to a separator in a sheet sufficiently thin to permit a satisfactory separation, because ore and rock overlying one another are bound together in the drum; but by feeding the fine ore by a belt up to the separator, the mass of material is agitated, and the ore flies to the drum and the rock falls, or remains on the belt. The greater the electric current which is carried on the separator, and the farther away from the separator the crude ore is when it enters the magnetic field, the higher will be the percentage of iron in the concentrates and the lower in the tailings. For instance, crude ore, containing 52 per cent. of iron and 0.224 of phosphorus, when treated directly on the separator, gave concentrates containing in one case 58 per cent. of iron and 0.215 of phosphorus; and in another case 60 per cent. of iron and 0.180 of phosphorus; while the same ore, treated by a belt-feed, not in contact with the separator, gave concentrates containing 67.07 per cent. of iron and 0.060 of phosphorus.

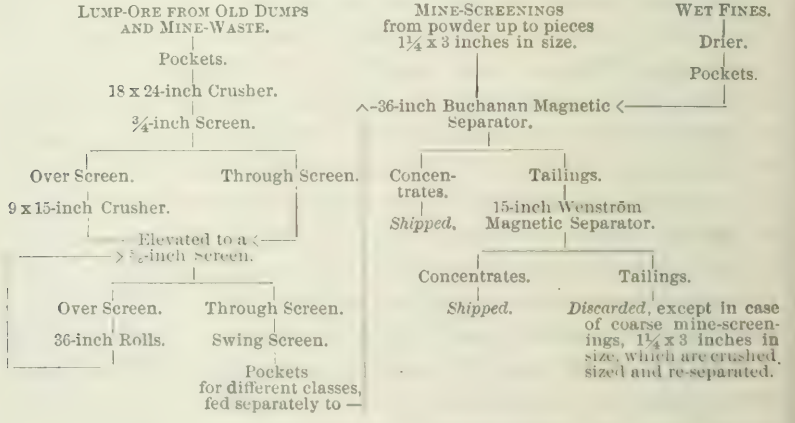
All of our underground skips that carry lean rock, or "waste," are dumped at the shaft-houses upon long, flat screens made of round bar-iron, with bars $1\frac{1}{4}$ inches apart. The material passing these screens ranges in size from powder up to pieces $1\frac{1}{4}$ inches thick, by 3 inches wide, some of which weigh as much as 4 pounds each. These screenings and the lean lump-ore are conveyed to the concentrating-mill over gravity roads. The tracks extend over the top of the mill, and facilitate a convenient delivery into separate large ore-pockets.

On the east and west sides of the mill are skip-roads, by which all the mixed ore from old dumps is hoisted up into the pockets over the crushers. These skip-roads run under the railroad-track at a vertical interval which permits the interposition of a pocket holding a railroad-carload of ore, 8 tons. By this arrangement the refuse from all parts of the property, when loaded into cars, can be switched at small cost to these pockets and dumped into skips.

The mill built here is 50 x 60 feet, and has cost $16,000. The machinery includes one 60-horse power engine, two dynamos run by one engine, one boiler, two drums for hoisting skips, one 18 x 24-inch crusher, one 9 x 15-inch crusher, one set of 36-inch rolls, one 36-inch Buchanan separator (Fig. 1), one 15-inch Wenström separator (Fig. 2), one shaking-screen and a swing-screen.

The method of treatment in the mill is shown in the accompanying scheme.

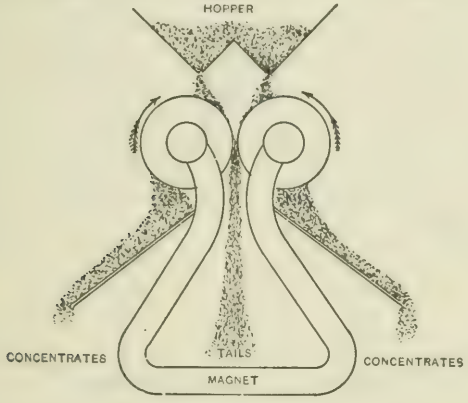
Scheme for Concentration.



The 18 x 24-inch crusher breaks even the largest of our lump-ore; its use saves us much expense in sledging. A coarse screen placed

beneath this crusher sifts out the first fines, and delivers all the coarser product to the 9 x 15-inch crusher. Upon passing through this machine, the ore and the first screenings are raised by a link-

Fig. 1.

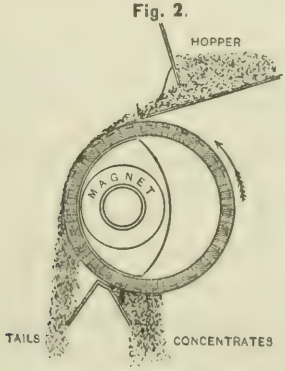


Buchanan Separator.

belt elevator, and delivered to a $\frac{5}{8}$-inch screen. Such material as is rejected by this screen, goes to the rolls, and is then again raised by the same elevator and re-sized as before.

The screen which we employ is a common coal-screen, suspended at top and bottom, and agitated by means of cams. We find this form

Fig. 2.

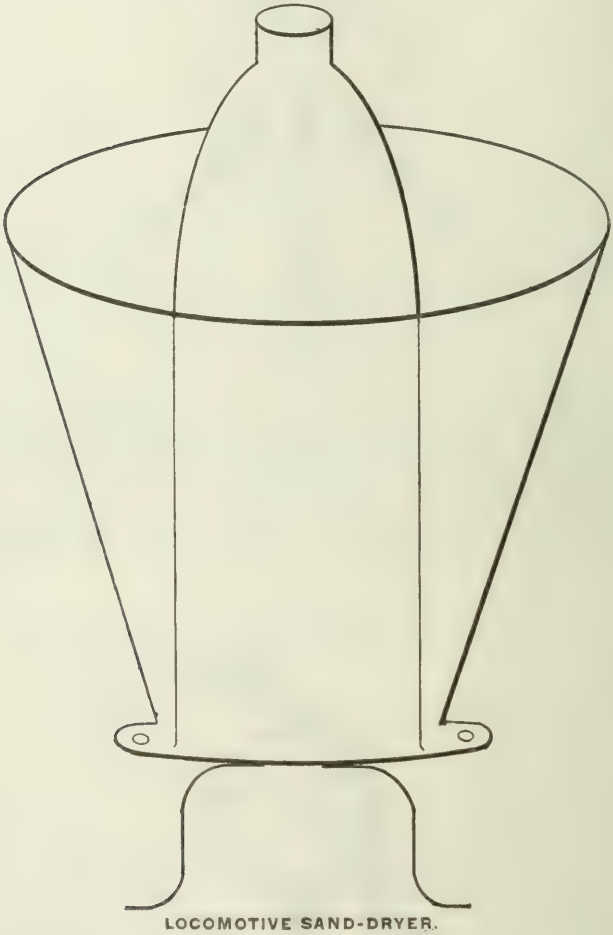


Wenström Separator.

durable, cheap and of great capacity—more so than revolving screens. Crushers, rolls, elevator and screen all work automatically, one man only attending to all.

The fine ore that passes through the screen, is delivered to a swing-screen, from which it drops into several pockets, and is then fed by gravity to the Buchanan separator. This separator also treats, independently of the crushed ore, all the previously-mentioned mine-screenings. These are fed to it direct from storage-pockets, without

Fig. 3.



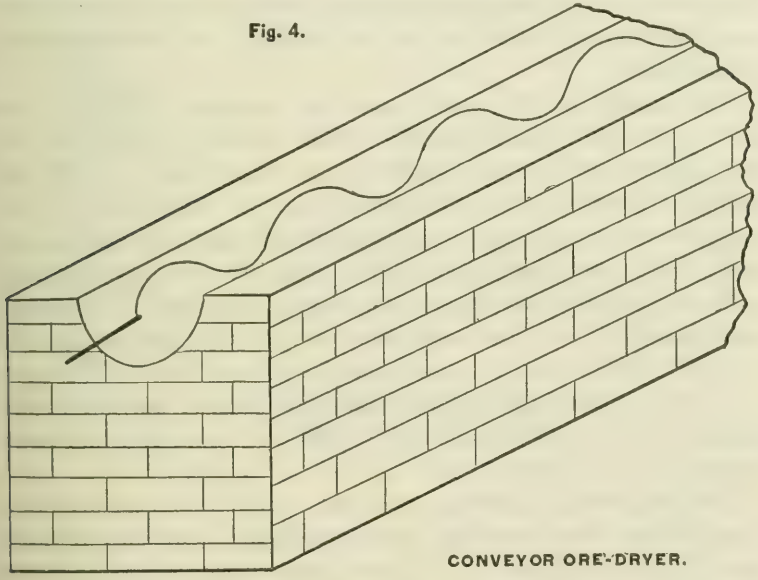
LOCOMOTIVE SAND-DRYER.

any preparatory crushing or sizing. The concentrate, or "ore," produced by the separator, is conveyed to railroad-cars and loaded for shipment. The tailings are run over our Wenström separator to take out any remaining ore. The Buchanan separator is very well built; it has a much larger capacity than the small Wenström

machine, and makes richer concentrates. It carries a current of 23 ampères, and is wound with heavy copper wire. The Wenström machine, on the other hand, can carry only 10 ampères, and is wound with wire of one-third the size. We keep the Wenström machine in use only because we happen to have it. It is our original Swedish machine and is a good one—much better, in fact, than one which was built for us in this country but which we do not use.

A large amount of fine powdered ore and $\frac{1}{4}$-inch screen-ore comes from the Michigamme mine in a wet state, and in order to make a good separation, it must be well dried. We have used locomotive sand-dryers, Fig. 3. They are good, but their capacity is too

Fig. 4.



CONVEYOR ORE-DRYER.

small. We also use a dryer, Fig. 4, made by using a half-round cast-iron trough, 16 feet long, with a spiral conveyor running in it, to mix up and agitate the wet, fine ore. This trough is set in brick, with a fire-box underneath to burn wood. Wet ore fed into the trough at one end comes out dry at the other end, where it is elevated into the pocket, from which it runs by gravity to the separator. This dryer is cheap and easily built, and by using heavy conveyor-flights, can be made durable.

To avoid danger from fire and the cost of a fireman to tend the dryer, we are putting in a revolving conveyor to dry by steam.

Fine wet ore is to pass by gravity from the feed-hopper through four tubes, which are fixed in an inclined, revolving drum. Live- or

exhaust-steam enters the drum through a stuffing-box at one end, surrounds the ore-tubes, and thoroughly dries the ore. A slightly different drying-drum has been designed, in which one large tube takes the place of the four small ones. It is expected that this second form will work satisfactorily when the fines are not extremely wet. The drier the material that is to be separated, whether coarse or fine, the greater the capacity of the separator. Up to present writing (August 1) we have shipped in this season from this mill 8000 tons of concentrates, and are now daily concentrating 180 to 200 tons, at a cost, including crushing, hoisting material into mill-pockets, separating ore and loading it into cars, of 18 cents per ton. On some days the concentrates at this mill cost us only 10 cents a ton, delivered in the railroad cars, including all the above-mentioned items of cost.*

The quality of the concentrates is regulated entirely by the electric current and the sizing of the material. Most of our concentrates will pass a $\frac{3}{4}$-inch screen, though some, which are produced from the mine-screenings, are of larger size. These mine-screenings go directly to the separator as 52- or 56-per cent. ore, and leave it as 64- to 65-per cent. ore. Considering their wide range of size, we regard this as very fair work. The waste-rock, resulting from the separation of the screenings, amounts to 10 to 15 per cent. of the total quantity of crude ore. Whenever this waste-product is considered sufficiently rich to warrant further treatment, it is crushed and sized with the lump-ore, and passes through a second separation.

The proportion of waste which we make from crushed ore is much larger than from the mine-screenings. This only indicates that the lean lump-ore is of poorer grade than the more friable material which breaks into small pieces in mining and forms the screenings. The amount of tailings varies, of course, with the kind of material we are crushing. It fluctuates between 30 and 50 per cent. for most of our separated ore, which runs in size from $\frac{1}{4}$ to $\frac{3}{4}$-inch. The tailings carry from 21 to 24 per cent. of iron. As we break down by far the largest proportion of our rock so as only to pass a $\frac{5}{8}$- to $\frac{3}{4}$-inch screen, it is evident that a lower percentage of iron could easily be obtained by finer crushing. But would such

* Since this paper was written, the complete returns for August have been made up, and show 3116 tons of concentrates shipped during the month, at the cost of 22 cents per ton for all the items named above. The total shipments to October 12th amounted to 15,000 tons.

practice be wise policy? I prefer to make a concentrate which sells readily at a good profit; and, to this end, the concentrate must be of such size that it can be worked at low cost and in large quantities by the blast-furnaces.

Assays of fine powdered ore up to $\frac{1}{4}$-inch screen-size:

1. Crude ore, 61.30 per cent. iron.
   " " .131 " phosphorus.

Concentrates.

	Per cent.
Iron,	64.68
Phosphorus,	.086
Iron,	67.07
Phosphorus,	.080
Iron,	65.99
Phosphorus,	.087
Iron,	63.70
Phosphorus,	.086

Assays of concentrates of mixed ores from $\frac{1}{4}$-inch screens up to pieces weighing 2 to 3 pounds, concentrated at the Michigamme mine by J. C. Fowle, from July 1 to December 1, 1889:

2. Crude ore, 52 to 54 per cent. iron.
   Tailings, 21 to 24 " " "

Concentrates.

	Per cent.
Iron,	62.14
"	63.11
"	61.30
"	62.72
"	64.87
"	63.66
"	63.00
"	63.50
"	63.66
"	60.93
"	62.12
"	60.38
"	60.20
"	60.66

Assay of fine powdered ore up to $\frac{1}{4}$-inch screen-size:

3. Crude ore, 54 per cent. iron.

Concentrates.

	Per cent.
Iron,	62.72
"	63.50

4. Crude ore, 56 per cent. iron.

Concentrates.

	Per cent.
Iron,	64.86
"	63.00
"	63.11

5. Crude ore, 58 per cent. iron.

Concentrates.

	Per cent.
Iron,	64.08
"	63.00
"	63.60
"	66.83

6. Crude ore, 56.12 per cent. iron.
" " .050 " phosphorus.

Concentrates.

	Per cent.
Iron,	62.40
Phosphorus,	.040

The ore was fed directly on to the drum of the separator; by using a belt-feed the concentrates could, in my judgment, have been brought to assay 66 to 67 per cent. iron, and only 0.02 per cent. phosphorus.

7. Crude ore, 61.30 per cent. iron.
" " .114 " phosphorus.

Concentrates.

	Per cent.
Iron,	66.83
Phosphorus,	.090

I consider the concentration of magnetic ores by electricity to be still in its infancy as to cost, capacity and quality. Our experience has been exclusively with separators having one or more drums, and I claim no knowledge of practice with other forms of apparatus. The many mills now in operation in the East will undoubtedly, during the present year, work out new methods for concentrating iron-ores, whereby the cost will be reduced and the output still further increased.

ORE-DRESSING BY ELECTRICITY AT THE TILLY FOSTER MINE.

BY F. H. McDOWELL, NEW YORK CITY.

(New York Meeting, September, 1890.)

THE electro-magnetic separator of Mr. G. Conkling, Glens Falls, N. Y., described by Mr. Birkinbine* in a paper at the New York meeting of the Institute, in February, 1889, has been in successful operation at the Tilly Foster mine for a considerable period, under the disadvantages which seem to be the inevitable attendants of pioneer enterprises in new technical fields. The results of a run of several months, which the Superintendent, Mr. S. P. Tomkins, has taken pains to collate, are here presented by his courteous permission, and that of Mr. E. S. Moffat, the energetic and skilful General Manager of the Lackawanna Iron and Coal Company, to whom our profession is already indebted for examples of bold and ingenious practice in mine-engineering. It is believed that these figures will assist those who are planning similar operations to decide intelligently what they may, and, more important still, what they may not, expect to accomplish.

The local difficulties in the way of this experiment at the Tilly Foster mine were many and great. Only about two-thirds of the waste-dump, resulting from the assorting of previous years, is ore; and this varies in content of iron from 20 to 28 per cent. The remainder is rock, some (from horses in the vein) showing particles of adhering mineral, some (from the foot- and hanging-walls) entirely barren. The expense of overhauling the dump and removing this rock is by no means the least of the numerous items which must be charged against the concentration. Associated with the lean ore are hornblende and other minerals, carrying iron in the non-magnetic state, which must necessarily be washed away with the tailings. The waste-dump was originally located without reference to future treatment, and now can only be handled by means of a locomotive,

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* *Trans.*, xvii., 739.

operating over heavy grades. The mineral is so widely and finely disseminated through the ore that it has been necessary to resort to fine-crushing. This (water being used in the separation) increases the proportion of slimes, which carry off mechanically small particles of mineral. Finally, the lean character of the ore calls for the handling and conveying from the mill of a large bulk of tailings. The process includes crushing the ore by a Blake rock-breaker on the dump; removing it in train-loads to the bins in the mill; passing it under two Ball stamps, provided with screens of $\frac{5}{16}$-inch mesh; elevating it to the Conkling electrical separating-belts; and delivering the concentrates to the cars and the tailings to the settling-reservoirs. It all seems simple enough in a printed description; but many practical difficulties have been met and overcome by Mr. Tomkins, to appreciate which one must visit the works and note the changes which have been made from time to time with increasing experience. In the face of these changes and the frequent accidents and break-downs incidental to the introduction of new machinery, the mill has been kept running almost continuously ten hours out of the twenty-four, and this has contributed not a little to the success of the undertaking.

By reference to the accompanying table it will be noted that, during the six months ending July 31, 1890 (the work of January being omitted from the calculation), there were treated 18,058 tons of lean ore, an average of 3009 tons per month. The concentrates produced amounted to 6236 tons, an average of 1039 tons per month. There were required, therefore, 2.89 tons of crude ore to produce one ton of concentrates. The average cost of treatment, including preparing the ore and getting it to mill, and disposing of the tailings (which items make up one-half of the charge for labor), was $2.25 per ton of concentrates. The actual expenses in the mill have averaged $1.63 per ton of concentrates. These figures will be materially reduced when the supply of waste ore from the dump has been exhausted, and a higher grade of lean ore is sent directly from the mine to the mill.

Since the following table was compiled the figures for August, 1890, have been received. They show 1391 tons concentrates from 3508 tons of crude ore, or 1 : 2.52. The cost per ton of concentrates was $1.89, as compared with $2.39 in July. The reduction of cost is due to the increased output of the mill in concentrates, a larger amount of crude ore having been treated, and also a larger yield obtained per ton of crude ore.

Table of Results for Seven Months of 1890.

	January.	February.	March.	April.	May.	June.	July.	Average For 6 mos. Beginning Feb'y 1.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Ore used	1719	3007	2278	3104	3619	3120	2930	3009
Concentrates made	445	893	785	1254	1176	1062	1066	1039
One ton of concentrate from	3.75	3.36	2.90	2.47	3.07	2.93	2.61	2.89
Labor, crude to crusher								
" crushing, and loading cars								
" repairing cars	.....	$0.66	$0.59	$0.47	$0.60	$0.63	$0.66	.....
" " carts								
" " wheelbarrows								
" locomotive crew								
" mill work and handling concentrates	.....	0.48	0.54	0.37	0.42	0.45	0.44	.....
" in ore-bins	.....	0.07	0.08	0.06	0.07	0.06	0.06	.....
" handling tailings	.....	.....	0.03	0.01	0.03	0.03	0.02	.....
" repairs of mill & mach'ry	.....	.....	0.17	0.06	0.08	0.10	0.11	.....
Analyses	.....	.....	.....	.....	.....	.....	0.03	.....
Material for repairs	.....	0.40	0.17	0.18	0.19	0.23	0.27	.....
Supplies, oil, waste, etc	.....	0.16	0.23	0.07	0.21	0.21	0.16	.....
Coal used	.....	0.66	0.74	0.53	0.55	0.63	0.64	.....
Total cost of one ton concentrates	$3.03	$2.43	$2.55	$1.75	$2.15	$2.34	$2.39	$2.25
Cost of labor in one ton "	.....	$1.21	$1.41	$0.97	$1.20	$1.27	$1.32	$1.26
Cost to crush and put crude ore in mill	.....	.19 7/10	.20 3/10	.18 5/10	.19 3/10	.21	.25 3/10	.....
Per cent. of iron in crude ore sent to mill	25.13	24.96	28.57	28.21	26.80	28.28	27.55	27.39
Per cent. of iron in concentrates	47.40	50.20	49.54	48.75	50.19	51.04	49.14	.....
" " " tailings	.....	.....	.....	.....	11.53	10.32	10.21	.....
Mill run.....Days	.....	20 8/10	15 7/10	23 8/10	25 3/10	22 2/10	22 2/10	.....

The following practical conclusions are suggested by these results:

1. Unless the location and other conditions are exceptionally favorable, it will not pay to erect works to treat the material of waste-dumps carrying less than 25 per cent. of iron.

2. Where the lean ore is mined in connection with shipping-ore, there must be a corresponding increase in the percentage of iron to offset the mining and royalty charges.

3. Where no shipping-ore is produced, there must be a still further increase in the percentage of iron, to warrant the erection of hoisting-, pumping- and dressing-works.

Mine-owners will readily understand that no rigid rules can be laid down where the ore-characteristics and local conditions are constantly varying. Before establishing large works, the most thorough and comprehensive tests should be made.

THE POTOSÍ, BOLIVIA, SILVER-DISTRICT.

BY ARTHUR F. WENDT, NEW YORK CITY.

(New York Meeting, September, 1890.)

Introduction.—In the year 1885, the writer signed a contract to take charge of the reopening and developing of the celebrated old mines of "El Cerro Rico de Potosí." Two years earlier, an English firm of engineers had made a very glowing report on the property. In the summer of 1885, telegrams arrived from the parties in charge of operations at the mountain, claiming the finding of one of the old veins, with an average richness of over 200 ounces to the ton. In the fall of 1886, the author first personally visited Potosí, but found neither the English report nor the telegram borne out by the facts. After an expenditure of over half a million of dollars and more than three years of continuous hard work, the writer succeeded in unwatering and developing one of the principal old veins of the mountain. This vein, called "Cotamitos," now yields ores of about 75 ounces to the ton, and the old historic mountains of Potosí may be said to have again fairly entered the list of silver-producers. The experience of the author, during his residence in Bolivia, and data obtained from Sr. Don F. Avelino Aramayo, form the basis of this description.

Potosí.

Etymology of the Name.—The Quichuá (pronounced Keechua) Indians, known to North American students as the "Incas," although the word "inca" merely signifies "king," or "chief," of the Quichuás, called the mountain "Sumac-orcko," or "the beautiful mountain." Eighty-three years before the discovery of the mountain by the Spaniards, the inca, or king, Huaina Ccapac, travelling from Cantumarca to Colque-Porco, saw the mountain of Potosí and, admiring its grandeur and beauty, said that within its bowels there was, no doubt, much silver. He ordered his vassals to work the mines. They, having carried their mining implements to the mountain, ascended it and were on the point of commencing work, when they heard a voice of thunder, which said: "Do not take

away the silver of this mountain, for it is for other owners." They returned to their king, and referred to the mountain as "the mountain of great noises," or "thunder," called in the Quichuá language "Potoçsi." This legend gave the name to the mountain and had no doubt its origin in the terrific thunder-storms which play around the summit of the Cerro.

Discovery of the Mountain of Potosí, 1545.—The Quichuá Indian, Diego Guallca, a native of Chunvivillca, near Cuzco, and a serf of Captain Juan de Villarroel, a miner of Porco, a mining-center, situated 15 leagues from Potosí, is said to have been in search of some lost llamas, and to have ascended the side of the mountain. Night overtaking the Indian, and the night being very cold, he made a fire with hay and small twigs, and slept on the mountain. The following morning, the rock below the ashes of his fire had melted, and a "river" of molten silver had formed. Guallca took some of the ore, returned to Porco, and there melted it and revealed his secret to Guanca, who, in turn, told Captain Juan de Villarroel, the miner above mentioned. The mines of Porco had been worked for many centuries by the incas, or kings, of the Quichuás. The writer has there seen tunnels, hundreds of yards in length, cut in the trachyte rock of the mountain by knives or chisels, and straighter and more perfect and true than can to-day be made by the use of powder. Systematic mining was carried on by the incas. The ores of the Porco mines were galena, blende, ruby silver and other silver minerals; and the incas extracted the silver by smelting the ore, cupelling the resulting bullion, and returning the litharge for further smelting. Large slag-dumps still attest to the former mining activity.

In December, 1545, eleven months after the discovery of the Potosí deposits by the Indian, Guallca, the mining captains Juan de Villarroel, Santandia, Diego Centeño and Pedro Cotamito commenced regular work on the mountain. Some of their names are found to this day preserved among the veins and mines of the district.

Historical.—The first vein was registered in April, 1545. From a letter written by Calderon de Salcedo to the king of Spain, and from other manuscripts, it is certain that the mountain produced between 1545 and 1572 surface-ores of great richness, native silver and chlorides yielding 25 per cent. of silver. All these ores were treated by smelting in small furnaces, called "guairachinas." These furnaces are undoubtedly the same as those which had been formerly used at Porco by the incas. The manner of procedure was to mix

the ore with litharge and charcoal, smelt it in a small shaft-furnace, and cupel the silver-lead produced. Blast was not used, but the furnace turned in the direction of the wind. The friar, Joseph de Acosta, estimates that $250,000,000 were produced in this way between 1545 and 1572, or nearly $10,000,000 a year.

In 1556, the Spanish government imposed a duty on the silver extracted from the ores; but naturally the amount of duty collected does not give a true measure of the total silver extracted from the mines, for all the silver used for the manufacture of ecclesiastical ornaments, articles of domestic use, and much that was smuggled away, escaped duty. However, the duty paid, and of which records were found, would seem to show an annual production of about $4,000,000 per year, during 246 years.

In 1572, the duties paid to the Crown had fallen off very seriously, owing to the fact that the rich surface-ores, susceptible of smelting, had been exhausted. The Spanish government dispatched a new viceroy, named Toledo, to increase the production and, incidentally, the revenue; and he, with the help of Don Pedro Fernandez de Velasco, introduced the "Patio" process. This process is supposed to have been invented only 15 years before, in the year 1557, by Bartolomé Medina, a miner of Pachuca, Mexico. The viceroy, Toledo, collected Indians from all parts of Peru, to be employed as slaves in carrying on the work. The Crown of Spain at that time not only imposed a government duty on the silver extracted, but also sold to the haciendas the necessary quicksilver, which was produced in large quantity from the mines of Guancavelica, Peru.

In 1599, great complaints were made that it was difficult to find rich ores easy of access, and that the poor ore yielded no profit.

In 1618, the Miners' Corporation addressed the king of Spain, praying for a reduction of duties. It represented that the ores then yielded at the rate of 52 ounces of silver per ton.

In 1621, large reservoirs for supplying the stamping- and amalgamating-works with power and water were completed. These reservoirs are used to the present day, although many of them are now in poor condition. One hundred and thirty-six reduction-establishments are said to have been in operation in 1621.

In 1623, the bloody civil war in Spain between the Basques and Andalusians extended to Potosí, and caused the abandonment of many mines.

On the 3d of March, 1626, during the rainy season, the dam of one of the reservoirs gave way, and the liberated flood of water de-

stroyed nearly all the reduction-works, causing a loss, according to Salcedo, of over $10,000,000. The Spanish government assisted the mining operations with a loan; and in 1633 the mines are said to have recovered to some extent from the effects of the disaster. The chief revenue was still derived from the superficial deposits, yielding about 50 ounces to the ton.

In 1636, the mining corporation again sent an address to the king of Spain. There were one hundred and thirty establishments, each having a capacity of 40 to 50 tons of 50-ounce ore per week; thus showing a productive capacity at the rate of over 12,000,000 ounces per annum. The corporation again complained of small profits. It represented that there were 70,000 Indians in Potosí, of whom 4707 had been allotted for working in the mines, but that only 1500 really went regularly to work. At this time the annual expenditure on law-suits was said to have been $200,000. Some of the papers of these law-suits are still in existence, and give a very exact history of the condition of the mining industry.

It is on record that in 1660 one Brigadier Montes, owner of the Guaillahuasi Mine, lost a suit brought by Estéves, owner of the Flamencos mine. Montes had cross-cut from the Guaillahuasi to the Flamencos vein, below the workings of Estéves, and claimed to be working another vein. The report says that six hundred men were at that time at work extracting 150-ounce ore for Montes. When the suit was decided against Montes he is said to have destroyed the mine. Be that as it may, neither the Flamencos nor the Guaillahuasi mine have been worked for over a century, although both are claimed to be among the richest on the mountain. It appears from this law-suit that considerable depth had been attained by the mines of that period, and that the ridiculous law of following a vein, which then held good in all Spanish-America, and still obtains in the United States, was even then resulting in interminable legal controversies.

In 1690 Quiroga worked the Cotamitos mine. Quiroga is on record as having paid $21,000,000 for his fifth, or duty, to the Crown of Spain. From the great profits which he derived from the mine he built the Cathedral of San Francisco in the City of Potosí, and there his remains lie buried.

In 1693 the production of the mines began to fall off very considerably. Count Camillo says that the government allotted Indians to less than one hundred reduction-establishments.

In 1712 Peru was ravaged by a fatal epidemic which destroyed many of the Indians.

In 1739 the tax on the production of silver was reduced from 20 to 10 per cent.

In 1759 a very full report was presented to the king of Spain by the Miners' Corporation. This report declares that the veins varied in width from a knife-edge to several yards; that the works on these veins had attained the depth of many hundred yards; that the deepest mines were becoming obstructed by water and want of air; and that the adits and shafts formed a complete labyrinth not understood by anybody. In the upper third, or three hundred yards, of the height of the mountain, there was no vein which had not been worked. In the Mendieta vein three hundred Indians and two miners were buried alive in a rich part of the mine, which had never since been reopened. From these causes the miners were quite unable to follow the veins in depth: "a great misfortune and drawback to the community, because the deepest ores are commonly the richest." The reporter mentions the Cotamitos vein as presenting the lowest workings. "And from this," says he, "metal was taken in 1720 like fringes of solid white silver, without any tarnish, and some of the finest kind of sulphides."

This is the first reference we have in any of the official reports to the working of sulphides. At that time the "pacos," or decomposed ores, were referred to as "a miserable resource;" and it is evident that they were practically exhausted.

Sulphides have certainly been worked in Potosí since the year 1609, and probably a considerable time before that date; and the stories that have been told about the inability of the ancient Spanish miners to reduce sulphides are not, in the author's opinion, based on facts. It is quite probable that the proportion of silver extracted was not as great as now, although even this is doubtful; but there is no doubt whatever that the oft-repeated assertion that the ancient Spanish miners stopped mining on reaching sulphides is a mistake. The Curé Alvaro Alonso de Barba, of the parish San Bernardo, near Potosí, in his work *Del Arte de los Metales*, published in 1640, describes a process of treating rebellious ores by means of cooking in a copper vessel, which is the "Fondo" process still universally used throughout Bolivia.

The following is the quaint language in which he describes his discovery: "In the year 1609, residing in Tarabuco, a town in the province of Charcas, eight leagues from the City of Silver (Potosí),

wanting to experiment with a method of fixing quicksilver in a dish or vessel of iron, and not having such a vessel, I used one of copper. Not having the expected result, I put in several materials, and amongst others, ground silver-ore, with the notion that the mineral virtue of the stone, with the heat and humidity of the cooking, would be of importance for my undertaking. At the end of the experiment I extracted a small quantity of silver amalgam and silver, which at first much surprised me ; but my astonishment ceased when I found that the silver collected by the quicksilver had come from the ore, and not from the quicksilver which had in part been transmuted. I remained very content with this new and quick method of beneficiating ores; and from that day, by reflection and continuous experiments, I have much improved it, using it and allowing it to be publicly known without making a mystery about it or reserving for me alone this or other secrets, etc. No one taught it (the process) to me, and I know of no author, either ancient or modern, who has mentioned it. The Royal Court of La Plata has therefore granted me an exclusive right, so that no one, without my license, can use this method of beneficiating ores. With some reservations, which in the chapters that follow will become manifest, I have permitted every one to practice it without any profit to myself."

A somewhat imperfect translation of a part of Barba's work is to be found on page 656 of Percy's *Metallurgy—Silver and Gold.* It is there erroneously called the Cazo or Caldron process. In Chapter iv. of his book, Barba calls the vessels in which the operation is carried out "Fondos," and they still retain this name in Potosí and in all of Bolivia and Peru where the process is still practiced. Barba distinctly refers to the treatment of sulphuretted ores of silver, or "negrillos," and gives extensive and exact instruction as to the method of roasting them before treating in the fondo. This description is so minute, that it is of value even at this late date.

In Book iv. of his treatise, Barba also describes at length the smelting of ores in blast-furnaces as then carried out ; and it must be said that absolutely no improvement has been made in such smelting from that day to this, excepting only the increase in the size of furnace and the mechanical improvements for doing the work. The chemical principles, if not understood, were *practically* known at that date.

Returning again to the year 1759, we find that complete surveys were made at that time, and it was proposed that an adit should

be commenced at the expense of the government, in order to cut the veins of the Cerro de Potosí in depth, and to ventilate and drain the whole mountain. The plan, however, was never executed, as the tunnel proposed would have been 3000 yards long, and the expense and time required appeared prohibitory. There were in 1759 only 55 reduction-establishments at work; and the annual production did not exceed $2,500,000.

In 1778 a renewed attempt was made by Escovedo, then governor of Potosí, to reach the veins by means of a deep adit. The work was continued until 1790, when 1000 yards had been driven.

In 1790, the Baron von Nordenpflicht, a Saxon mining engineer, employed by the king of Spain, arrived in Potosí with a large staff of scientific men. He suspended the work commenced by Escovedo, and advised the continuance of the Royal Tunnel or "Real Socavon," which had been previously worked by a miner named Berrios. This adit runs almost parallel with the veins, from the north face towards the center of the mountain, 2250 feet below the summit.

In 1794 the last Governor of Potosí, Francisco de Paula Senz, described the mines as in a very bad condition, all shafts filling with water when they had reached the depth of 80 yards.

In 1799 the Miners' Corporation again addressed the king of Spain. There were only 35 reduction-establishments at work, and 7970 Indians, 2943 slaves and 5027 free workmen were employed. There were still pacos to be had in the middle of the mountain, if the mines were made accessible; but to reach the negrillos and deeper parts, it would be absolutely necessary to finish the Real Socavon.

In 1807 Potosi was visited by Anthony Zacariah Helms, a director of the mines near Cracow, Poland. He refers to the mines as full of water, and suggests the employment of skilful men to erect machinery for the purpose of draining them to the bottom. He writes: "As soon as the water in the pits can be got out, the mines of Potosí will be in a more flourishing condition than ever." In speaking of the Guaillahuasi level, which he "had been able to inspect," he says: "A hundred years before it led to many rich veins of ruby and gray silver-ore, but as they had no proper machinery the pits filled with water." He adds: "We found on the conduit good gray and other silver-ores mixed with other materials, which, by proper management, might yield the proprietors a considerable quantity of metals; but it would be still more to their

advantage if they employed skilful men to erect machinery for the purpose of clearing the bottom of the mines from water."

In 1809 the War of Independence broke out, and Potosí became for many years one of the principal centers of the struggle; it was repeatedly taken and pillaged. This was the end of successful mining in the Cerro de Potosí, until recent times.

From 1800 to 1809 the product had been $32,000,000, or over $3,000,000 per annum.

In 1825 an English company sent out a vessel loaded with machinery to open the mines of Potosí; but the panic of 1826 coming on, the machinery was abandoned on the coast and never reached Potosí. Since that day a number of local companies have attempted to work the mines, but none of them with sufficient capital to reopen the old workings. Mining was still carried on in a desultory fashion near the surface in small veins overlooked by the ancients; but the great veins which had formerly made the mountain famous remained unopened until two years ago, when the writer, after spending, as before said, over half a million dollars, succeeded in unwatering the Cotamitos mine, and encountered a pay-chute averaging 60 ounces from wall to wall of the vein.

The Royal Tunnel or Real Socavon, also above referred to, has been, under the writer's management, completely cleaned out, rails have been laid in the tunnel and compressed air-drills introduced. Several long cross-cuts to the east have been run; and the last reports are that rich ore has been discovered in one of these cross-cuts, selected specimens assaying 700 ounces per ton. This tunnel is, at an average, 500 feet below the lowest point reached by the old miners; they having penetrated to a maximum depth of 1700 feet from the croppings with absolutely no other means of extracting water and ores than on the backs of their Indian slaves.

A more detailed account of the results of the last few years' work in Potosí will be given below.

Geology.

The "Cerro Rico de Potosí," or Rich Mountain of Potosí, rises 16,000 feet above sea-level, in latitude 19 degrees, 22 minutes south; longitude, 65 degrees, 32 minutes west, and about three hundred miles east of Cobija, on the Pacific Ocean. It is part of the mountain range that forms the eastern boundary of the great table-land of Bolivia, the average altitude of the latter being 12,500 feet. The waters descending the sides of the mountain finally enter

the affluents of the Rio de la Plata, reaching the sea at Buenos Ayres. It is therefore located on the eastern watershed, although now most readily approached from the west coast.

Leaving the dioritic eruptive rocks which form the immediate shore of the Pacific, the geologist at once encounters Jurassic rocks, which extend to the summit of the first or western range of the Cordillera. This is composed entirely of modern eruptive rocks, having an average width of 100 miles, and containing many enormous active volcanoes reaching a height of over 20,000 feet. Passing through the defiles of the volcanoes the great table-land of Bolivia is reached, which is composed almost entirely of Jurassic and Cretaceous shales and sandstones, with a few beds of limestone and occasional local Carboniferous measures. All over the great plateau, vast masses of what the writer has determined, microscopically, to be dacite are encountered. It has been described as trachyte by David Forbes. These great masses have evidently been ejected from long fissures. They have a uniformly level surface, and are often covered with glistening sand, the product of their decomposition. At the eastern edge of the plateau many and various eruptive rocks are met with, but principally andesites and rhyolites. Still further to the east Silurian slates are encountered. It has been shown by David Forbes, who examined the northern part of Bolivia, that the great ice-capped range of mountains extending from Illimani to Llampo, the enormous glaciers of which feed the headwaters of the Amazon, are of Silurian age: Forbes failed to find eruptive rocks in these great mountains, among the highest in the world. The eruptive rocks in northern Bolivia are not encountered till the western flanks of the mountains or the table-land are reached. Further south, however, in the district examined by the writer, in the latitude of Sucre,* the capital of Bolivia, the backbone of the continent is composed of great peaks of eruptive rocks 18,000 to 20,000 feet high, and the Silurian slates are generally confined to the east flank of these mountains. The water from all these great peaks enters the Rio de la Plata; while the water from the great range of enormous glaciers in the northern part of Bolivia enters the Amazon river. It is in the eruptive rocks which penetrate the table-land of Bolivia that all the valuable silver-mines are situated.

Some twenty specimens of the eruptive rocks of Bolivia have been examined under the microscope by Mr. James P. Iddings, of

* Also called Chuquisaca and La Plata.

the United States Geological Survey, and also by Mr. F. J. H. Merrill. The writer can do no better than quote from the report of Mr. Iddings:

"The twenty-one thin sections appear to come from a closely-related series of eruptive rocks. Nearly all of them are characterized by porphyritic crystals of quartz, feldspar and biotite, with comparatively large microscopic crystals of dusted apatite. In some of the sections, there is much sanidine, or orthoclase, but in most of them it is not so abundant as the plagioclase. In several sections, quartz occurs in small quantities; and in one thin section it is entirely absent. This section carries considerable hornblende. Mineralogically, therefore, they range from quartz-orthoclase-biotite rocks, with some plagioclase, to quartz-plagioclase-biotite rocks, and in one instance to a hornblende-biotite-plagioclase rock. This mineralogical variation might take place in a single body of rock. In such a case, those variations, which form but a small portion of the whole, would be called mineralogical facies of the predominant rock, etc."

With due allowance for the exact location of the specimens in the field and their relative occurrence, a series of results were obtained, which are recorded in the geological map accompanying this paper.

All the principal mining camps now worked in Bolivia, appear to be in either rhyolite or dacite, generally the latter. Such is the case with the great Huanchaca mine, and also with the no less remarkable mines of Colquechaca; while the Oruro mines are also probably in dacite, although some of the specimens seem to leave a doubt, and possibly there has also been an eruption of rhyolite at Oruro.

With reference to the Potosí district, the examinations have been so conclusive that there is no room for doubt as to the proper name of the eruptive rocks. It is a peculiarity of all the eruptive rocks of Bolivia, that, almost invariably, they contain angular fragments of the sedimentary rocks through which they have forced their way; and it is a further peculiarity that near the line of contact of eruptive and sedimentary rocks these fragments increase in frequency, until in places a veritable conglomerate of the eruptive and the sedimentary rocks has been formed. Precisely to what cause this phenomenon can be traced, the writer is at a loss to say, unless it be that the sedimentary rocks of Bolivia, at the time of these eruptions, were still in a comparatively soft and friable condition.

On the accompanying map of the Cerro de Potosí, the sedimentary rocks have been marked, as definitely determined by fossils found in all of them, with the single exception of the Jurassic formation. Forbes claims to have proved these rocks to be Jurassic, and he has so marked them in his geological map of Bolivia, and they certainly are an exact counterpart of the Jurassic rocks further to the north. The Silurian rocks do not appear on the maps of the Cerro, but are encountered 20 miles to the east of the mountain, at or near the town of Keysa.

To Mr. Frederick A. Canfield belongs the honor of finding the Tertiary fossils upon the Cerro de Potosí. They had been overlooked by all previous engineers, even by scientific men, like the Messrs. Francke, who have been familiar with the Cerro for many years. The description of Tertiary fossils from the Cerro, published by H. Engelhart, on page 36 of *Abhandlungen der Naturwissenschaftlichen Gesellschaft Isis*, in Dresden, 1887, is erroneous in giving the honor of this discovery to Mr. Francke. The writer gave these fragments of fossils to that gentleman. A large collection of these fossil plants is now in the hands of Prof. N. L. Britton, who will describe them at length in a separate paper. Prof. Britton writes me as follows about them:

"The collection comprises some 200 specimens of leaves, fruits and fragments, representing about 25 different species. All appear to belong to living genera and probably some of them to living species existing in other parts of the Andes at the present day. All these facts point to the deposit being of recent Tertiary age. Among the genera represented are *Cassia*, *Amicia* and *Swectia* of the Leguminosæ; *Lomatia* of the Proteaceæ, and *Dodonæa* of the Sapidaceæ."

These fossils are of great importance, as proving that the eruption not only of the Cerro de Potosí, but of all of the eruptive rocks of this part of South America, has been *post*-Tertiary, not post-Jurassic and pre-Tertiary, as supposed by Forbes and d'Orbigny, who have studied the geology of Bolivia. The fossils that characterize the upper Cretaceous rocks were discovered by the writer, and the remarkable Silurian or Devonian fossils, described and illustrated in the annexed very complete paper by Prof. R. P. Whitfield, came from the range of rocks, 25 miles to the east of the Cerro de Potosí, which range extends along the whole flank of the Cordillera. The orthocera were found near the town of Quechisla, on the flanks of the great mountain of Chirolque, close to where the bismuth-mines

are situated, and the brachiopod near the road to the town of Sucre. From the structure of the stratified deposits around Potosí, it is evident that the mountains of Chalviri, composed entirely of one vast eruption of andesite, lifted the measures and left them in their present condition. Subsequently, the rhyolite of the Cerro de Potosí cut its way through and across these measures, and in so doing almost destroyed the fine-grained Tertiary sandstone which adjoins the rhyolitic eruption. According to Prof. J. S. Newberry, whose able help in the preparation of the geological map I take pleasure in acknowledging, these Tertiary measures are undoubtedly a small local basin. I have found no similar measures anywhere in Bolivia. Subsequent to this eruption, the great mass of conglomerate material at the contact of the eruptive and sedimentary rock, must have had time to become completely hardened and form a solid rock before the fissures were opened which were subsequently filled by the vein-matter. These fissures have a general north and south strike, and seem to radiate from a point down in the center of the mountain, almost like the veins of a palm-leaf fan. No doubt, they have a common origin. They cut indiscriminately through the rhyolite, the Tertiary sandstone, whether lying flat or at a steep angle, the rhyolitic conglomerate, and the Jurassic shales and sandstone. They form true fissure-veins, in every sense of the word, and sometimes have a banded structure. The limit of the fissures to the north seems to be the edge of the glacial drift; to the south, their boundary seems to be on the south flank of the principal rhyolite mountain or Cerro de Potosí. They do not extend into the southern peak, which is a portion of the same rhyolite eruption, and is in part composed of rhyolitic ash. The mountains of Chalviri and Kari Kari, composed, as above stated, entirely of andesite, average from 1500 to 2000 feet higher than the Cerro de Potosí, and were formerly the home of some enormous glaciers which formed the magnificent moraines shown on the geological map. It is an open question in the mind of the writer, to what cause can be ascribed the disappearance of these glaciers, whether to a general lowering of the elevated table-land, or to a climatic change. In the north of Bolivia the great glaciers of Illimani and Llampo begin at an elevation of about 18,000 feet, the mountain range itself having an average elevation of perhaps 23,000 feet, and the peaks rising to much greater height. The Chalviri mountains have a present maximum elevation of about 18,000 feet; but the ancient glaciers reached down to about 15,000 feet above sea-level. I think it very likely that the disappearance of the

glaciers must be traced to a complete change of climate. It is known that the great plateau of Bolivia formerly held an enormous inland sea, many hundred miles in length and extending south as far as the mountains of Lipes, and north past the laguna of Poopo to Lake Titicaca. On the wagon-road from Huanchaca to the Pacific, the sides of the mountains still show the lines of the old lake, and a fresh-water lime deposit, in the former bed of the lake, now the bottom of the great plain, gives further evidence of the vast extent and depth of this inland sea. Can it be that to the disappearance of this great sea may be traced the change in climate and disappearance of the glaciers?

The great moraines near Potosí are such a striking feature in the landscape that attention is at once called to them; but a close examination of the table-land of Bolivia proves the recurrence all over the southern half of Bolivia, the part examined by the writer, of large moraines, proving the existence of a glacial period for all this part of South America. Large moraines are found 100 miles south of Potosí in the Huanchaca mountains, and they are also reported 100 miles north of Potosí, near Colquechaca.

The evidence of the Tertiary fossils proves that at one time the climate of this part of South America was about the same as that of British Guiana. Now snow is more frequent than rain, while at no very distant time great glaciers covered the country. It is a noteworthy fact that, in the exceptional climate of Bolivia, the snowfalls occur in summer, which is the rainy, or, perhaps more correctly speaking, snowy season. In winter the air is so dry that only an occasional passing storm precipitates any moisture.

To this almost entire absence of moisture is due the bleak and sterile aspect of the mountains. Not a tree is to be seen in any direction; rocks and glistening sand meet the eye of the weary traveller, and only in the narrow valleys, where irrigation moistens the soil, can vegetation be found. Even there the vegetation is of the sparsest description, and the vegetables of the temperate zone do not thrive. About the only grain which will grow is barley, and this crop is sometimes a failure. The price of barley, including the straw, which is fed to the mules and other stock, is generally $20 per ton, sometimes much higher. The mining towns of the great table-land, therefore, have to depend almost entirely on food brought either from the valleys to the east, or from the west coast.

The Cerro de Potosí, seen from the city on the north, forms an almost perfect cone, the upper part of a reddish-brown color, the

lower part gravish or bluish. It would draw the eye in any landscape, but in Bolivia, encompassed as it is by legendary lore, it has always formed one of the great attractions of the country and of the old city at its base. The view published herewith is from a photograph taken by the author.

The greatest vein of the mountain, known as "Mendieta" or "Veta Rica," crosses the summit or "punto" of the mountain, and, curiously enough, the thermal action has there so changed the rhyolite that it is a practically pure quartz, containing over 95 per cent. of silica. The crystals of feldspar have been dissolved, and have left only hollow spaces or pseudomorphs in the mass of quartz. It is probably to the hard "punto" of the mountain that the pyramidal shape is due. The sides, composed of softer shales, have worn away, both by the action of water and of ice, until the present shape was reached. On the north side of the mountain this action is very plain. It is on that side that the flow of ice from the Chalviri glaciers impinged against the Cerro, which seems to have broken the river of ice into two great flows, one to the east and the other to the west of the mountain.

Vein-Structure.

The developments made in the last few years, and especially the accurate surveys made by Mr. F. A. Canfield, have laid bare the whole vein-structure of the mountain and have upset many preconceived notions. There has always been an idea in Potosí that the veins of the mountains were of enormous length, extending from one side to the other. When a vein was lost it was supposed to have been faulted; and as recently as twenty years ago good engineers, and notably Hugo Reck, who made a thorough study of the mountain at that time, laid down on their maps great lines of faults, especially where the Mendieta, the principal vein of the mountain, crosses the summit of the Cerro.

Such faults do not, as a matter of fact, exist; nor do any long veins exist; and to this fact I attribute the confusion of names on the Cerro. The truth is that the Cerro is crossed from north to south not by single veins, but by vein systems, or what the Germans call "Gangzüge." Dr. Albrecht von Groddeck, in his book, *Die Lehre von den Lagerstätten der Erze*, classes the veins of Potosí as belonging to the same type as those of Kremnitz, and this characterization is perfectly correct. What is known as the Mendieta, or Veta Rica, in Potosí is not one vein; but these two fissures and the veins Cota-

mitos, Flamencos and Guaillahuasi form a true vein-series or "Gangzug," and cannot, therefore, truly be called one vein. The same holds good of the Veta Tajo-polo, which is made up of at least four or five principal smaller veins and innumerable branches. It is a general rule in Potosí that on or about the termination of one vein a parallel vein comes in, and this holds good very generally for "Gangzüge." It is this feature, no doubt, that led Reck and other miners to suppose that there were great faults. Some faults do exist in the mountain; they have a general strike of northeast and southwest, dip slightly to the north and are very plainly marked, but they never throw the veins more than four or five feet, and are no serious impediment to mining.

Great claims have been made by the ancients as to the width of the veins in the mountain of Potosí, and if some of the excavations at the top of the mountain are to be taken as constituting one vein, the width would indeed be very great—in some instances over 100 feet. But a close examination of these excavations and of the vein-structure in general proves a very different state of affairs. As is often the case in fissure-veins, the veins of the mountain of Potosí near their surface are very much split up. The waters carrying the silver-salts in solution evidently did not issue through one well-defined fissure, but through a mass of shattered rock. The foot-wall is fairly well defined, as a general rule, but there is no precise limit of the ore towards the hanging-wall, or any real hanging-wall. Instead of a single strong fissure with well-defined walls, the veins practically constitute a "stockwerk," and such ore-bodies or stockwerks were the basis of the former great production of silver from the surface, or decomposed ores of the Cerro. Near the top of the mountain, in one place, is a depression fully 100 feet wide, which is said to be the result of a great cave in the heart of the mountain, by which many miners are supposed to have lost their lives. Trustworthy eye-witnesses who, forty or fifty years ago, entered some of the older workings, now inaccessible, report great openings in the heart of the mountain, many hundreds of feet high and long, and 40 or 50 feet wide. In one of them there is said to be a lake of fresh water. Although all the reports in an old Spanish camp must be taken *cum grano salis*, the vein-structure proves that stockwerks were the great reliance of the ancient miners. With their slave-labor they could afford to take out large masses of low-grade ore and, by careful assorting, select the small part that was rich enough to work.

CITY AND MOUNTAIN OF POTOSI, FROM THE CATHEDRAL TOWER

The sides of the mountain of Potosí are covered from top to bottom, with enormous accumulations of "desmontes," or what we call the "rock-dump"—the waste from the mines. These "desmontes" have evidently all been broken to an almost uniform size in selecting the ore therefrom. Even to this day, Indian women ascend the mountain and cull the "desmontes," selecting small pieces of ore, generally not larger than a ten-cent piece, that have an assay-value of perhaps 15 to 20 ounces per ton. The women may, with great diligence, pick out in a day's work 100 pounds of such ore, which shows how very thoroughly the ancient miners understood the process of ore-sorting, and incidentally, also, how very cheap and plenty their labor must have been. It is generally supposed that the cost of Indian slave-labor did not exceed 10 cents per man per day, while the amount of work done by an Indian slave was certainly in excess of that done by the Indians of to-day, and the silver produced was considerably more valuable than at present. It is therefore not to be wondered at that the ancient miners could profitably work comparatively low grades of ore, and that only the great depth attained, to wit, over 1500 feet, and the influx of water finally put a stop to their operations.

In depth the great veins of the mountain have, in the judgment of the writer, remained just about the same as they were on the surface, with this notable difference, that whereas on the surface the various ramifications formed a stockwerk which could be worked at a profit, such a condition of things has ceased to exist in the harder and denser rock 2000 feet below the top of the mountain. The fissures still continue at the level of the Royal Tunnel, but only the strong veins, that is, those of sufficient width to be worked by themselves, can be made profitable. Furthermore, the ores being "negrillos" or dense sulphurets of iron, are more expensive to mine and reduce.

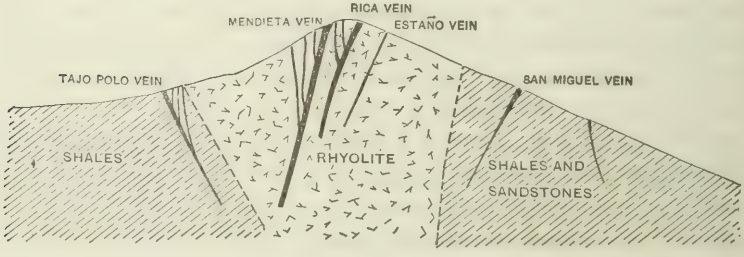
From the diagram herewith given, showing the cross-section of the mountain, it will be seen that the Mendieta vein series dips toward the east, while the Tajo-polo vein series dips toward the west, and at no great depth below the Real Socavon they ought to intersect each other and perhaps form a single strong vein. At any rate, the intersection of these veins must at some future time be the objective point of the mining, as it is the region most likely to be valuable.

The strongest and widest veins of the mountain, *i.e.*, those of the Guaillahuasi and Flamencos, lying to the south of the "punto" or

top of the mountain, have not to date been reached by the Real Socavon, and these two may also join in depth and form a single strong vein. The last owners of Guaillahuasi are said to have extracted $400,000,000.

Character of the Ores.

As is usual in silver-mines, the surface-ores or pacos of the Cerro de Potosí were wonderfully rich and composed principally of chlorides mixed with native silver. They were also much more siliceous than the ores in depth—precisely why, it is difficult to say. On the flanks of the mountain, but especially on the north and west sides, are found conglomerates in which the cementing material is oxide of iron, evidently derived from a flow of water from the top of the mountain. These conglomerates conform to the general inclination of the surface and are of very recent



Cross-Section from East to West through the Cerro de Potosí.

origin; and it seems likely that they were derived by leaching from the veins at the top of the mountain. By some means or other the iron in the surface-ores or "pacos" has been dissolved and re-precipitated in these beds at the foot of the mountain. As we penetrate deeper into the mountain, or follow a vein from the surface-cropping down, the pacos commence to contain an appreciable quantity of oxide of iron and also a little sulphur. The ores are then called "mulattos." An analysis of a sample of such ores mined from a small vein, a branch of the San Miguel vein, is as follows:

	Per cent.
Silica,	23.90
Iron,	18.11
Sulphur,	3.99

Penetrating still deeper into the heart of the mountain, or further

away from the action of surface waters, the true sulphuret-ores or "negrillos" are eventually reached. In the mountain of Potosí the negrillos are principally compact, hard, iron pyrites containing a small percentage of copper pyrites, and in some of the veins some zincblende and a very little galena. The silver occurs as tetrahedrite, containing about 700 ounces of silver per ton.

An almost constant accompaniment of the silver-ores of Potosí, and of a great many of the silver-ores of the plateau of Bolivia, is binoxide of tin, in the shape of gray or yellow oxide. Some of the silver-veins are very rich in oxide of tin, notably so the Tajo-polo and the Veta Estaño, which latter was, as its name implies, named after its contents of tin. Formerly, the old miners left the tin standing in the veins; but of late years a profitable tin-industry has been based upon the occurrence of these ores. The following average analysis of the pay-chute in the Cotamitos mine gives a fair indication of the character of the ores:

	Per cent.
Silica,	17.90
Copper,	2.51
Tin,	3.52
Sulphur,	31.83
Iron,	44.64

The average assay in silver was, in round numbers, 60 ounces per ton. There is also a trace of gold in the ore.

In the level of the Royal Tunnel, 2000 feet below the top of the mountain, a number of the veins known on the surface can be examined. The Cotamitos vein is the only one which up to date has yielded rich ore. It is a noteworthy fact that this is the only vein at the bottom of the mountain which is in the rhyolite. The other veins examined leave the rhyolite and enter the shales or sandstones of the mountain and are all poor. All the rhyolite at the level of the Real Socavon contains small crystals of iron pyrites, and it all shows the decomposing effect of thermal water, in its structure. It seems as if the whole mass of the mountain of Potosí had been subjected to thermal action, and it is difficult to find any mineral in the mountain that does not contain a trace of silver. The hard quartz rock at the top of the mountain all contains a few ounces to the ton; generally, however, below three ounces. The pyrites crystals disseminated through the rhyolite at the level of the Real Socavon generally assay one-quarter ounce to the ton. A cross-cut driven fifty or one hundred feet anywhere

in the mountain will be sure to cut a number of veins more or less wide. Any of these veins, if followed a sufficient distance, will connect with others that will as surely lead to some of the pay-chutes or bonanzas of the mountain. This great multiplicity of veins, branches and spurs is the bane of the miner, for it is extremely difficult to determine where and when to stop the work of prospecting. The old miners had abandoned the Cotamitos mine, reopened by the writer, (and in which, at last reports, a large body of 60-ounce ore was developed), after driving a cross-cut at a certain point to within 3 feet of the vein they were looking for. They stopped in a parallel vein of iron pyrites of considerable width but extreme poverty. Exact surveys led to the belief that the vein sought for lay somewhat further east from the face of this cross-cut, and after driving only 3 feet, it was found. This, of course, was more good luck than management, but it is a striking example of the difficulties of mining such "Gangzüge" as exist in Potosí.

Modern Age of the Veins.

We have seen that the rhyolitic eruption of the Cerro de Potosí was of the post-Tertiary age; that subsequent to this eruption many centuries must have elapsed to consolidate and harden the conglomerated material at the contact of the rhyolite, shales and sandstone; that at some subsequent time fissures rent the mountain, which at a still later age became the scene of thermal activity and the deposition of the silver-ores and other mineral matter in the veins. These facts in themselves would make the age of the veins of the mountain of Potosí very modern, but the author is tempted to believe, from a number of facts that have come to his notice in the study of the geology of Potosí and vicinity, that these veins were still in course of deposition during the glacial period of Bolivia. The study of silver-mines in various quarters of the globe made by the writer has often impressed him with the belief that most of them are much more modern than has been generally supposed, and that the deposition of ore in a vein is a comparatively rapid process if the proper conditions exist. The author would not be surprised if, on further careful study, it should be determined that the age of many silver-deposits of the North and South American continents is as modern as the advent of man.

PRODUCTION OF THE CERRO DE POTOSI SINCE ITS DISCOVERY.

Many and conflicting are the reports of the annual and total production of Potosí, as the following tables, from what the author considers the least objectionable sources, abundantly show.

From the Year.	To the Year.	Annual Product.	Royal Tax.	Principal.	Author.
1545	1556	$7,534,000	$16,575,000	82,875,000	Humboldt.
1556	1578	445,500	1,960,000	9,800,000	Humboldt.
1579	1736	4,100,000	129,400,000	647,000,000	Humboldt.
1736	1789		14,500,000	72,700,000	Humboldt.

From the Year.	To the Year.	Annual Product.	Royal Tax.	Principal.	Author.
1545	1591	$39,000,000	$360,000,000	1,800,000,000	Juan Diaz de Lupidana.*
1545	1603	51,000,000	59,600,000	2,980,000,000	Pedro de Lodana.†
1556	1800	3,170,000	152,000,000	760,000,000	Lamberto de Sierra.‡
1790	1800	2,000,000	4,000,000	20,000,000	Lamberto de Sierra.

Besides the foregoing estimates there are many others, some of them claiming a production far beyond the figures quoted. Thus, Bartolomé Astete de Ulloa claims a production up to 1632 of $4,900,000,000. Bernardo de Vega, up to 1597, claims a production of $1,780,000,000. Many authors claim a product of more than $1,000,000,000 up to a century ago.

The writer doubts very much whether any such production actually took place; he thinks it more than likely that in the first ten or twenty years after the discovery of the mountain $10,000,000 per annum may have been actually produced; but it must be remembered that at that time silver was worth vastly more than to-day, so that the production in ounces probably did not exceed for the

* In his examination of the Royal accounts; also confirmed by Bernardo de la Vega.

† In his examination of the Royal Treasury made by order of the King in 1603.

‡ Minister of the Royal Treasury and Chief Accountant of the Tribunal of Accounts of the Vireinato.

first century of the existence of the Cerro 5,000,000 ounces a year. A century ago the production certainly fell to a value of $2,000,000, or perhaps 1,500,000, or 1,250,000 ounces per annum. From about the beginning of this century, or the date of the War of Independence, it is doubtful whether the production per annum has averaged 300,000 ounces.

The fact, however, remains, that the Cerro de Potosí, even upon the basis of the lowest estimates, has been the largest producer of silver of all districts in the world; and there is very little doubt that the aggregate product has exceeded 1,000,000,000 ounces.

The product of the mountain of Potosí at the present time is about 400,000 ounces per annum; but it must be remembered that only one of the great old veins of the mountain is being worked in depth. The principal ones are still unwatered, and may yet at some future time add considerably to the increase in production.

Population of the City of Potosí.

A counterpart to the largely-varying estimates of the production of the Cerro is found in the great differences in the estimates of the population of the city of Potosí. Some writers have placed it as high as 170,000. Early in the seventeenth century it was said to have 12 banking-establishments, 72 warehouses, 140 large shops, 260 retail shops, 212 market-places and 132 mills (ingenios) worked by water-power; also 48 mills worked by hand-power and 30 amalgamation-works. About the time of the breaking out of the War of Independence in 1809, the population of the city was estimated at 60,000. In 1846 it was estimated at 40,000, and although at the present time some estimates place the number of inhabitants as high as 15,000, they probably do not exceed 10,000.

The ruins of the city surrounding that part now inhabited, would not permit the residence of any such number as are supposed to have lived at one time in Potosí. It is quite possible that during the time of the slavery of the Indians, a population of 60,000 or 70,000 people was employed in or about the mines, but they were probably distributed over a great area, and not, properly speaking, residents of the city. Judging from the existing ruins, the town could not have contained at any time more than 40,000 people.

Lakes or Lagunas.

Mention has already been made of a system of lakes or dams built by the Spaniards at the cost of some millions of dollars to

impound water for use in amalgamating-establishments. The names of these ponds or lagunas are as follows: 1. San Ildefonso; 2. Crusisa; 3. Chalviri; 4. San José; 5. San Sebastian; 6. Lobato; 7. Ulistia; 8. Pisco-cocha; 9. Huacani; 10. Mazuni; 11. Santa Lucia; 12. Candelaria; 13. Atocha; 14. Muniza; 15. San Fernando; 16. Illimani; 17. San Pablo; 18. San Buenaventura; 19. Llama-mica; 20. Huaca; 21. Providencia; 22. Planila.

Of these, some are of great extent; thus, Laguna Chalviri, shown on the geological map, is several miles in length and a mile in width. These artificial lakes are located in the beds and at the foot of the former great glaciers. The water from them has been carried by a race to near the mouth of the Royal Tunnel, where a fall of 460 feet was obtained, and the power is there utilized by turbines for operating the mill and air-compressor.

Many of the dams are now in bad condition, so that the average flow of water which can be impounded in the rainy season does not exceed 5 cubic feet per second. Formerly, no doubt, double that amount was at the disposal of the miners of Potosí.

Cost of Mining.

The cost of mining ore in Potosí varies so greatly that it is impossible to fix even an approximate rate per ton. Owing to the peculiar nature of the veins, the amount of dead-work that should be carried on is extremely large. In general, at least from one-half to two-thirds of the men ought to be employed in dead-work. Although the rate of wages is not high, the natives receiving only from two to three Bolivians, or, at the present rates of exchange, from $1.50 to $2.00 in gold per day, the amount of work that these natives do is so small, and their work so irregular, by reason of the numerous feast-days or "fiestas," that the actual cost of dead-work is very heavy. Furthermore, the cost of all materials, and notably of dynamite, is very great, owing to the enormous freight-charges from England. When the writer introduced machine-drills in Potosí, he found that there was practically no saving in the cost of labor, owing to the higher price of the labor employed; but of course the speed of driving was very much increased. Thus, for example, the cost of driving the Royal Tunnel in the rhyolite of the mountain averages about one hundred Bolivians, or, say, $60 per meter, about half that cost being for the materials, and the other half for labor. The rate of progress with two machine-drills in this tunnel was not far from one yard for each working day of twenty-four

hours ; but it must be borne in mind that in Bolivia you cannot count on over twenty days of work per month, so that necessarily progress is slow. The rate in the hard iron pyrites of the veins of the mountain is very much slower, rarely exceeding seven or eight meters per month with two shifts of workmen. In 1885 in Potosí it was the custom of the native miners or "barrateros," to enter the mines and work continuously (or, at least, they claimed that they did work continuously) for thirty-six hours. During this time, and indeed at all times when they are awake, the natives chew the coca leaf and the supposed virtue of these leaves, that is, of the cocaine contained therein, is such as to permit them to work without lassitude for that length of time. The writer did not, however, find that the cocaine had any such effect. The rate of progress of men working in that manner in one of the drifts in the veins, never exceeded three or four meters a month. It was only after forcing the natives into the system of twelve-hour shifts that a greater speed was obtained.

As a further indication of the cost of the work, I give the cost of drifting in the shale of the mountain. A drift run with one machine-drill in charge of an American miner, one helper and two fillers, advanced about an average of a yard a day at a contract-price of $60 a meter. About half the cost of this drift was for materials. A large incline run in the rhyolite on double turn was paid for at the rate of 110 Bolivians per meter. Stations were paid for at the rate of 7 Bolivians per cubic meter. All materials were charged against the contract, as was generally done in all operations controlled by the writer. A great many of the mines of Bolivia still furnish materials to the natives, with all the consequent thefts and troubles involved in such a pernicious system.

Metallurgy.

Reference has been made before to the Fondo process, invented by Alonso de Barba. As generally used in Bolivia, the ores are first crushed in stamp-mills, or in arrastres. They are then roasted dead in adobe furnaces in charges of about 500 pounds. Such a furnace puts through four charges per day, and the fuel used is llama-dung, locally known as "ucha," or "tachia." When the ore is roasted practically dead, from 10 to 15 per cent. of salt is mixed with it, and it is almost immediately drawn out of the furnace and allowed to cool. From this chloridizing-roasting it is conveyed to the fondos. These are cast-copper kettles about 3 feet in diameter and 1 inch

thick, set in masonry with a fireplace below them, the tops being at such an elevation above the ground that a man can conveniently insert a wooden pole and stir the contents. The copper castings referred to are made all over Bolivia by the Indians, and they are not, strictly speaking, copper, but rather bronze, containing from 5 to 8 per cent. of tin. It may be said that to-day Bolivia is still in its bronze age; for it is only within the last year or two when more ready access has been given to Bolivia through the Antofagasta railroad, that iron castings have become cheaper than bronze. About 120 pounds of the roasted chloridized ore is put into the hot brine of one of the fondos, containing about 5 per cent. of salt. An Indian then commences to stir the mixture in the fondo, and after a certain time a small quantity of quicksilver is put in, the stirring being meanwhile continued. Not sufficient quicksilver is put in at once to amalgamate all the silver, the object being to produce "pella seca," or dry amalgam, that is, an amalgam containing about 15 per cent. of silver. Constant tests of the contents of the fondo are made in an equivalent of the horn-spoon of the Mexicans, here called "chua," a small saucer made of wood. A spoonful of the mass in the fondo is taken out and washed in this chua, and by the appearance of the quicksilver or amalgam the process is regulated. After a very short experience a scientific observer is able to tell whether or not copper is entering into the amalgam; and whether the ores are working too hot, or too cold. In fact, the ores in the fondo go through all the stages of the Patio process, with the exception that in the Patio weeks are consumed in doing what the fondo accomplishes in a few hours. As soon as all the quicksilver put into the fondo has been converted into dry amalgam, another small charge of quicksilver is put in; and these charges are repeated until finally the chua-test proves that all the silver is amalgamated. It is very easy to determine this by the presence of small globules of quicksilver which no longer change into amalgam. Generally a larger amount of quicksilver is then added to dissolve the dry amalgam, and immediately the whole contents of the fondo are dipped out with large wooden pails and poured into a large barrel of water. The barrel is kept in motion by hand, and a regular settling-process is gone through in the barrel. Finally, the clean amalgam and quicksilver remain at the bottom, and the amalgam is strained and retorted.

Such, in short, is the "Fondo" process, and the results obtained by such simple means are somewhat remarkable. The first striking

fact is that all classes of ore, not even excepting galena, are worked by this process, although the extraction in the latter, and in blende, is not as great as in ores of other classes.

A second striking fact is the great purity of the silver obtained from very base ores. The amalgam, after being strained, is retorted below a bronze bell, or "caperuza," and it is rare, indeed, that the resulting retorted silver, known as "Piña" silver, is discolored. Generally, it is of a pure white color, and above 900 fine in silver; often, it is 990 to 995 fine, the impurity being principally copper.

A further fact is that the ores are not roasted with salt, but that the salt is added after the roasting, and that consequently the chloridization must be very low; yet this seems to have no influence whatever on the yield in the fondo.

The chemical principles involved in the process are the same as those of the "Kroehnke" process, formerly so successful in Antofagasta, and of the "Tina" process, generally known as the "Francke-Tina" process in Bolivia, so named from the inventors, the Messrs. Francke.

In all these processes, subchloride of copper is the active chloridizing and dissolving agent of the silver-minerals. Kroehnke, in experiments published by him, shows that even the most rebellious silver, the black and red arsenites and antimonites, dissolve completely in a solution of subchloride of copper in the presence of a bar of zinc. This seems to indicate some electrical action. In the "Fondo" process, the oxidized ore (*i.e.*, the ore after it is roasted) is chloridized wet in the fondo by the action of a subchloride of copper, and at once amalgamated. In the Kroehnke process the same chemical reactions were carried out in Freiberg barrels, and the subchloride of copper was especially prepared in separate tanks, and introduced as required. Instead of adding only zinc, lead was very generally used. In the fondo process, it is usual in "hot" ores to add what is known as "material," or an amalgam of tin and quicksilver. These additions have, however, only the object of reducing or preventing the loss of quicksilver by solution.

The tina process of Messrs. Francke is practically the same thing as the fondo process, the only difference being the use of steam instead of direct heat below the fondo, for heating the liquor in the tina, and the application of steam-power to produce the necessary motion and mixing of the material in the machine. The tina, in fact, is a fondo on a large scale. The "Fondon," illustrated on page 662 of Percy's *Metallurgy*, is practi-

cally a tina operated by mule-power. The only real difference between the fondo, as now used in Bolivia, as illustrated in Percy and as invented by Messrs. Francke, is the capacity of the machine. The fondos have a capacity, in 24 hours, of, say, 8 to 10 charges of 120 pounds each, or perhaps one-half ton of ore. The "Fondon," illustrated in Percy, has a capacity of something over one ton per day, while the Francke tinas, when built of a diameter of 6 feet, treat 5 to 6 tons per day.

In the Kroehnke process, the charge of a barrel was generally 2500 kilogrammes in the Antofagasta works, and two or three charges were made per day in each barrel and no attempt was made to roast the ores, the principal ores treated by that process being the rich chlorides from Caracoles. An extraction of 92 to 96 per cent. was obtained from these very rich ores. The fondo process extracts from 80 to 85 per cent. of the silver in the very base ores common in Bolivia. The tina process, owing to the more thorough mechanical agitation of the ore, averages better.

The average cost, in various localities of Bolivia, of the fondo process, is from 120 to 180 Bolivians for a cajon of 5000 pounds. In Potosí the cost of the fondo process, with stamp-batteries run by water-power, is 150 Bolivians a cajon, or not quite $40 in gold a ton.

With a view to reduce the cost of the treatment in Potosí, the writer made a series of experiments, resulting in the building of the "Real Ingenio." An automatic ten-stamp battery was put in place of the commonly-used bronze stamps. Tinas took the place of fondos, and a rotary roasting-furnace the place of the small hand furnace. It was the original intention to stamp the ore raw, pass it through the rotary furnace, and then amalgamate it in tinas; in fact, to practically copy the American process. But this proved a failure, owing to the baseness of the ore. The ore passing through the rotary furnace came out practically raw, and it was found necessary to oxidize the ore before it entered the rotary furnace. With this object in view, the writer made some experiments which finally led to the building of roasting kilns in which the ores were roasted while still in lumps, and with no expense for fuel. The cost of this roasting is now only 40 cents per ton. The resulting changes in the ore are shown by the following two analyses, which are averages of several hundred tons:

Raw Ore Entering Kilns.

	Per cent.
Moisture, at 212° F.,	0.32
Silica,	13.50
Alumina,	3.04
Iron,	29.06
Lime,	0.32
Magnesia,	0.00
Sulphur,	32.35
Copper,	5.15
Carbonic acid,	0.00
Zinc,	0.63
Lead,	0.13
Arsenic,	1.30
Antimony,	1.45
Tin,	3.06
Gold, ounces troy, per ton,	0.045
Silver, " "	104.655

Kiln-roasted Ore.

	Per cent.
Moisture,	2.58
Silica,	19.05
Alumina,	2.40
Iron oxide,	48.40
Lime,	0.67
Magnesia,	0.00
Sulphur (total),	7.37
Copper,	5.13
Carbonic acid,	0.00
Manganese,	0.00
Zinc,	0.44
Lead,	0.16
Arsenic,	0.03
Antimony,	0.17
Tin,	7.30
Gold, ounces troy per ton,	0.045
Silver, " "	104.655

A sample of the kiln-roasted ore extracted with hot water proved the ore to contain the following amounts:

	Per cent.
Sulphuric acid (in compounds soluble in water),	2.49
Copper corresponding to this,	0.90
Iron " "	0.34

Perhaps the most remarkable feature about the kiln-roasting is the almost complete volatilization of antimony and arsenic. In the flues of one of the kilns fine crystals of artificial stibnite were found.

They contained by analysis 44.30 per cent. of antimony and 22.42 per cent. of sulphur. The antimony and arsenic evidently were volatilized as sulphides in the upper part of the kilns. Oxygen cannot reach the ore in that part of the furnace, but the heat of the burning ore below it was sufficiently intense to volatilize the sulphides of antimony and arsenic. The kiln-roasted ore does not clinker, and leaves the kilns in lumps of the same size as the ore charged. But the texture of the ore is completely changed, and it is very easily crushed and stamped. The only drawback to the process is found in the dust made in stamping. Owing to the slight percentage of sulphates the dust is very irritating to the membranes of the nose and throat. Wherever preliminary calcination and subsequent crushing are used a very complete apparatus must be employed for carrying off the dust.

Some of the Potosí ores were roasted in a pile by the writer; but while the reduction in sulphur was even greater than in the kiln-roasted ores, antimony and arsenic were not volatilized to the same extent.

An assay of a pile of roasted ores was as follows:

	Per cent.
Antimony,	0.60
Sulphur (total),	4.39
Sulphuric acid (in compounds soluble in hot water),	2.92
Copper corresponding to this,	0.58
Iron " "	0.47

When a second trial of the rotary furnace was made with kiln-roasted ore in place of raw ore, it proved a perfect success so far as thorough oxidation and chloridization went, but continued experiments since the writer left Potosí have, he is told, demonstrated the fact, which even during the presence of the writer was patent, that there was a large loss of silver by volatilization in this furnace. This loss, which was not in the dust, but by true volatilization, averaged from 10 to 15 per cent. of the total silver-content of the ore, and has finally led to the abandonment of the furnace and the adoption of three-floor reverberatory gas-furnaces, in which the loss by volatilization is kept down to 5 per cent. I judge that the cause of the large losses in the rotary furnace may be traced entirely to the presence of the salt mixed with the ore, a long series of experiments, just carried out, having proved the fact conclusively that these very base ores must be oxidized completely before salt is added and chloridization allowed to take place. It is precisely here that the

tina process shows itself to be valuable, for it is quite immaterial in the tina process whether the chloridization of the ores is 20 per cent. or 90 per cent., the extraction remains practically the same. This is so entirely different from the generally-conceived notion of silver-milling in the United States that too much emphasis cannot be laid on this point. Yet I am told that at Sombrerete, where base ores with 15 per cent. zinc and much pyrites are treated, recent experiments lead in the same direction. All mill-men in the United States who are working very base ores generally find it profitable to put bluestone into the pans. The action of the bluestone is undoubtedly to produce a subchloride of copper, and the consequent chloridizing action that takes place assists in the extraction of the silver. But the iron pan constantly destroys the subchloride, while the copper tina is as constantly regenerating it, and has, therefore, certainly some advantages for very base ores. Undoubtedly the very same result can be obtained by a sufficient addition of bluestone in an iron pan, but this may be more expensive. Experiments recently carried out with some very base South American ores by Mr. C. Rueger and the writer, at the Lexington Mill, at Butte, Montana, gave us extractions exceeding 90 per cent. on ores that were chloridized to 20 per cent. To accomplish this result in an iron pan, a large amount of bluestone was added to the charge.

The choice as between the tina and the iron-pan process must therefore depend upon the relative price of sulphate of copper, iron and copper castings. In a locality like Butte, for example, where an impure copper casting can be obtained from copper made on the spot, the choice would be in favor of the tina process as against the iron-pan and sulphate of copper process for very base ores.

The Francke tina process has been described at some length by Mr. Edgar P. Rathbone in August, 1884 (*Proceedings* of the Institution of Mechanical Engineers of Great Britain). Professor Egleston, describing the process in his *Metallurgy of Silver*, has copied this article.

The loss of quicksilver in the tina process is very small, although in Potosí at the present time all of the ores work very "hot," and there is a consequent chemical loss, or loss by solution. The consumption of quicksilver at Potosí is 4½ to 5½ ounces to the mark, or 8 ounces of fine silver. In the great Huanchaca mines the loss is considerably less, being about 2½ ounces to the mark of silver. At Guadaloupe the loss is a trifle over 3 ounces to the mark. The ores at present being treated at Potosí average 75 to 80 ounces per

ton, and the extraction is about 80 per cent. The bulk of the loss is in the roasting, the extraction of the roasted ore generally averaging 90 per cent. The tailings assay about 15 ounces to the ton, or say a loss of 10 ounces per ton of ore treated. The average fineness of the silver produced in Potosí is about 900. The oxide of tin present in all the ores of Potosí does not seem to interfere in any way in the roasting and subsequent amalgamation, but becomes troublesome in the settlers by reason of its great gravity. Where the fondo process is used in Potosí the tin settles in the great wash-tank, which is generally placed in front of the fondos. The Indians take the "relaves" or tailings from there and concentrate them by hand, making a good merchantable tin-ore.

An analysis of these concentrated tailings is as follows;

	Per cent.
Silica,	1.05
Binoxide of tin,	88.71
Ferric-oxide,	7.72
Alumina,	0.61
Copper,	0.90
Lead,	0.23
Sulphur,	0.45
Total,	99 67

These tin-concentrates are either shipped to England for sale, or they are smelted on the spot with rich tin-ores from the mine. They produce a rather impure bar-tin, but one containing considerable silver, which about makes up in value for the impurity.

The cost of treatment in the "Real Ingenio" erected by the writer at the mouth of the Royal Tunnel (which works, by the way, are not by any means perfect, but the best that could be erected with a lot of machinery sent out from England) has been reduced to 90 Bolivians a cajon; and, from late reports, it is about to be still further reduced by increasing the capacity of the plant. The cost formerly, as has been before stated, was 150 Bolivians a cajon. The saving is 60 Bolivians a cajon, or about $15.00 per ton. The works have a capacity of about 8 tons per day, but their cost has been enormous by reason of the great distance from railroads. Almost $100,000 was spent for freights alone on the machinery brought to Potosí, much of it costing 15 and 20 cents a pound for wagon freight alone. However, even at this great cost, the saving of about $30,000 per annum pays a large interest on the investment.

RESUMÉ.

The mines of the Cerro de Potosí, located anywhere within the borders of the United States, would be a center of great mining activity; and the same can be said of all the districts now operated on the table-land of Bolivia. The greatest drawback of the country is not its distance from the sea, but the abominable character of the labor. Any one who wishes to have a positive proof of the fallacy of the oft-repeated assertion that American labor cannot compete with cheap, or so-called "pauper labor," has only to go to Bolivia, or, in fact, to any country where labor is extremely cheap, and compare the product of a dollar's worth of labor in such a country with the same product in the United States. The observer will then be forced to admit that the high-priced labor of the United States is not really dear labor, but almost invariably cheaper than the so-called "foreign pauper labor." The future of the mountain will, in the judgment of the writer, depend entirely on the approach of railway facilities. These, and these only, will correct the evil of inferior labor; and without an influx of better labor from abroad, the mines of the Cerro de Potosí are not likely to occupy again a leading position in the world's production of silver.

APPENDIX.

NOTES ON SOME FOSSILS FROM BOLIVIA, COLLECTED BY MR. ARTHUR F. WENDT, AND DESCRIPTION OF A REMARKABLE NEW GENUS AND SPECIES OF BRACHIOPOD, BY R. P. WHITFIELD.

AMONG some Devonian fossils presented to the cabinet of Columbia College School of Mines, from near Sucre and Quechista, Bolivia, by Mr. Arthur F. Wendt, there are two forms which appear to be new to science. At least, after a thorough search through both the geological and zöological records, as well as much other literature, no mention of them can be found.

One of these, a Brachiopod, is especially interesting from its great size and peculiar form, and in not being readily classed under any existing genus, but showing characters, externally, which ally it to *Stringocephalus* (Defrance and Sandberg) and also to *Centronella* (Billings). The specimens are contained in a sandy ferruginous limestone and are associated with *Spirifera*, *Quichua* and *Terebratula Antisiensis* (D'Orb.), a *Schizodus* or *Paracyclas*-like bivalve

Fig. 1.

Fig. 3.

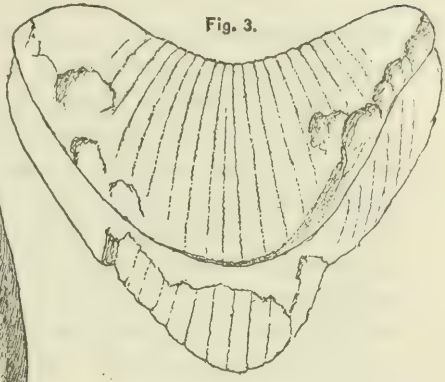


Fig. 2.

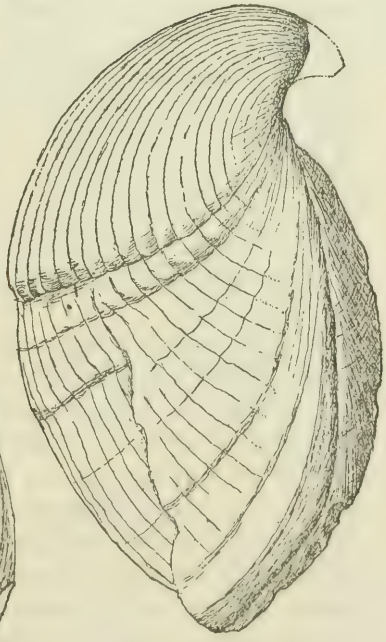
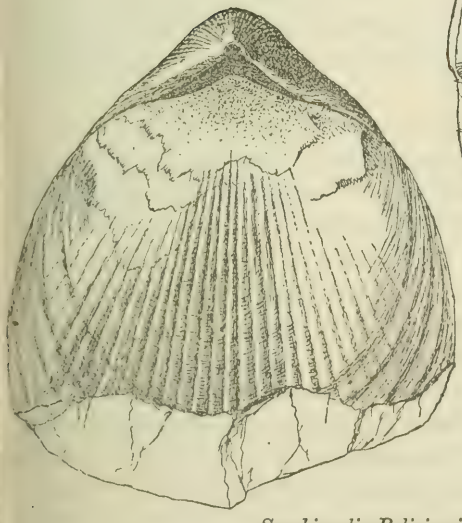


Fig. 4.



Scaphiocœlia Boliviensis, WHITFIELD.

FIG. 1.—Dorsal view of the most perfect specimen, the beak of which is much worn. FIG. 2.—Profile or side view of the same, showing the curvature of the ventral valve; the dotted line at the beak represents what would properly be its form. FIG. 3.—Front view showing the concavity of the dorsal valve. The specimen has been somewhat crushed. FIG. 4.—The dorsal view of the upper half of a specimen showing the beak more entire, but still worn.

and a *Pleurotomaria*, or *Straparollus*-like gasteropod, neither of these being in a condition to identify fully.

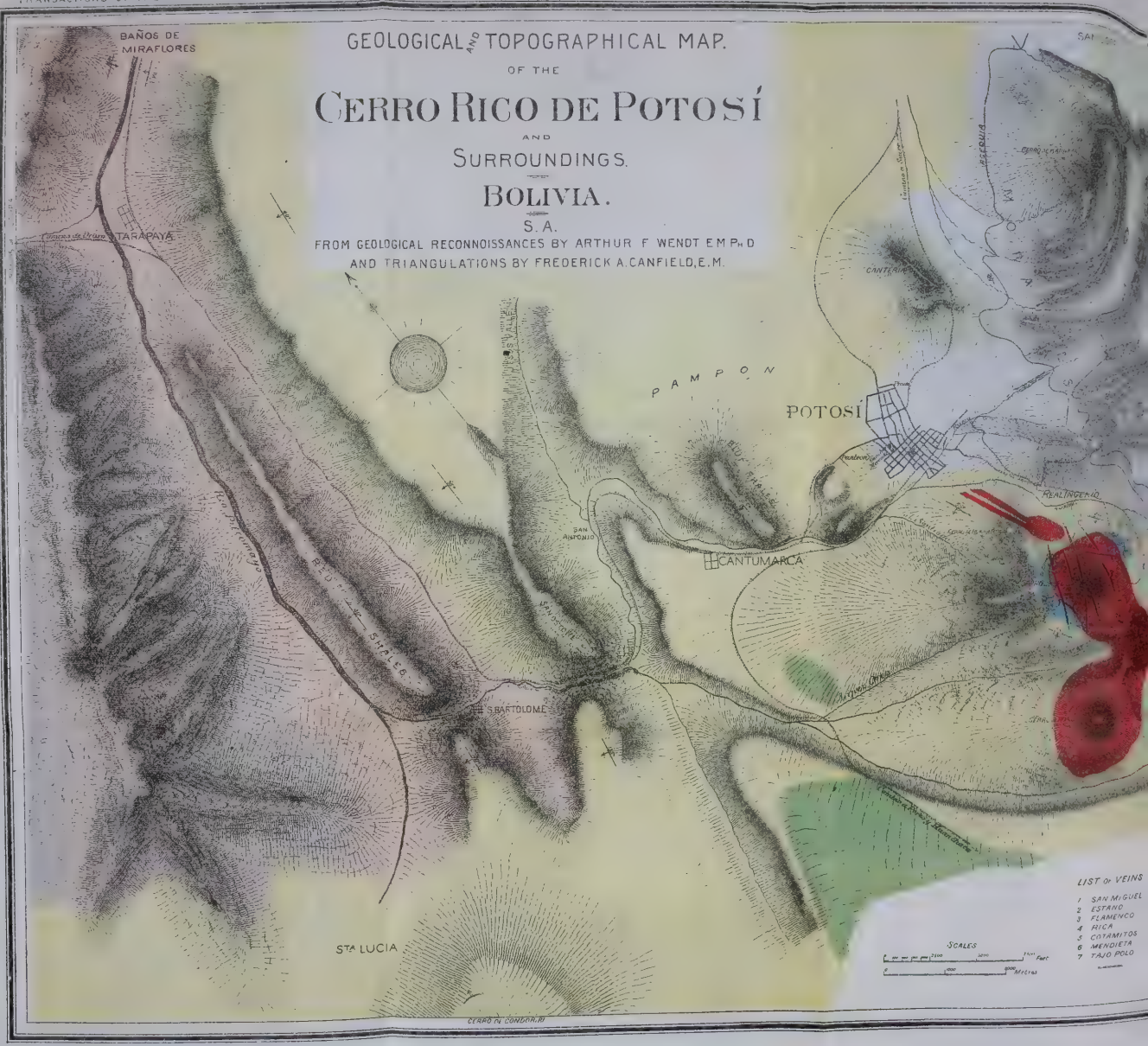
The second interesting form is an *Orthoceras*, which occurs in numbers in a limonitic shale at Quechista, Bolivia, and is probably of the same age as the Brachiopod above mentioned, but has no associated forms, so far as is seen upon any of the rock-fragments accompanying the shells. The associated *Spirifera Quichua* and *Terebratula* (*Rhynchonella*) *Antisiensis*, two well-authenticated Devonian forms, are sufficient evidence of the similar age of the accompanying new forms. From the impossibility of referring the new and very remarkable Brachiopod to any known genus, I am obliged to form a new genus for it under the name,

Scaphiocœlia,

From σκαφη, a boat, and κοιλια, abdomen, in reference to the form of the shell.

Generic Description.—A terebratuloid, brachiopodous shell, having a strongly convex ventral valve, and a longitudinally and angularly sulcated dorsal valve; both of which are strongly plicated. Internally, the ventral valve has a strong, deep, triangular byssal opening and muscular scar, and the dorsal has strong crural processes; but the loop or calcified appendages are unknown. Shell structure strongly fibrous, without any punctæ under a hand-magnifier.

The only species yet known of this genus reminds one somewhat, by its external form, of *Stringocephalus* (Sandberg), but it possesses no flattened cardinal area on the ventral valve. It bears, however, a much greater resemblance, both externally and internally, so far as the internal features of this one are known to the small shells of *Centronella* (Billings); but its strongly plicated exterior is a very marked feature, while none of that group yet known show any tendency whatever toward plications. The strongly fibrous structure of the shell is another distinction not easily reconciled with the compact lamellose or flaky, highly-punctate shell-substance prevailing in *Centronella*. Unfortunately, the specimens show nothing of the internal appendages and but little of the internal surface of the valves of muscular markings. On a single internal cast there are deep, wide cavities made by the removal of the crural processes, which are seen to have been bent toward the dorsal valve when the cast was broken and probably supported some loop-formed


GEOLOGICAL AND TOPOGRAPHICAL MAP.
OF THE
CERRO RICO DE POTOSÍ
AND
SURROUNDINGS.
BOLIVIA.
S. A.
FROM GEOLOGICAL RECONNOISSANCES BY ARTHUR F WENDT E.M. Ph.D
AND TRIANGULATIONS BY FREDERICK A. CANFIELD, E.M.
BAÑOS DE MIRAFLORES
TARAPAYA
PAMPON
POTOSÍ
CANTUMARCA
SAN ANTONIO
S. BARTOLOME
STA LUCIA
CERRO DE CONDARIRI
CANTARIAS
REALTAGENIO
RED SHALES
LIST of VEINS
1 SAN MIGUEL
2 ESTAÑO
3 FLAMENCO
4 RICA
5 COTAMITOS
6 MENDIETA
7 TAJO POLO
SCALES

SAN JOSÉ
POTOSÍ
REALINGERIO
CERRO DE CHALVIRI
CERRO DE CHALVIRI
LIST OF VEINS.
1 SAN MIGUEL
2 ESTAÑO
3 FLAMENCO
4 RICA
5 COTAMITOS
6 MENDIETA
7 TAJO POLO
Scales
Feet
Metres
EXPLANATION
STRATIFIED ROCKS
TERTIARY
CRETACEOUS
JURASSIC
GLACIAL DRIFT
STRIKE AND DIP OF ROCKS
ERUPTIVE ROCKS
ANDESITE
DACITE
RHYOLITE (NEVADAITE)
TRACHYTE
VEINS

appendage. Beyond this nothing is known of the interior. The following species is the only one known.

Scaphiocœlia Boliviensis, n. sp.—Shell very large: one individual, which is by no means the largest, but which shows the entire form, measures nearly three and a half-inches in length and almost two in width; ovate longitudinally in general outline along the junction of the valves, with an angularly ventricose and strongly arcuate ventral and a somewhat concave dorsal valve. Ventral valve deeply concave with a large, strongly incurved triangular beak which is marked by a large, broad and deeply excavated foraminal cavity beneath, but without apical perforation, at least in the adult stages; viewed in profile, this valve is strongly arcuate on the back, the lateral margins strongly lobed on the sides and the front margin deeply sinnate; the visceral portion of the valve deep, triangularly-trough-shaped. Dorsal valve gently arcuate longitudinally, but with the central portion rather strongly depressed from beak to base, the sides rising above it along the middle of its length to conform with the margin of the ventral valve. Surface of both valves marked by longitudinal ribs or plications, very strong along the central line, but indistinct or sub-obsolete on the lateral portions, where they are also much finer; on the dorsal valve twelve or fourteen may be seen on each side of the center in the somewhat worn specimens, while on the ventral from eighteen to twenty-four can be distinguished.

Of one of the other two new species there is only a single poor, rather worn cast, not sufficiently marked for description, and of the *Orthoceras*, although numerous fragments are known showing distinctive features in the rapidly increasing dimensions of the tube and the deeply concave septa, none fit for description and illustration.

PNEUMATIC HOISTING.

BY H. A. WHEELER, ST. LOUIS, MO.

(New York Meeting, September, 1890.)

THE great depths attained by some of the older mines, and the much greater depth at which they will have to be worked in the not distant future, strongly emphasize the imperfect and inadequate character of our present hoisting-practice. The depth of 3000 feet having been exceeded at several places, and 4000 feet almost reached

at Mons, Belgium, we are brought to face the unpleasant fact that in these instances the mere dead weight of the rope itself is not only the principal part of the load on the winding-engine, but that the limit is being rapidly approached at which the ordinary cylindrical rope cannot safely be used, lest it break by reason of its own weight.*

While we Americans are not apt to borrow trouble by anticipating the problems of the future involved in the latter feature, the first-named defect of the present system of hoisting deserves our immediate attention, because it calls for an excessive consumption of power and consequent increased cost for fuel and plant. The European engineers have reduced the consumption of fuel by different systems of counterbalancing, while by the use of tapered ropes they have very much extended the practical working length of the rope before it will break of its own weight. Theoretically the tapered rope has no such limit, but practically, as used at present, it has its limit, though very much exceeding that of a cylindrical rope.

In attempting to replace the rope-system of hoisting, on account of the disadvantages alluded to, the French engineers have tried the endless-chain system of Saint Jacques at Montlucon, the Méhu man-engine at Anzin and Ronchamp, and the Blanchet pneumatic system at Epinac. The two former systems were found in practice to be too complicated, too expensive in repairs, and too dangerous, and have been abandoned; and while the pneumatic system was not a success, the failure to find workable coal and the consequent incompleteness of the plant explain that result and leave this most promising of the new systems still entitled to consideration.

The pneumatic system is entirely free from the above-mentioned defects of ropes, since it is not subject to the influence of depth, and gives no initial heavy dead-load due to the weight of the rope, being in this respect a theoretically perfect medium for "pulling" mineral out of the deepest possible mine. These advantages seem so great,

* Taking the weight of a cast-steel hoisting-rope of $1\frac{1}{8}$th inches diameter at 2 pounds per running foot, and its breaking-strength at 84,000 pounds, it should, theoretically, sustain itself until 42,000 feet long before breaking from its own weight. But taking the usual factor of safety of 7, then the safe working-length of such a rope would be only 6000 feet. If a weight of 3 tons is now hung to the rope, which is equivalent to that of a cage of moderate capacity with its loaded cars, the maximum length at which such a rope could be used, with the factor of safety of 7, is 3000 feet, or

$$2x + 6000 = \frac{84,000}{7} \therefore x = 3000 \text{ feet.}$$

in view of the rapidly increasing depth of mines,* that a sketch of the practical application of this system at Epinac, France, which the writer had an opportunity of visiting last summer, is herewith given. A complete description, giving the details of the calculations, construction and operation of the plant, with illustrations, will be found in the *Annales des Mines* of September–October, 1878, Paris.

The Blanchet Pneumatic System.

The pneumatic system, as installed at the Hottinguer shaft of the Epinac colliery, Saône-et-Loire, France, was designed by M. Zulma Blanchet, who was the managing director of a large company operating several mines and a local coal-railroad at Epinac, in central France. The first suggestion for thus using air was made in 1855 by M. Gruner, the director of the School of Mines of St. Etienne, and a model constructed on this principle was exhibited by M. Cavé at the first Paris exposition; but the credit of first putting the system into practical operation is due to M. Blanchet. The vertical Hottinguer shaft, started in 1863, had reached a depth of 2133 feet in 1871, and was to be sunk to a depth of at least 3300 feet; but unfortunately for the company and the entire mining world, no workable coal has thus far been met with, so that instead of developing a large mine to raise over 700 tons a day, it still remains (1889) a prospect, while the plans of M. Blanchet were not satisfactorily completed on account of the great expense of installation and the discouraging nature of the explorations. So, although a thorough test of the system was not made, according to M. Blanchet's more advanced ideas, yet sufficient experience was gained to furnish evidence as to the practical desirability of this novel system of extraction.

M. Blanchet's plan was to have one or two continuous air-tight sheet-iron cylinders or large tubes extending from the bottom to the

* To appreciate the rapid rate at which the mines in some of our older camps are growing deeper, it may be well to mention that the Comstock lode, of Nevada, was first worked in 1850, and had reached a depth of 3250 feet by 1886, when work was abandoned in the lower levels. The Lake Superior copper-district, which has several very deep mines, was first worked in 1844, and the Tamarack mine, though only started in 1882, has already attained the depth of 2820 feet, and is sinking two more shafts that are expected to cut the lode at a depth of 3700 feet within four years (a rate of over 1000 feet per year has been attained at this mine in shaft-sinking). A new mine, called the Tamarack-Junior, is likely to reach a depth of over 4000 feet within the next ten years if a chute of copper is found that has thus far been very persistent.

top of the shaft. See Fig. 1. Within the cylinder moved a piston, to which was hung a cage with as many decks as desired, nine decks being used at Epinac, while suitable air-tight doors open out at each level for allowing the decking of the cars. It was operated by exhausting the air from above the piston, and as the lower side was open to the atmosphere, the piston was raised by the difference in pressure of the atmosphere and the vacuum produced by the air-pump. For lowering, it was merely necessary to allow the air to re-enter above the piston by a throttle-valve, while the speed could also be regulated by throttling the escape of the expelled air from below the piston. It is obvious that, neglecting the friction of the piston, any lifting capacity desired can be obtained, for a given vacuum, by merely increasing the diameter of the tube, while the speed of hoisting will depend on the capacity (size and speed) of the exhausting-engine to maintain this vacuum.

Besides its legitimate function as a hoisting device, M. Blanchet was enthusiastic about the additional services it would perform in cooling and ventilating the mine through the introduction at each trip of the volume of air represented by the depth of the shaft and the diameter of the cylinder. He also proposed to make a special use of the system in fiery coal-mines by closing all openings of the mine, and then by working the exhausting-engine, to aid the disengagement of the fire-damp which is supposed to take place freely during a barometric depression. He urged that on subsequently reopening the mine and sweeping out the gas with the regular ventilating appliances, the mine would be made much safer.

Experimental Plant.

Before the erection of the final plant, an experimental one was built consisting of a tube $6\frac{1}{3}$ inches in diameter by 95 feet high, connected with a 12 horse-power vacuum-pump. With a vacuum of 18 inches to $19\frac{2}{3}$ inches, a velocity of 1181 feet per minute was attained in lifting 264 pounds, which is equivalent to an efficiency of about 80 per cent.* of the engine-power. The friction of the piston in the tube amounted to 10 per cent.

* A subsequent statement gives 18 horse-power instead of 12, as above, for the vacuum-pump; and only 50 per cent. instead of 70.7 per cent. of the gross load as live or coal load, while the gross efficiency is given as 40 per cent.; as the live load is only half of the gross load, 20 per cent. is the net or coal-raising efficiency for the system, beginning with the vacuum-pump as the source of power or 100 per cent.

Fig. I.

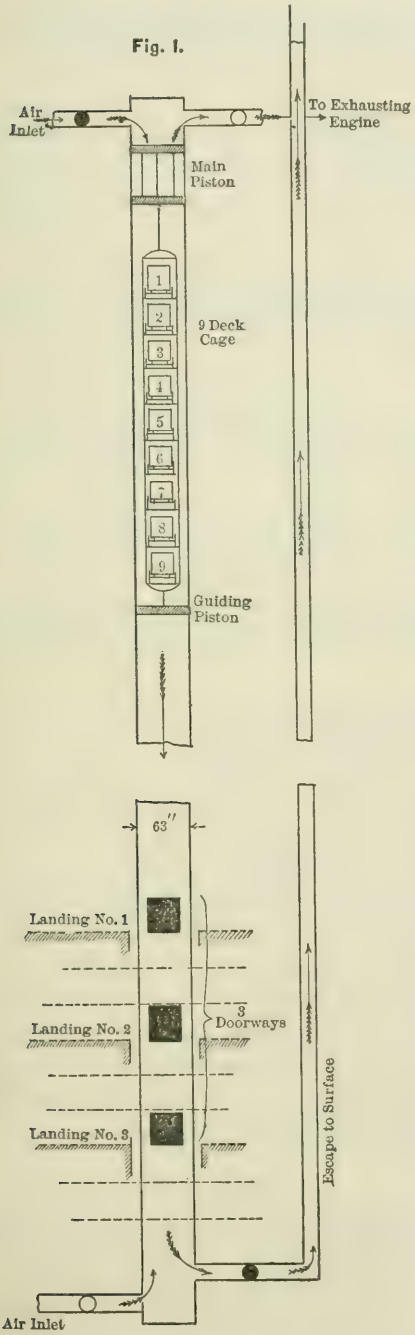


Diagram Illustrating Pneumatic Hoisting.

The load of 264 pounds was made up as follows:

		Pounds.	Per cent.	Per cent.
Dead Load	Piston,	31.9	= 12.1	29.3
	Cage,	37.4	= 14.2	
	Cars,	7.9	= 3.0	
Live Load, or Coal,		186.8	= 70.7	70.7
Total,		264.0	= 100.0	100.0

From these very encouraging figures the following estimate was made for the final plant, in which the tube was to be 63 inches in diameter:

Piston,	2.25 gross tons,	= 18.75 per cent.	Dead load.	
Cage,	3.00 "	= 25.00 "		
9 cars, at 0.25,	2.25 "	= 18.75 "		
Coal,	4.50 "	= 37.50 "	Live load.	
Total,	12.00 gross tons,	= 100.00 per cent.		

The time required for the round trip at the greatest contemplated depth, 3280 feet, was estimated at 7 minutes, as follows:

	Minutes.	Seconds.
To load at bottom (3 deckings),	0	30
To hoist,	3	00
Discharging at surface (3 deckings),	0	30
Descent,	3	00
Total time for round trip,	7	00

If this speed could be maintained, it would give a capacity of $38\frac{3}{4}$ gross tons per hour from a depth of 3280 feet with a single tube. With two tubes, the capacity would not alone be doubled, but a great advantage would be gained by operating them together, the exhaust-engine alternately exhausting and discharging from one into the other, practically counterbalancing one piston with its cage, cars, etc., against the other. Accordingly, M. Blanchet designed the Epinac plant to be operated with two tubes.

Final Plant.

The tube or cylinder was made of sheet-iron, 0.27 to 0.31 inch thick, in circular sections $4\frac{1}{2}$ feet in length, having butt-joints for the vertical seams, secured by countersunk rivets. The sections were bolted to one another by flanges made of angle-iron, using rubber packing to keep the joints tight.

The alignment of the tube was preserved and its weight sustained by suitably connecting it to the shaft timbers, so that the tube stood vertical, within the inclosing special timbering, without bearing any excessive weight on its base.

Cast-iron doors, opening outward, were placed at the landings, it being arranged at Epinac to load three decks at once, in order not to lose too much time in handling the 9 cars, as each deck only accommodated one car.

The chairs or catches for sustaining the cage at the landings passed through stuffing-boxes, to prevent any leakage of air, and were operated in the usual manner.

The total weight of a single tube, 63 inches in diameter and 1979 feet high, with all its doors, valves, chairs, hanging rods, bolts, etc., was estimated at 342 gross tons, of which the tube alone amounted to 245½ gross tons.

The train, or moving members, consisted of an upper double piston, to which was hung the cage with its nine decks, and a lower or guiding piston, the whole having a vertical length of about 52 feet. The pistons were packed with leather, backed by springs, and the whole train was made as light as possible by using steel in its construction.

For indicating the position of the piston, push- or touch-buttons were located along the tubes, which, on being touched by the piston in moving past them, made an electrical connection indicated on a series of tell-tale dials in the engine-room. Barometers and second-chronometers were also used as additional safeguards for indicating the position of the cage. The latter did so, in a rough way, by the time usually required to make a trip, while the former were connected with the tube about every 100 yards, and so showed the progressive position of the cage. Suitable valves at the landings, connecting with the atmosphere and the exhaust-engine, permitted the safe and easy landing of the cage, on their being operated by the lander, while safety-valves were used at the top and bottom of the tube to check slowly the velocity of the cage, and prevent the piston from knocking out the tube-heads, in case of inattention on the part of the lander.

After seven months spent in erection, a single tube, 63 inches in diameter, was ready for trial in 1876, it having been decided to first build only a single tube, and to subsequently add a second, making it a duplex plant, if found desirable.

A temporary exhaust-engine was improvised from a former hoist-

ing-engine of 64 horse-power, which raised a gross weight of 6 tons in 20 minutes, with a vacuum of 9.45 to 9.85 inches of mercury, from a depth of 1979 feet.

The piston was lubricated with very dilute soap-water, containing 0.27 per cent. soap, and 0.1 per cent. oil, and from 11 to 55 pounds were used per trip, depending on the dryness of the atmosphere.

Of the 64 horse-power developed by the exhaust-engine, 75 per cent. was utilized during the period of ascent, but as no useful work is accomplished during the preliminary exhausting of the tube, in producing the vacuum by which the train is finally started, the average efficiency was 56 per cent.; and as 50 per cent. of the gross load was coal or net weight, this gives a net efficiency of 28 per cent. of the engine-power, with a single non-counterbalancing tube.

The final exhaust-engine is a duplex, direct, horizontal engine of 739 horse-power, with 46 by 46 inches steam-cylinders, 46 by 113½ inches air-cylinders, and when making 23$\frac{1}{20}$ revolutions per minute, with 75 pounds steam pressure, cutting off at $\frac{1}{3}$ stroke, has a calculated efficiency of 70 per cent. With this powerful engine the time required to make a round trip, going a distance of 1979 feet, was as follows:

	Minutes.	Seconds.
For the ascent,	2	30
For the descent,	6	00
For making the landings,	3	00
Total time for round trip,	11	30

The load raised was as follows:

Weight of pistons, cage, etc.,	6,600 lbs.	= 54 per cent.
Weight of 9 cars @ 550 lbs.,	4,950 lbs.	
Weight of 9 car-loads coal @ 1100 lbs.,	9,900 lbs.	= 46 "
Total,	21,450 lbs.	= 100 "

The live load or coal amounts to 46 per cent. of the total weight raised.

With the conditions for hoisting at the other shafts at Epinac, using tapered manilla ropes, winding on reels, with one cage counterbalancing the other cage, M. Blanchet makes the following comparisons at different depths of rope and single-tube pneumatic hoisting:

Depth in Feet.	Cable Hoisting. Horse-Power.			Pneumatic Hoisting. Horse-Power.			Coal Consumed per ton Coal Extracted.	
	Expended.	Utilized.	Efficiency. Per cent.	Expended.	Utilized.	Efficiency. Per cent.	Cables.	Pneumatic.
820	225	90	40	800	360	45	55 lbs.	16.5 lbs.
1640	500	180	36	862	"	42	110 "	33.0 "
3280	800	224	28	985	"	36	220 "	66.0 "

These figures are favorable to the pneumatic system, as with only a single tube the useful efficiency is greater in all cases, and decreases at a less rapid rate than with rope hoisting for increasing depth, while the coal consumed per ton of coal raised is in consequence very much less in the pneumatic than with the cable system of hoisting. With two tubes, where the dead loads of the pistons, cage, cars, etc. (amounting to 54 per cent. of the total load), counterbalance one another, the efficiency would be correspondingly greater, and *very much higher* than in cable-hoisting. This feature was unfortunately never developed at Epinac, as the second tube was not erected, and hence the minimum working-cost of the system was not attained.

Advantages of the Pneumatic System.

M. Blanchet sums up the advantages of the pneumatic system of hoisting as follows:

1. It permits the working of mines at all depths.

2. It gives, with a single tube, a higher efficiency than cables, which increases as the depth increases; with two tubes, the efficiency will be still greater.

3. It does away with ropes, and is therefore more economical.

4. It assists in the ventilation of the mine, with the lowering of the temperature resulting therefrom.

5. It permits, while the miners are absent, the conversion of the entire mine into a pneumatic receiver, and, by the creation of an artificial barometric depression, facilitates the removal of the escaping fire-damp, and so makes the mine safer.

6. It leaves the shaft readily accessible for examination, repairs, and sinking, without interfering with the hoisting.

7. It removes the danger of accidents through the breaking of cables.

8. It gives greater security to the miners in ascending and descending.

9. It not only permits the lowering of the men, timber, filling, etc., without the expenditure of any power, but utilizes the work produced by the descending train in imparting a higher pressure to the air escaping from the bottom of the tube.

The only disadvantage, in M. Blanchet's opinion, is the greater cost of installation, which in his first estimate for the Hottingeur shaft he places at about 20 per cent. greater than that of installation for rope-hoisting.

The statements made thus far concerning the Epinac plant have been quoted from the several brochures* by M. Blanchet, and as he was an enthusiastic advocate of the pneumatic system, his claims in its favor demand investigation.

Criticism of the Pneumatic System.

I. The statement that the pneumatic system has no limit in depth is certainly true theoretically, and there seems to be no practical difficulty in applying it to the greatest depth attainable. This is the salient and vital idea of the system, which would seem to entitle it to precedence over ropes after a certain depth has been attained.

II. That the efficiency of the pneumatic system is greater than that of cables must necessarily follow *if the depth is very great and the ropes are not* counterbalanced, as the principal part of the load is then the cable; but if the ropes are counterbalanced, especially if by an under-rope, then the pneumatic system cannot be equal in efficiency to cables, if both systems are operated on the double or duplex plan, so as to counterbalance the dead weight of the cage, cars, etc. For if we start with equal amounts of energy in the delivery of the steam-cylinders of the exhausting- and winding-engines, in the one case this must overcome the appreciable friction of the stuffing-box, piston and valves of the vacuum-cylinder, suffer the decided waste of energy due to the clearance of the valves, ports, etc., overcome the slight frictional resistance of the air travelling through the connecting-pipes, valves and tubes, do the useless and uncertain work of removing the air brought in by leakage at joints, valves, stuffing-boxes and doorways of the tube, and overcome the friction of the pistons in the tube before it begins to be usefully expended in raising

* *L'Exploitation de la Houille*, Autun, 1872. *Tube Atmosphérique du Puit Hottinguer*, St. Etienne, 1876. *Houilleres d'Epinac*, Paris, 1878. *L'Exploitation des Mines à toute Profondeur*, Paris, 1878. *Annales des Mines.*

the load. In the other case, a direct, first-motion, hoisting-engine must overcome the friction of the shafts of the drum and sheaves (well lubricated), expend the small amount of energy required to bend the rope about the drum and sheaves, and overcome the slight friction of the cage in its guides before it can be utilized in useful work. While these losses of energy can be expressed in an equation, the coefficients by which they should be modified have not all been satisfactorily determined. M. Blanchet puts the losses in the vacuum-pump alone at 25 per cent. (and the makers at 30 per cent.), while the friction-loss of the piston in the tube was 10 per cent. in the experimental tube, though undoubtedly it is much less in the large tube when well lubricated.

The entire losses of energy in rope-practice usually range from 10 to 20 per cent. Leaving out of consideration the greater efficiency of a direct-acting hoisting-engine over a vacuum-pump, and assuming that the friction of the air through the pipes, valves and large tube do not consume more power than that required to bend the rope over the sheaves and drum, we still have the inappreciable friction of an ordinary cage in its vertical guides, which is almost *nil* with proper alignment, as against the decided and unavoidable friction of the air-tight piston in the tube. So that while M. Blanchet's figures speak strongly for the greater efficiency of the single-tube air-system, when compared with a duplex cable-hoist they will not bear investigation. His own broad and sanguine statement that the efficiency of the vacuum-pump plant will be as great as 70 per cent. at once gives an effective or coal-raising efficiency of only 32.2 per cent., when it is remembered that only $\frac{9900}{21450}$ of the load raised is coal, whereas he quotes about 42 per cent. for the pneumatic system and 36 per cent. for the cable system. As he does not give the details by which he obtained his figures, further errors cannot be specifically pointed out.

The actual fuel-consumption, when the pneumatic plant was operated at Epinac, amounted to 10 tons per day in raising from 10 to 50 tons of coal; but as the fuel was inferior coal, and as hoisting in carrying on prospecting is very irregular as well as petty in amount, this excessive consumption cannot fairly be charged against the system, though it certainly does not encourage the hope of a marked economy in fuel.

III. That there will be no expenditure for ropes in the pneumatic system follows necessarily, together with the avoidance of their renewal every six to eighteen months. But with proper size of drums and sheaves, feed-screws to prevent side-cutting of the

rope as it winds on the drum where the engine is close to the shaft, and springs at the capping, a first-class steel rope should, as it does in the best English practice, last from two to five years, or raise from 200,000 to 600,000 tons of ore or coal before it is worn out. But while the cables are dispensed with, the new feature is introduced of a thin wrought-iron cylinder, held by countersunk rivets, which deteriorates every time that the snugly-fitting piston passes through it. The piston-packings, of which there are three sets, need frequent removal, as they must be kept tight to avoid leakage; while there is quite an appreciable daily outlay for lubricating the tube, as from ten to fifty-five pounds of 3 per cent. solution is used per trip. If the shaft is not sufficiently wet to prevent the decay of the timber, the heavy timbering system required to sustain the tube will also need renewal, at no small cost, while the expense of maintaining a very large, fast-running, expanding exhausting-engine, with all its valves, will be greater than that of an equivalent direct hoisting-engine. Unfortunately, the Epinac plant was not operated long enough to develop these maintenance expenses, but it is quite evident that the repairs and renewals of a cable-plant will finally be decidedly less than those of the pneumatic system.

IV. The aid to ventilation is of some advantage, particularly in metal-mining, where only natural ventilation is so largely depended on. With a tube 3000 feet long and 63 inches diameter, there would be delivered into the workings during a round trip of twelve minutes, a volume of 5425 cubic feet of air per minute, which *positive acquisition* would be no insignificant item; and the deeper the shaft, the greater would be the volume of air expelled at each trip. In coal-practice, however, this amount of air would be entirely too small to be entitled to much consideration.

V. The novel application of the exhausting-engine, through the medium of the tube, to the production of an artificial barometric depression throughout a tightly-sealed mine, is at least attractive for mines troubled with gas, and undoubtedly would be advantageous *in some cases*. Where the gas is generated in the goaf or old rooms, and consequently its escape into the workings is facilitated by the lowering of the atmospheric pressure, a benefit will be derived in thus frequently forestalling nature and keeping the air safe. But where the gas is being given off by the slow continuous escape under great pressure from the coal (which is the case most frequently, though, fortunately, it is usually not so dangerous), or where the gas results from blowers due to falls in the roof or breaks in the floor

(the most dangerous instances), no good will result in thus turning the mine into a vacuum chamber. It is also obvious that, in very extensive workings, the time required to produce an appreciable lowering of the pressure of such a huge volume of air would exceed the period of an off-shift, while no repairing or other off-shift work could go on, on account of the danger of gas, with the ventilation at a standstill. Hence only Sundays or holidays could be used for this purpose during the regular working of the mine, which is not frequent enough to make it perfectly safe. If the roof is badly broken, especially if the breaks connect with the surface, the application will be useless, while in many cases it will be difficult to prevent excessive leakage into the mine at the shaft-entrances. This novel feature, therefore, is capable of only a few applications in the gas-troubled mines, and even then may not be valuable on account of the magnitude of the workings. Finally, in any mine troubled with gas from any source, the regular ventilating facilities should be on so large a scale that the gas is so diluted as to preclude any danger of explosion, except from sudden large blowers.

VI. The ease of repair of the shaft is greater in the pneumatic system, since it is possible from the capstan-engine to make any inspections or repairs in the shaft, short of renewing the tube, without in the least interfering with the hoisting.

VII. While accidents due to cable-breakage disappear, a more serious danger arises in the much greater vigilance required on the part of the lander in keeping track of the noiseless invisible cage, as the tell-tales, though ingenious, are hardly satisfactory. The unreliability of electric sounders or tell-tales in damp shafts is well-known, while barometers are very crude aids in making a landing, where the margin for dropping the cage on the chains is so small. Although a large safety-valve is placed at the top and bottom of the tube to slowly cushion the cage and prevent accidents arising from carelessness in over-hoisting, it is evident that the momentum of the 10-ton train will demolish any such device, should it pass the terminal landing with its speed unchecked, in spite of any reasonable margin for over-hoisting. Furthermore, while the breakage of ropes has by no means ceased to occur, good devices can be used, at least on vertical shafts, that act promptly and arrest the cage in case the rope breaks, if *reasonable attention* is bestowed upon keeping the safety-catches in good condition.

VIII. That the pneumatic system will be more exempt from accident than the cable system seems hardly probable in view of the

previously-mentioned want of a reliable, continuous system of recording the positions of the cage in the tube. Overwinding or carelessness on the part of the landers or engineers, quite a frequent cause of accidents at present, seems likely to become even more dangerous in the pneumatic system. The trouble arising from the distortion of the shaft with consequent jamming of the cage in swelling or heaving ground, would be exaggerated in the pneumatic system, with its train fifty-two feet long; for while the tube stands free by itself away from the shaft-lining, the displaced timbers supporting it would so distort and throw it out of line as to readily cause the jamming or sticking of the train in the tube.

The Epinac plant was not operated sufficiently long to develop reliable figures as to accidents with the pneumatic system. It created, however, a favorable opinion among the miners as being at least a very pleasant way of being raised and lowered.

IX. The want of merit in the last claim over rope-hoisting, when properly operated, is self-evident, and needs no comment.

How well these claims were realized at Epinac is most quickly appreciated upon finding a winding-engine busy in the same shaft in carrying on the exploratory work, while the pneumatic plant alongside of it, with one tube and its massive vacuum-engine in complete running order, has been idle for several years, as it was too expensive to keep in operation.

One item of excessive cost in the pneumatic system, to which M. Blanchet makes no allusion, is the greater labor-expense per ton hoisted. If worked at full speed, the Epinac plant could raise about 250 tons in ten hours. The force required was as follows: four firemen, one engineer, one runner, two landers, four cagers, one machinist; total, thirteen men, or $\frac{250}{13} = 20$ tons per man in the hoisting-department.

Winding-engines of even less power, hoisting from as great a depth, frequently raise from 1000 to 1500 tons in 10 hours, with a force no greater than two more cagers, or hoisting from 60 to 100 tons per man, making the labor-charge for hoisting $\frac{1}{3}$ to $\frac{1}{5}$ of that of the pneumatic system.

The Cost of the Pneumatic System.

Before alluding to the outlay involved in the Hottinguer shaft, it is necessary to deal with the great difference between French and American practice in plant-expenditure.

The French mining engineers, with usually an abundance of

cheap money at their disposal, are characterized by their very heavy outlays in permanent plant. They make everything to last, and also, be it said to their credit, design with regard to appearances and artistic effect. Hence there is a large outlay at the Hottinguer shaft for a masonry lining for the entire shaft, and a very large, substantial, brick shaft-house, with a high brick chimney for the boilers which would be regarded as extravagant in America. Moreover, in executing their plans, they have recourse to a staff of trained engineers that is as yet far in advance of our so-called "practical" way of doing the same work. For these reasons, there has been a lavish outlay at the Hottinguer shaft that is, by no means, to be all charged against the pneumatic system. The statement made by one of the directors, that the experiment had cost them over $500,000, refers to the entire outlay at the shaft. The expense of the pneumatic system proper is given to me by M. Naugerode, the present superintendent, as follows:

Tube and connections, ...	$70,000
Erection, including winch-engine, etc., ...	74,000
Exhausting-engine (or vacuum-pump), ...	42,000
Total, ...	$186,000

These figures show that the erection of the tube cost more than the tube itself, and that the tube (with its connections) cost about $35 per running foot. As there are 775 pounds metal per linear foot this shows a factory-cost of the tube of about 4½ cents per pound, a figure that is about half what such work would usually cost in this country, when it is remembered that 72 per cent. of the metal required makes up the countersunk-riveted main tube. It is probably safe to say that such a tube erected complete in this country would cost over $100 per linear foot; and this does not include the vacuum-engine, boilers, housing and the primary cost of first sinking the shaft. So that a pneumatic plant, complete, would call for at least double the capital of a rope-hoisting plant of equivalent grade, and when finished, aside from probably much greater operating expenses (fuel and repairs), would not have a fifth of the capacity of the cable-system, as reckoned on the actual figures of the Epinac plant.

Summary of the Pneumatic System.

With a so much heavier investment of capital, with such a very small capacity compared with winding-engines of equal power, and

with the probability of a much greater daily operating expense, it is needless to say that the pneumatic system, as thus far developed, cannot compete at all with the rope-system of hoisting, in spite of the theoretical and the few practical advantages it possesses over the cable system.

It is, therefore, necessary to look deeper into our American winding practice, to search for opportunities for improvement, so that the requirements for increased depth in the not-distant future may be met.

Considerations for the Future.

Theory indicates that tapered ropes have no limit in depth, and consequently that, by tapering and counterbalancing, ropes may be used in the deepest mines; but practically, in using flat ropes with reels, as they are now made, there is great risk of the rope slipping off the top coils and jamming down in the narrow V-space between the side of the reel and the coiled rope, with dangerous consequences, if the rope tapers to half of its extreme width. By using a guide-sheave to wind the rope hard against one side of the reel, this danger may be overcome, though at the expense of more rapid wear of the rope, especially of the lacing. Flat ropes are now successfully and easily made of tapering section, and perhaps a better device can be used than the above guide-roller, to avoid jamming in the reel.

Round ropes are made tapering, and if the drum is grooved to properly receive the rope, it gives no trouble in winding like the flat rope. But the American rope-makers do not as yet advocate round tapering ropes on account of the difficulty of manufacturing with perfect reliability, though I understand that they are made in a satisfactory manner in Germany.

If tapered ropes are ignored on account of their greater cost, and because of the objections of manufacturers, cylindrical ropes may still be used for unlimited depth by submitting to the low efficiency of the system adopted at the deepest shaft of the colliery of the "Société des Produits," at Mons, Belgium, where a duplex, first-motion engine hoists from the 1000 meter (3280 feet) level to the surface. At the 1000 meter level another hoisting-engine, run by compressed air supplied from the surface, hoists from the 1100 meter level, the lowest producing level at present, though the shaft is being sunk to the 1200 meter level (3937 feet). The surface engine is 27½ inches by 78¾ inches, with a 25-foot reel, on which winds

a flat manilla rope that tapers from 11 inches down to 6½ inches, while the cage holds only 2 cars. The air is supplied by a duplex Dubois-François air-compressor with 15½ inches by 59 inches air cylinders, run at 18 revolutions per minute and furnishing air at 45 to 50 pounds pressure.

While such a system of establishing new hoisting stations with engines run by compressed air whenever the rope becomes inconveniently long (it is not counterbalanced at the above pit) can be carried on indefinitely, the outlay for plant becomes so great, the extra labor and time involved in charging cages at the intermediate levels are so expensive, and the efficiency of a compressor-plant is so low, with its consequent heavy fuel consumption, that the cost of installation is almost as great as in the pneumatic system, while the operating expenses are probably greater.

Improvements in American Hoisting Practice.

The simple, first-motion, cylindrical rope-hoist, constructed with the latest improvements, possesses a limit so far beyond our present practice, when properly designed, that there need be no anxiety for a very long time to come as to its range of application. It possesses the advantages of a minimum cost of installation, the cheapest operating plant and the greatest capacity of all known systems of hoisting. Thus far, in America, our hoisting practice, with few exceptions, is anything but creditable. We are behind European practice. The following suggestions show how we can still very much exceed our present limits.

(*a*) *By Counterbalancing the Dead Load.*—If the system is arranged duplex, or with a double-compartment shaft, the dead loads of the cage and cars counterbalance one another; and if the ropes are counterbalanced, which is perfectly effected by an under-rope, then there remains only the work of raising the coal or live load plus the small effort necessary to overcome the friction due to the dead load (rope, cage, cars). Hence only a moderate-sized engine is called for, instead of the large engines that are so frequently seen at our deep mines. Thus, in the Lake Superior copper-district, where over 7000 tons of copper-rock are raised per day from a depth of 1000 to 2500 feet, 60 to 75 per cent. of the engine-power is wasted (except at two mines), by using single hoists in which a skip weighing from 1½ to 3 tons, and a rope weighing 1½ to 4 tons, are hoisted each trip in bringing up 1½ to 2 tons of rock; while the energy of the descending 3 to 7

tons of skip and rope adds to the expense by wearing out the brakes by which the speed is kept within a safe limit. Such a grossly extravagant system of hoisting not only calls for an excessively large hoisting-engine, but the fuel-consumption is about three times as great as it should be to do the above amount of work.

(*b*) *By Using a Smaller Factor of Safety for the Rope.*—For hoisting-ropes the factor of safety should be made as low as possible, consistent with uncertainty in manufacture, abuse in use, sufficient margin to permit the reduction in section due to wear, and a working-strain safely below the limit of elasticity. For the larger the factor of safety used, the greater is the section of the rope for a given load. This increases the dead load due to the rope's own weight, which even if counterbalanced augments the mass to be put in motion each trip, with the consequent strain in starting.

The increased size of the rope decreases its life, as the greater the diameter, other things being equal, the more severe are the bending strains and the quicker the rope destroys itself.

Now the factor of safety usually taken for steel ropes is 7, with 5 as a minimum and up to 10 as a maximum. For standard practice I would advise 4, provided: that the diameter of the sheaves and drums be *at least* 100 times the diameter of the rope for slow hoisting and *at least* 150 times for fast hoisting; that a spring* be interposed between the cage and the rope-capping, to ease the sudden strain in starting; that the rope be kept properly oiled: and that the rope be inspected daily for its entire length, for careful observation of the condition of the wires and prompt rejection of the rope when a certain number are broken.

By thus easing the bending of the rope by the use of much larger sheaves and drums than is usual in American practice; by removing the shock due to starting; and by frequent oiling to prevent corrosion, they would last *at least as long* as they now average in American practice. By lagging the sheave with wood; by having the engine set far enough from the shaft to avoid side-thrust and consequent lateral wear of the rope at the sheave and drum (carrying the rope on *wooden rollers* from the sheave to the engine if necessary); and by carefully maintaining the entire hoisting-plant in proper alignment, the life of the rope would probably be found

* The intervention of an elastic connection, whether a spring or solid rubber, between the capping and the cage, need not in the least interfere with a chain connection of the rope to the cage, by which latter device the cappings have to be less frequently renewed.

to be considerably greater than at present, in spite of the much smaller factor of safety used. By thus using a factor of safety of 4, the safe working-length of the $1\frac{1}{8}$-inch cast-steel rope, when sustaining a 3-ton cage-load, will be increased from 3000 feet to 7500 feet.

By decreasing the speed of hoisting, the wear of a wire rope is diminished; but this necessitates increasing the load to maintain a given output in the same time. Aside from the other disadvantages of excessive load, the maximum size of the cars used is frequently fixed by conditions that it is not economical to change in order to use large cars. Hence the more rapid wearing out of the ropes due to high speed in deep hoisting is willingly tolerated, in order to secure the capacity and economy due to fast hoisting; and the tendency of the times is to use larger and more powerful engines in order to obtain higher speeds as the mines become deeper; so that this method of prolonging the life of a rope is not in favor at present.

(c) *By the Use of the Best Material for the Rope.*—Fiber and iron ropes are no longer used in deep hoisting in American practice, but the term "steel" covers a very broad range of material used in our present practice. The very soft steel ropes certainly possess the greatest flexibility, but they are deficient in strength, as the material has a breaking-strength of only about 60,000 pounds to the square inch. The best grade of plough-steel, on the contrary, has a strength of 300,000 pounds per square inch, but does not always give satisfaction as used with sheaves only 50 to 100 times the diameter of the rope, since the wires break too readily under such sharp bending. But if used with sheaves and drums 150 to preferably 200 times the diameter of the rope, and if the wires are laid at an easy pitch with the "Lang lay," no such trouble should arise; with such a high-grade plough-steel rope of $1\frac{1}{8}$ inch diameter, the safe working-length, using a factor of safety of 4, would now be about 30,000 feet, or 12,500 feet with 10 as the factor of safety. While this represents the highest grade of rope that is made to-day, we see that it is possible to hoist with perfect safety with a cylindrical rope from a depth that is vastly beyond any limit that we can reach, on account of the heat alone.

The price of aluminum has not yet permitted a thorough investigation of its use for hoisting-ropes as the basis of trustworthy predictions. But the unprecedented progress made in the past year in lowering the price from $6.00 to $1.50 a pound, and the encourag-

ing outlook for further material reduction of the present price, makes the possibility of using this metal or its alloys for hoisting-ropes worthy of consideration.

The very low specific gravity of aluminum, 2.6, which is about one-third of that of steel, and its great strength, make it an ideal metal for a deep-hoisting rope, where a *minimum* of *weight* and a maximum of strength are demanded. If this metal or its alloys should prove to have the requisite flexibility, toughness and strength, to make a reliable, durable, hoisting-rope, the question of deep-hoisting with only cylindrical ropes is again settled beyond all cavil.

(*d*) *By Designing the Cage with a Minimum of Weight.*—As our mines get deeper, economy demands that we shall not only use at least 2-deck cages, but rather 3- or 4-, or even 6-deckers, as used at some of the deep mines in Belgium. The prevalent objection to multiple-deck cages is the time required in decking, even if the decking is carried on simultaneously from two landings or levels. To deck from more than two, or at most three, landings or levels at once makes the stations very complicated underground and is not desirable, even should the surface-plant be so arranged that there is no objection to having the ore or coal coming off on two or three levels. As our American plants are too frequently arranged, the time required to run off the loaded and run on the empty car seldom takes as little as 10 seconds, more frequently over 15 to 20, and sometimes even 30 seconds; and if this amount of time is multiplied by 4 or 6, for a 4- or 6-deck cage, it makes a serious inroad into the hoisting-capacity of the plant.

But with ample facilities for accommodating both loaded and empty cars at the landings, with *sufficient help* to quickly handle the cars, there is no reason why we should not at least do as well as the Belgians, who load and unload a 6-deck cage in 30 seconds, or 5 seconds per deck,* as their regular daily speed.

A multiple-deck cage, with only one car on a deck, can be designed of steel, with a very light, and yet stiff strong frame, in which the ratio of the cage-weight to that of its contents is de-

* At the "Sacre Madame" colliery at Charleroi two men and a girl pull off the loaded cars from one side of the cage, starting it off before the cage rests on the chairs, while two stout girls push on the empty car from the other side at the rate of 5 seconds per decking. The cars weigh 660 pounds each and hold about 880 pounds, and 350 to 400 tons of coal are raised per day. The shaft is 2625 feet deep, and 80 to 90 seconds are required to make the trip, with a 600 horse-power engine (using a flat manilla rope made of 8 strands, that tapers from 12½ inches down to 7½ inches in width).

cidedly less than with our much heavier single-deck cages; so that the strain on the rope due to the weight of the cage (immaterial as to whether it is operated duplex so as to counterbalance) will be very much less for a given carrying-capacity, than with our usually strong, needlessly heavy single-deck cages. Hence the rope can profit by this decrease of its load in sustaining its own weight for a still greater depth.

If two cars are run on a deck with the idea of saving the time that is supposed to result therefrom in decking, the cage-weight per ton of carrying-capacity will be increased over that of single-decks, in consequence of the much stronger bracing required for a cage long enough to carry two cars.

The six-deck steel cage at the Sacre Madame pit weighs 3600 pounds, while the weight of the 6 loaded cars is about 9000 pounds, which gives a ratio of cage-weight to carrying-capacity of 1 : 2.5.

Conclusion.

The pneumatic system of Blanchet is practicable, quite safe, and unhampered by depth; but the installation is very much more expensive than that of the cable-system (approximately twice as great), while the operating expenses are greater for labor, repairs and fuel.

The cable-system can be so improved by counterbalancing as to materially decrease the present expense of hoisting, while by more careful designing and operating, without materially increasing the plant-outlay or the operating expenses, the present type of cylindrical ropes can be used for depths beyond what we are likely ever to attain by making the weight of the cage a minimum, by using a lower factor of safety, and by adopting the strongest rope obtainable.

The possibility of using aluminum may still further increase the range of the cylindrical rope.

When our manufacturers meet with sufficient demand to encourage them to perfect the round type of tapering rope, there will then be absolutely *no limit* to the range of cable-hoists, although the best grades of cylindrical steel ropes will carry us, with the highest factor of safety ever used (10), to a greater depth than we will probably ever be able to attain.

THE PHYSICAL AND CHEMICAL EQUATIONS OF THE OPEN-HEARTH PROCESS.

BY H. H. CAMPBELL, STEELTON, PA.

(New York Meeting, September, 1890.)

Introduction.

The following pages discuss some problems connected with the manufacture of steel by the open-hearth process. The methods employed necessarily enter into the domain of what is called theory; but the results are eminently practical. The determination of the various losses of thermal power and the investigation into the chemical reactions in the bath, are important to every manufacturer. It will be understood that definite figures concerning one furnace and one system, will not apply to all other furnaces and systems; but it is hoped that methods are developed, which are capable of general application and which will shorten the labors of others.

The gas-calculations contain nothing entirely novel; but certain methods appeared to be very convenient and they are here elaborated in order that others may have a model to follow.

The determination of the loss of iron by weighing the slag and finding the percentage of iron therein, is probably unusual. If so, the fact is to be regretted, since this is the only way that an accurate result can be obtained. The common method of estimating the loss in open-hearth work, is to subtract the weight of metallic products from that of the materials charged. Nothing could be more unsatisfactory, inaccurate and meaningless. Pig-iron, steel, wrought-iron and ore are charged into the furnace; complicated chemical reactions occur; the sand of the pig-iron, the slag from the wrought-iron, the gangue of the ore, all rise to the top; the silicon, manganese and carbon burn and leave the metal; some of the iron oxidizes and goes into the slag; some of the ore is reduced and gives up metallic iron; the recarbonizer is added, the mission of which is to waste itself; in the pouring of the steel, pit-scrap is made in the ladles, around the moulds and under the spout; the errors in weighing the products join hands to make accuracy impossible; and then the "*loss*" is determined *by difference!*

If by "loss" is meant a name for undiscovered errors, the word is appropriate; but such a term has no place in metallurgical science. The factor in this "loss" which tells the story of good or bad practice, is the waste of metallic iron, and there is only one way of finding this. It is the method employed in all chemical determinations: weigh the mass containing the element sought, find the percentage of the element and calculate its weight.

There will also be found herein described a mode of calculating the weight of the slag at any given period of the operation. The data necessary for this must be collected with great care; but the validity of the method cannot be questioned.

In some of the calculations, as, for instance, on the reactions in gases passing through checkers, it will be found that the results do not accord with our expectations. It is, therefore, proper to remark that the investigations were carried out by men of scientific training and practical experience; that all known causes of error were carefully eliminated and that no personal reason existed for the falsification of returns. Under such circumstances, a seemingly absurd result should be chronicled, as it indicates either the incorrectness of our theories, or, by proving the possibility of unseen errors, shows that happy answers may be due to a fortuitous cancellation of mistakes.

The following is a synopsis of the argument and arrangement:

Synopsis.

General Data:

Sec. 1. Chemical methods and physical data on which calculations are based.

Thermal Calculations:

Sec. 2–7. The methods employed, with examples.

The Producer:

Sec. 8. General observations on certain elements of producer-practice.
Sec. 9–10. Calculations in producer-work.
Sec. 11. On averaging gas-analyses.
Sec. 12–15. Use of steam in producer.
Sec. 16–18. Loss of thermal power in producer.
Sec. 19. The gas used as a basis for subsequent calculations.
Sec. 20 The general equation of the producer.
Sec. 21-25. Experiments on the use of steam.

The Combustion of Gas—Theoretical:

Sec. 26–28. The theoretical combustion of various gases.
Sec. 29–30. The latent heat of products of combustion of various gases.

The Combustion of Gas—Practical:

Sec. 31. Analyses of products of combustion.

Sec. 32–36. Excess of air in products, and loss of heat thereby.

Sec. 37–42. Deficiency of air in products, and loss of heat thereby.

Sec. 43–59. Analyses and calculations of products of combustion.

Note—Sec. 52–57. The influence of the chemical reactions of the bath upon the products of combustion.

The Use of Oil-Fuel:

Sec. 60. Analyses and calculations on products of combustion.

The Chemical Equation of the Acid Process:

Sec. 61–68. The chemical reactions considered with particular regard to the formation and reduction of iron oxide.

The Chemical Equation of the Basic Process:

Sec. 69–81. The chemical reactions considered with particular regard to the formation and reduction of iron oxide and the oxidation of phosphorus.

The Thermal Equation:

Sec. 82–86. Determination of factors.

Sec. 87. General thermal equation.

Sec. 88. Consumption of coal in an empty furnace.

The Dynamical Equation:

Sec 89. General observations.

General Data.

Section 1. *Chemical Methods and Physical Data.*—The following is a brief outline of the methods employed in making the analyses of gases and slags mentioned in this paper.

Gas-Analysis:

Orsat apparatus (with spiral of platinum for combustion of H and CH_4) for estimation by absorption of CO_2, C_2H_4 (C_3H_6—C_4H_8), O, CO, H, CH_4 and N.

(1) CO_2 by solution in caustic potash.

(2) C_2H_4 (C_3H_6 — C_4H_8) by bromine water, subsequently removing vapor of bromine by potash pipette.

(3) Oxygen by alkaline pyrogallol.

(4) CO by cuprous chloride and hydrochloric acid. [Absorption is complete, if cuprous chloride is fresh. If solution is used too long, it will give up CO to a gas containing none.]

(5) H and CH_4 by combustion with oxygen in platinum spiral.

(6) Absorb CO_2 from combustion of CH_4 in potash.

(7) Nitrogen remaining unabsorbed.

Slag-Analysis:

One gramme dissolved in HCl plus a little HNO_3. Evaporate to dryness, redissolve in HCl + H_2O. Filter off SiO_2. In filtrate expel HCl by HNO_3 and precipitate Mn by $KClO_3$, weighing Mn finally as pyrophosphate. Filtrate from MnO_2, run down to small bulk, add HCl, and make basic acetate separation.

P_2O_5 determined from basic acetate precipitate by dissolving in HCl, running down, adding citric acid, ammonia and magnesia mixture; weigh as pyrophosphate of magnesia.

CaO and MgO determined in filtrate from basic acetate separation. FeO determined in a separate portion by dissolving in HCl, reducing with stannous chloride, adding mercuric chloride and titrating with bichromate of potash.

Phosphorus in Pig-Iron:

This was determined by the acetate method, as described by Blair.*

THE PHYSICAL DATA.

	By weight.			By volume.	
	(a)	(b)	(c)	(d)	(e)
Air, . . .	1.29	.237		.307	
N, . . .	1.26	.244		.307	
O, . . .	1.43	.218		.312	
H_2O, . .	0.80	.475		.382	
CO_2, . .	1.97	.216		.426	
H, . . .	0.09	3.405	34,500	.305	3,090
			30,000		2,660
CO, . . .	1.25	.248	2,400	.310	3,000
CH_4, . . .	0.72	.593	11,900	.424	8,480
C_2H_4, . .	1.25	.369	11,200	.463	14,000

1 kilo C burned to CO_2 produces 8080 calories.
1 " C " CO " 2470 "
1 " CO " CO_2 " 2400 "

(a) Weight of 1 cubic meter in kilogrammes.
(b) Specific heat per kilogramme.
(c) Calorific power per kilogramme.
(d) Specific heat per cubic meter.
(e) Calorific power per cubic meter.

THERMAL CALCULATIONS.

SEC. 2. *Determination of the Specific Heat of a Mixed Gas.*—The number of calories absorbed in heating a given volume of a mixed

* *Chemical Analysis of Iron*, Lippincott, 1888, p. 72. Mr. Blair has also described his practice by this method in a paper entitled "Determination of Phosphorus in Iron and Steel," *Trans.* iv., 212.

gas through a given temperature, will be the aggregate number of calories absorbed by the several components.

The number of calories absorbed by each component will be the product of its volume, expressed in cubic meters, and its specific heat. The calculation is illustrated by the following examples:

Gas A.

	(a)	(b)	(c)
CO_2,	5.2	× .426 =	2.215
O,	0.4	× .312 =	0.125
C_2H_4,	0.4	× .463 =	0.185
CO,	22.8	× .310 =	7.068
H,	8.5	× .305 =	2.593
CH_4,	2.4	× .424 =	1.018
N,	60.3	× .307 =	18.512
Specific heat, 100 cubic meters,			31.716
" " 1 " "			0.3172

Gas B.

	(a)	(b)	(c)
CO_2,	2.8	× .426 =	1.193
O,	0.4	× .312 =	0.125
C_2H_4,	0.0	× .463 =	——
CO,	27.6	× .310 =	8.556
H,	2.7	× .305 =	0.824
CH_4,	2.2	× .424 =	0.933
N,	64.3	× .307 =	19.740
Specific heat, 100 cubic meters,			31.371
" " 1 " "			0.3137

(a) Composition in percentage by volume.
(b) Specific heat per cubic meter.
(c) Specific heat of 100 c.m. of mixed gas.

Gas A is an average gas; Gas B, an abnormal specimen. It will be noticed that the results are nearly alike. This follows from the fact that the specific heats of O, CO, H and N are nearly the same.

The amount of C_2H_4, in any ordinary producer-gas, is too small to affect the result materially. The only factors, the variation of which alters to any extent the general value, are CH_4 and CO_2. The influence of each can be calculated as follows:

Specific heat, CH_4,	0.424
" average gas,	0.317
Difference,	0.107
Specific heat, CO_2,	0.426
" average gas,	0.317
Difference,	0.109

It follows that a variation of 1 per cent. in the amount present of either of these gases will cause a variation of 1 per cent. of this difference = 0.0011, or one-third of 1 per cent. of the total.

For practical work, therefore, the variations in specific heat, due to ordinary irregularities in composition of producer-gases, fall below the unavoidable errors of temperature determinations. Hence, in subsequent calculations, the value of 0.317 calories per cubic meter will be taken as the specific heat of any ordinary producer-gas. The average temperature of such gases, as they leave the fire, may be assumed to be 750° C. Hence, the loss in sensible heat, per cubic meter of gas produced, will be $750 \times 0.317 = 238$ calories.

Sec. 3. *Determination of the Calorific Power of a Mixed Gas.*— The number of calories, produced by the combustion of a given volume of a mixed gas, will be the aggregate number of calories produced by the combustion of the several components.

The number of calories produced by each component, will be the product of its volume, expressed in cubic meters, and its calorific power, expressed in calories per cubic meter. This calculation is illustrated by the following example:

	(a) c.m.	(b)	(c) Calories.
CO_2,	5.2		
O,	0.4		
C_2H_4,	0.4	× 14,000 =	5,600
CO,	22.8	× 3,000 =	68,400
H,	8.5	× 2,660 =	22,610
CH_4,	2.4	× 8,480 =	20,352
N,	60.3		
	100 give		116,962
	1 gives		1,170

(a) Composition in percentage by volume.
(b) Calorific power per cubic meter.
(c) Calorific power in 100 c.m. of mixed gas.

Sec. 4. *The Carbon-Ratio.*—In the operation of a producer for the manufacture of gas, a considerable portion of the fuel is removed from the grates with the ash, in the form of coke or carbon. This loss will vary from 5 to 20 per cent. of the weight of the original coal. A part of this coke may be separated and put back into the fire, or it may be used for other purposes. Another portion of the fuel is converted into soot and tar; these are condensed and deposited in the tube and serve no useful purpose. From these considerations

it follows that a given weight of coal is not a convenient basis for some thermal calculations; in place of it, there may be taken one kilogramme of carbon in a gaseous form. Knowing the percentage of carbon in the coal, the amount lost in the ash, in soot and in tar, the results of a particular practice may be derived in terms of coal.

The amount of hydrogen in the gas for one kilogramme of carbon depends primarily upon the relation in the coal; but this proportion is changed by the loss of carbon in the ash, by the decomposition of steam, and by the loss of carbon and hydrogen in the soot and tar. The result is that the relative amount of hydrogen in the gas is from 10 to 50 per cent. higher than in the coal; this proportion may most conveniently be stated as the numerical value of the total carbon by weight, divided by the total hydrogen by weight in a given volume of the gas, and will be designated as C/H = carbon-ratio.

In gases made under the same general practice, this value indicates the special conditions under which a particular specimen is manufactured.

Sec. 5. *Determination of the Relation by Weight between Carbon and Hydrogen in a Mixed Gas.*—Every gas contains the same number of molecules in one unit of volume; hence, the figures which express percentage by volume, also express the number of molecules of each component in a total volume of 100 molecules. Knowing thus the relative number of molecules, we may calculate the relative number of atoms of any element, and, from the atomic weight, find the relative weight of each element. The following is an example of the work:

	(a)	(b)	(c)	(d)	(e)
CO_2, ...	5.2	5.2		62.4	
O, ...	0.4				
C_2H_4, . . .	0.4	0.8	1.6	9.6	1.6
CO, ...	22.8	22.8		273.6	
H, ...	8.5		17.0		17.0
CH_4, ...	2.4	2.4	9.6	28.8	9.6
N, ...	60.3				
				374.4	28.2

$$C/H = \frac{374.4}{28.2} = 13.3$$

(a) Composition by volume = number of molecules.
(b) Relative number of atoms of carbon.
(c) Relative number of atoms of hydrogen.
(d) Relative weight of atoms of carbon.
(e) Relative weight of atoms of hydrogen.

Sec. 6. *A Shorter Method for the Same Determination.*—It will be noticed that the relative weight of the atoms can be obtained directly by multiplication of the figures expressing the percentages by volume. The multiplier will be as follows for the different gases:

CO_2,	12		CO_2,	6
C_2H_4	C, 24; H, 4	This can be reduced to the simpler ratio:	C_2H_4,	C, 12; H, 2
CO,	12		CO,	6
CH_4,	C, 12; H, 4		CH_4,	C, 6; H, 2
H,	2		H,	1

Using this simpler method the work stands as follows:

CO_2,	5.2×6	31.2		
O,	0.4			
C_2H_4,	0.4×2			0.8
	$\times 12$	4.8		
CO,	22.8×6	136.8		
H,	8.5×1			8.5
	$\times 2$			4.8
CH_4,	2.4×6	14.4		
N,	60.3			
		187.2		14.1

$$C/H = \frac{187.2}{14.1} = 13\,3.$$

Sec. 7. *The Empirical Translation of these Relative Results into Gravimetric Terms.*—The method just described furnishes a short cut to the determination of relative amounts. It contains no factor of definite gravimetric value. It follows, however, from the necessities of the case, that the results of this system will always bear a definite relation to the results of gravimetric calculation. This relation may be found by comparison:

Carbon.

(a)	(b)	(c)	(d)	(e)	(f)	(g)
$1\,CO_2 \times$	6 =	6	$1.9666 \times$	$\frac{3}{11}$	$= 0.53635$	11.187
$1\,CH_4 \times$	6 =	6	$0.7155 \times$	$\frac{3}{4}$	$= 0.5366$	11.181
$1\,CO \times$	6 =	6	$1.2515 \times$	$\frac{3}{7}$	$= 0.53636$	11.187
$1\,C_2H_4 \times$	12 =	12	$1.2520 \times$	$\frac{6}{7}$	$= 1.07314$	11.182

Hydrogen.

(a)	(b)	(c)	(d)	(e)	(f)	(g)
$1\,CH_4 \times$	2 =	2	$0.7155 \times$	$\frac{1}{4}$	$= 0.1789$	11.179
$1\,C_2H_4 \times$	2 =	2	$1.2520 \times$	$\frac{1}{7}$	$= 0.1789$	11.179
$1\,H \times$	1 =	1	0.0896×1		$= 0\,0896$	11.161

(a) One volume of gas.
(b) Multiplier. See Section 6.
(c) Product. See Section 6.
(d) Weight of 1 cubic meter of gas in kilogrammes.
(e) Proportion by weight of carbon (in second table of hydrogen) in gas.
(f) Weight of carbon (in second table of hydrogen) in 1 cubic meter of gas.
(g) $\frac{c}{f}$ = relation between the relative and the gravimetric systems.

It will be seen that the relation is practically the same for all the gases which are found in producer-products, and the results obtained by the method given in Sections 6 and 7 represent the actual weight in kilogrammes of the carbon and hydrogen in multiples of 11.18 cubic meters of any mixed gas whose carbonaceous and hydrogenated components include no members except CH_4, C_2H_4, CO_2, CO and H.

The Producer.

Sec. 8. *The Calorific Efficiency of the Producer.*—Two methods are in common use as a means of comparing the economy of producer-practice: (1) the determination of the amount of CO_2 in the gas; and (2) the calculation of the calorific power per unit of volume. Practically, when running under regular conditions, either of these methods furnishes fairly comparable results. As a theoretical standard, each is deficient.

We must know not only the heating-power per unit of volume, but also the amount of coal used in the production of that volume and the amount of heat lost. In other words, we must have the relation between power sacrificed and power obtained.

The first factor in the problem is the loss of carbon in the ash, as pointed out in Section 4. If 10 per cent. of the coal used is removed from the grates in the form of carbon, the gas will contain all the hydrogen of the fuel and only about 90 per cent. of the carbon; the result will be a gas of higher heating-power than would be obtained under better practice; similarly, if a sample of gas be taken soon after a fresh supply of coal has been added, a high calorific value will be found, owing to the presence of an abnormal amount of hydrocarbons.

These two conditions will announce themselves in a decreased carbon-ratio. From these considerations it is plain that every specimen of gas used as a sample should represent a set of normal conditions; should not be a creation of temporary factors, but a chapter from a continuous history. Thus the calculation of a water-gas is not a measure of its economy; all the gases must be collected

which are evolved at all periods of the intermittent work, and the calorific value must be obtained, corresponding to 1 kilogramme of carbon. The steps of the work may be summarized as follows:

(1) Determination of weight of carbon per cubic meter.
(2) Number of cubic meters per kilo of carbon.
(3) Calorific value per cubic meter.
(4) Calorific value per kilo of carbon.

Having obtained these data, we may calculate the number of calories made in the producer in the combustion of carbon to CO and CO_2, the steps being as follows:

(1) Number of cubic meters of gas per kilo of carbon.
(2) Weight of CO and CO_2 per kilo of carbon.
(3) Weight of C burned to CO, and calories produced.
(4) Weight of C burned to CO_2, and calories produced.

SEC. 9. *Example of Determination of Carbon in Gas, and the Volume of Gas from* 1 *Kilo of Gasified Carbon.*

	(a)	(b)	(c)	(d)
CO_2,	5.2	1.97	10.24	2.79
O,	0.4	.....	.....	.....
C_2H_4,	0.4	1.25	0.50	0.43
CO,	22.8	1.25	28.50	12.21
H,	8.5	.....	.....	.....
CH_4,	2.4	0.72	1.73	1.30
N,	60.3			
Carbon in 100 c.m.,				16.73

(a) Composition by volume.
(b) Weight per cubic meter.
(c) Weight of component in 100 c.m. of gas.
(d) Weight of carbon in 100 c.m. of gas.

Carbon in 100 c.m. = 16.73 kilos.
Carbon in 1 c.m. = 0.1673 kilos.

$$\text{Volume of gas containing 1 kilo carbon} = \frac{1.0000}{0.1673} = 5.98 \text{ c.m.}$$

Calorific value per cubic meter, 1170 calories.
Calorific value per kilo of gasified carbon, $1170 \times 5.98 = 6997$ calories.

SEC. 10. *Example of Calculation of Heat Made in Producer per Kilo of Gasified Carbon.* (See data in Section 9.)

	(a)	(b)	(c)	(d)	(e)	(f)
To CO_2,	2.79	0.0279	5.98	0.1668	8080	1348
To CO,	12.21	0.1221	5.98	0.7301	2470	1803
Calories made in producer,						3151

(a) Carbon burned per 100 c.m. of gas. See Section 9.
(b) Carbon burned per 1 c.m. gas.
(c) Cubic meters of gas per kilo of carbon. See Section 9.
(d) Kilos of carbon burned to CO and CO_2 per kilo of gasified carbon.
(e) Calorific value of combustion.
(f) Calories produced per kilo of gasified carbon.

SEC. 11. *Averages of Gas-Analyses.*—It is evident from a consideration of the facts given in Section 8 that each particular gas should be considered by itself. It is the product of certain definite conditions among which may be mentioned:

(1) The loss of carbon in the ash.
(2) The thickness of the fire.
(3) The temperature of the fire.
(4) The amount of steam entering.

Even were these the only conditions involved it is certain that, theoretically, no average of gases would give an average of results. This truth may be stated mathematically by the axiom: The mean quotient is not the quotient of the mean.

If, however, the gases are made under practically the same conditions, and, within practical limits, have a similar composition, the method of finding the average gas by obtaining the average of each component will be very near the truth, and will tend to reduce to a minimum the errors of collection and determination. The following is a calculation showing the true equation of two different gases and the equation of the mean gas:

	CO_2	O	C_2H_4	CH_4	H	CO	N	Cal. val.	C/H
(1)	5.6	0.4	0.4	3.6	5.6	20.0	64.4	1110.2	13.18
(2)	4.0	0.4	0.4	4.4	5.3	23.6	61.9	1278.1	13.17
Av.	4.8	0.4	0.4	4.0	5.45	21.8	63.15	1194.2	.....

Calculating the calorific efficiency per kilogramme of gasified carbon, we have:

	Kilo C per c.m.	Kilo H per c.m.	C.m. from 1 kilo C.	Cal. value.	Calories per kilo C.
(1)	0.16097	0.01221	6.212	1110	6.895
(2)	0.17601	0.01336	5.682	1278	7.262
Average, ...					7.079

The average gas gives as follows:

0.16849	0.01279	5.935	1194	7.086

These gases differ as widely as is usual with specimens made under

regular conditions, and the error caused by the incorrect system of averages is found to be only 7 calories per kilogramme, or about one-tenth of 1 per cent. of the total value.

SEC. 12. *A Comparison of Two Methods of Making Gas.*—Coal used: Carbon, 82 per cent.; hydrogen, 4.7 per cent.

Method A.—Open grates, no steam in blast. Loss of carbon in ash = 20 per cent. of weight of coal; hence carbon gasified = 62 per cent. of weight of coal. Loss of potential heat in ash per kilo of coal:

$$0.20 \times 8080 = 1616 \text{ calories.}$$

Method B.—Open grates, steam-jet in blast. Loss of carbon in ash = 6 per cent. of weight of coal; hence carbon gasified = 76 per cent of weight of coal. Loss of potential heat in ash per kilo of coal:

$$0.06 \times 8080 = 485 \text{ calories.}$$

SEC. 13. *General Samples of Gas Made by Method A:*

	CO_2	O	C_2H_4	CO	H	CH_4	N	Cal. val.	C/H
(1), .	3.6	0.4	0.0	24.8	8.5	4.0	58.7	1309	11.8
(2), .	4.8	0.4	0.2	21.8	6.0	5.2	61.6	1283	11.5
(3), .	5.6	0.4	0.4	20.8	7.2	3.0	62.6	1125	12.9
(4), .	5.6	0.4	0.4	20.0	5.6	3.6	64.4	1110	13.2
(5), .	5.2	0.4	0.4	21.6	7.2	3.2	62.0	1137	12.8
(6), .	5.2	0.4	0.4	22.0	6.0	3.0	63.0	1130	14.3
(7), .	4.4	0.4	0.4	23.4	8.0	3.4	60.0	1259	12.3
(8), .	4.4	0.4	0.4	22.8	5.7	3.4	62.9	1180	14.2
(9), .	4.0	0.4	0.4	23.6	5.3	4.4	61.9	1278	13.2
(10), .	5.6	0.4	0.4	20.2	8.5	4.2	60.7	1244	10.4
Av.,	4.84	0.4	0.34	22.1	6.8	3.74	61.78	1209	

C/H of average gas = 12.58.

SEC. 14. *General Samples of Gas Made by Method B:*

	CO_2	O	C_2H_4	CO	H	CH_4	N	Cal. val.	C/H
(11), .	5.2	0.6	0.4	24.0	6.9	2.2	60.7	1146	16.0
(12), .	5.2	0.4	0.4	22.8	8.5	2.4	60.3	1170	13.3
(13), .	5.2	0.4	0.4	23.2	8.5	2.8	59.5	1216	12.9
(14), .	5.0	0.4	0.2	24.0	8.9	2.4	59.1	1188	13 5
(15), .	5.4	0.6	0.4	22.2	8.3	2.6	60.5	1163	13.0
(16), .	6.0	0.6	0.4	21.0	8.0	3.4	60.6	1187	12.0
(17), .	5.2	0.8	0.4	24.0	9.8	2.6	57.2	1258	12.4
(18), .	5.2	0.4	0.2	21.8	8.0	2.4	62.0	1098	13.5
(19), .	4.6	0.4	0.4	23.6	8.7	2.4	59.9	1200	13.2
(20), .	6.0	0.8	0.4	20.8	8.1	2.4	61.5	1099	13.1
Av.,	5.3	0.54	0.36	22.74	8.37	2.56	60.13	1172	

C/H of average gas = 13.22.

SEC. 15. *Comparison of Average Gases Made by Methods A and B:*

	A.	B.
(a) Carbon in one cubic meter,	.1683	.1680
(b) Cubic meters of gas per kilo of gasified carbon (1 ÷ a),	5.94	5.95
(c) Calorific value per cubic meter,	1209	1172
(d) Calorific value per kilo of gasified carbon (b × c),	7181	6973
(e) Calories lost in production of CO_2, } See Sections 9 and	1248	1374
(f) Calories lost in production of CO, } See Sections 9 and	1760	1791
(g) Total calories made in producer, } 10 for method	3008	3165
(h) Total calories in gas (d + g),	10,189	10,138
(k) Per cent. in producer (g ÷ h),	29.5	31.2
(l) Per cent. in gas (d ÷ h),	70.5	68.8
(m) Hydrogen per kilo of gasified carbon,	0.0795	0.0756
(n) Hydrogen per kilo of coal,	0.0493	0.0575
(o) Hydrogen in one kilo of coal, by analysis,	0.047	0.047
(p) Hydrogen from air and steam (n — o),	0.0023	0.0105
(r) Per cent. hydrogen from air and steam (p ÷ n),	4.5	18.3
(s) Cubic meters of gas per kilo of coal (See Section 12), { 62 per cent. of b,	3.68	
(s) Cubic meters of gas per kilo of coal (See Section 12), { 76 per cent. of b,		4.52
(t) Calories in producer per kilo of coal, { 62 per cent. of g,	1865	
(t) Calories in producer per kilo of coal, { 76 per cent. of g,		2405
(u) Calories lost in ash per kilo of coal (See Section 12),	1616	485
(v) Calories in gas per kilo of coal, { 62 per cent. of d,	4452	
(v) Calories in gas per kilo of coal, { 76 " " " d,		5299
(w) Total calories per kilo of coal (t + u + v),	7933	8189
(x) Per cent. in ash (u ÷ w),	20.4	5.9
(y) Per cent. in producer (t ÷ w),	23.5	29.4
(z) Per cent. in gas (v ÷ w),	56.1	64.7

SEC. 16. *The Loss of Thermal Power in the Producer.*—If we assume that, for gases made under the same general conditions, the loss of carbon in the ash is the same, we may compare the various samples on the basis of 1 kilo of gasified carbon, without reducing to terms of coal.

The following is such a comparison, with samples arranged according to the percentage lost in the producer. Each set is divided into two groups, and averages are taken in order that a comparison of the higher and lower classes may be made. It will be noticed that, as a rule, economy of practice is portrayed by a low percentage of CO_2. Samples varying from this law, will be found to have a low carbon-ratio, showing an abnormal amount of hydrogen which makes itself felt in the calorific value.

SEC. 17. *Gases Made by Method A, Compared:*

	(a)	(b)	(c)	(d)	(e)	(f)	(g)	(h)	(i)	(k)	(l)
(1),	3.6	11.8	898	1890	2788	1309	.1738	5.75	7532	10,320	27.0
(9),	4.0	13.2	979	1769	2748	1278	.1769	5.65	7225	9,973	27.6
(2),	4.8	11.5	1202	1669	2871	1283	.1730	5.78	7416	10,287	27.9
(7),	4.4	12.3	1110	1808	2918	1259	.1717	5.82	7332	10,250	28.5
(10),	5.6	10.4	1468	1619	3087	1244	.1654	6.05	7521	10,608	29.1
Av., .	4.48	11.84									28.02

	(a)	(b)	(c)	(d)	(e)	(f)	(g)	(h)	(i)	(k)	(l)
(8),	4.4	14.1	1131	1794	2925	1180	.1686	5.93	6999	9,924	29.5
(5),	5.2	12.8	1362	1731	3093	1167	.1653	6.05	7060	10,153	30.5
(4),	5.6	13.2	1505	1644	3149	1110	.1610	6.21	6894	10,043	31.35
(6),	5.2	14.3	1356	1753	3109	1130	.1662	6.02	6799	9,908	31.38
(3),	5.6	12.9	1498	1702	3200	1125	.1621	6.17	6940	10,140	31.6
Av., .	5.2	13.46									30 88

(a) Percentage of CO_2 in the gas, by volume.
(b) Carbon-ratio.
(c) Calories produced by combustion of C to CO_2 in the producer.
(d) Calories produced by combustion of C to CO in the producer.
(e) Total calories made in the producer.
(f) Calorific value of gas per cubic meter.
(g) Carbon per cubic meter.
(h) Cubic meters per kilo of carbon in the gas.
(i) Calorific value per kilo of carbon in the gas.
(k) Total calories in the producer and in the gas.
(l) Per cent. in the producer.

SEC. 18. *Gases Made by Method B, Compared:*

	(a)	(b)	(c)	(d)	(e)	(f)	(g)	(h)	(i)	(k)	(l)
(19),	4.6	13.2	1188	1855	3043	1200	.1684	5.94	7128	10,171	29.9
(17),	5.2	12.4	1293	1818	3111	1258	.1749	5.72	7196	10,307	30.2
(13),	5.2	12.9	1309	1786	3095	1216	.1717	5.82	7077	10,172	30.4
(12),	5.2	13.3	1349	1803	3152	1170	.1673	5.98	6997	10,149	31.1
(14),	5.0	13.5	1275	1862	3137	1188	.1707	5 86	6962	10,099	31.1
Av., .	5.04	13.06									30.54

	(a)	(b)	(c)	(d)	(e)	(f)	(g)	(h)	(i)	(k)	(l)
(15),	5.4	13.0	1406	1769	3175	1163	.1662	6.02	7001	10,176	31.2
(16),	6.0	12.0	1559	1662	3221	1187	.1674	5.98	7098	10,319	31.2
(18),	5.2	13.5	1401	1803	3204	1098	.1599	6.25	6863	10,067	31.8
(20),	6.0	13.1	1616	1712	3328	1099	.1609	6.22	6836	10,164	32.7
(11),	5.2	16.0	1301	1840	3141	1146	.1727	5.79	6635	9,776	33.3
Av., .	5.56	13.5									32.04

SEC. 19. *The "Standard" Gas.*—In the subsequent calculations there will be taken, as a basis, the results of Method B; the gas taken

will be number (12), as being an actual sample and practically coincident, in composition and calorific power, with the average of the ten gases made by this method. This gas, it will be noticed, has been used in the foregoing examples of calculations wherever possible. In the following pages it will be designated by the word "Standard." The following are its factors:

Carbon in 1 cubic meter of gas,	0.1673
Cubic meters of gas per kilo of gasified carbon,	5.98
Calorific value per cubic meter,	1170
Calorific value per kilo of gasified carbon,	6997
Calories lost in production of CO_2,	1349
Calories lost in production of CO,	1803
Total calories made in the producer,	3152
Total calories in the gas,	10,149
Per cent. calories in the producer,	31.1
Per cent. calories in the gas,	68.9
Hydrogen per kilo of gasified carbon,	0.0752
Hydrogen per kilo of coal,	0.0572
Hydrogen in one kilo of coal,	0.047
Hydrogen from air and steam,	0.010
Per cent. hydrogen from air and steam,	18
Cubic meters of gas per kilo of coal,	4.55
Calories lost in ash per kilo of coal,	485
Calories in producer from combustion of one kilo of coal,	2396
Calories in gas per kilo of coal,	5317
Total calories per kilo of coal,	8198
Per cent. calories in the ash,	5.9
Per cent. calories in the producer,	29.2
Per cent. calories in the gas,	64.9

SEC. 20. *The Equation of the Producer.*—The loss of heat in the producer, resulting from the combustion of carbon, is made up of several factors, among which the following present themselves:

(1) Loss in ashes, exclusive of the carbon they contain.
(2) Latent heat of volatilization.
(3) Sensible heat of gas evolved.
(4) Losses due to vapor of water.
  (a) Evaporation of water in coal, and heating of resultant steam.
  (b) Heating of the steam that passes through the fire undecomposed.
  (c) Decomposition of steam.
(5) Radiation and conduction.

(1) The heat of the ashes is nearly all utilized while lying in the lower part of the fire and in the ash-pit, in heating the incoming air and moisture; the amount of heat actually carried away as specific heat may be neglected.

(2) The latent heat of volatilization of the hydrocarbons is given by Bell as 600,000 calories per ton of coal, or 600 calories per kilogramme.

(3) The volume of gas we have found to be 4.56 c. m. per kilo of coal. The temperature may be assumed at 750° C. The specific heat of the gas, 0.317. (See Section 2.)

The amount of heat carried away as sensible heat of the gas will therefore be

$$4.56 \times 0.317 \times 750 = 1085 \text{ calories.}$$

(4a) Evaporation of moisture in coal, and heating of resultant steam.

Coal, 2 per cent. moisture.
1 kilo of coal therefore contains 0.02 kilo H_2O, which will require for:

	Calories.
Heating from 30° C. to 100 C.,	1.4
Latent heat of vaporization, $537 \times .02$,	10.7
Heating steam from 100° C. to 750° C. $= 650 \times 0.475 \times .02$,	6.2
Total heat absorbed, due to moisture in coal,	18.3

(4b) Heat absorbed by undecomposed steam.

Determinations of moisture in the gas from a fire which had evidently driven off all moisture from the coal gave the following results:

First determination,	27.51	grms.	H_2O to	1 c.m.
Second "	24.90	"	"	"
Third "	28.50	"	"	"
Average,	26.97	"	"	"

One kilo of coal = 4.56 c.m. of gas.
Moisture to 1 kilo of coal = 123 grms.
Heat required to raise this amount from 100° C. to 750° C. =

$$650 \times 0.475 \times 0.123 = 38 \text{ calories.}$$

(4c) Decomposition of steam.

According to Sec. 19 the hydrogen derived from air and steam equalled 0.01 kilo per kilo of coal. The heat absorbed by dissociation equals the heat produced by the combination; hence, the heat absorbed equals $0.01 \times 30{,}000 = 300$ calories.

The absorption of heat due to vapor of water amounts therefore to the following:

	Calories.
(a) Evaporation of water, etc.,	18
(b) Undecomposed steam,	38
(c) Decomposition of steam,	300
Total,	356

The equation of the producer, therefore, stands as follows, per kilo of coal:

	Calories.
Latent heat of volatilization, ...	600
Sensible heat of the gas, ...	1085
Losses due to water and steam, ...	356
Radiation and conduction (by difference), ...	355
Total as by Section 19, ...	2396

SEC. 21. *The Decomposition of Steam.*—Experiments were made to determine the amount of steam decomposed when, with an open grate, the draught was saturated with steam from a jet. A fire was selected which was at a dull red temperature on top and free from holes. All samples of gas were taken through a space of five minutes to insure an average result; the fire was stirred only when necessary to close an incipient hole.

The first sample was taken with the steam on; the second without steam, an interval being allowed for the ash to dry; the third, with steam.

Under regular conditions it is reasonable to suppose the second sample would be an average of the first and third; under the above conditions, we may ascribe the variation from this average to the change in the moisture supplied. This system of averages supposes that the normal diminution of hydrocarbons in the gas during regular intervals may be represented by a straight line. The following are the results obtained:

SEC. 22. *The Experiments:*

	CO_2	O	C_2H_4	CO	H	CH_4	N	C H
First determination:								
(a) With steam, . .	4.8	0.4	0.4	23.6	9.5	2.8	58.5	12.08
(b) Without steam, .	4.6	0.4	0.4	22.2	9.1	2.4	60.9	12.25
(c) With steam, . .	4.0	0.4	0.2	24.6	9.9	2.0	58.9	13.01
Second determination:								
(a) With steam, . .	3.6	0.4	0.4	25.4	9.9	4.3	56.0	10.52
(b) Without steam, .	4.0	0.4	0.4	23.0	6.7	3.2	62.3	13.38
(c) With steam, . .	3.6	0.4	0.4	24.8	8.5	3.2	59.1	12.38
Third determination:								
(a) With steam, . .	3.6	0.4	0.4	24.8	12.1	4.2	54.5	9.40
(b) Without steam, .	3.6	0.4	0.4	23.4	9.1	3.6	59.5	11.02
(c) With steam, . .	4.0	0.4	0.4	24.8	10.4	3.4	56.6	11.00

SEC. 23. *Calorific Efficiency of the Samples:*

	(m)	(n)	(o)	(p)	(r)
First determination:					
(a) With steam, . .	.1719	5.817	0.0832	1254	7295
(b) Without steam, .	.1613	6.200	0.0818	1168	7242
(c) With steam, . .	.1673	5.978	0.0771	1199	7168

Second determination:					
(a) With steam, . .	.1835	5.450	0.0954	1454	7924
(b) Without steam, .	.1660	6.024	0.0753	1196	7205
(c) With steam, . .	1739	5.750	0.0817	1297	7458
Third determination:					
(a) With steam, . .	.1792	5.580	0.1064	1478	8247
(b) Without steam, .	.1689	5.921	0.0907	1259	7455
(c) With steam, . .	.1721	5.647	0.0909	1321	7460

(m) Kilos of carbon in 1 cubic meter of gas.
(n) Cubic meters of gas from 1 kilo of gasified carbon.
(o) Hydrogen per kilo of gasified carbon.
(p) Calorific value of gas per cubic meter.
(r) Calorific value per kilo of gasified carbon.

SEC. 24. *Average of Results.*—Calculating the results as described in Section 21 for the hydrogen per kilo of gasified carbon, we have:

	1st det.	2d det.	3d det.
Average two steams (a and c), . .	0.0801	0.0886	0.0987
Without steam (b), ...	0.0818	0.0753	0.0907

The average of the three experiments gives the following:

Average of those with steam, ...	0.0891
Average of those without steam, ...	0.0826
Gain with steam, ...	0.0065

In these experiments, therefore, the hydrogen from the decomposition of the steam amounted to 8 per cent. of the total hydrogen present. The calorific efficiency is greater in the case of the samples using steam, but a comparison is invalid. A continuation of the dry air might cause an increase in temperature in the lower region, giving a decrease in CO_2 and a consequent economy of practice. The necessary additions of coal from time to time, and the constant changes in the composition of the fire, render impracticable any direct experiments upon this point. Some information may be gained from the calculations in Section 15 *et seq.*

SEC. 25. *Moisture Passing Undecomposed Through the Fire.*—In the preceding sections no mention has been made of the moisture passing through the fire; determinations were made of the vapor found in the gases from fires with and without steam. In each case fires were selected which appeared to be in the same and in a normal condition; three determinations were made as follows:

	Grammes of H_2O in one cubic meter,	
	With steam.	Without steam.
First experiment, ...	27.51	24.84
Second experiment, ...	24.90	20.93
Third experiment, ...	28.50	29.00
Average, ...	26.97	24.92

Within the limits of error, therefore, the amount of moisture passing undecomposed through the fire is the same, whether dry or saturated air is used in combustion.

The Combustion of Gas—Theoretical.

Sec. 26. *The Combustion of Producer-Gas.*—The combustion of the components of ordinary producer-gas may be represented by the following formulæ:

$$C_2H_4 + 6O = 2CO_2 + 2H_2O$$
$$CH_4 + 4O = CO_2 + 2H_2O$$
$$2H + O = H_2O$$
$$CO + O = CO_2$$

It is necessary, also, to know in what proportions they combine, by weight and by volume. The gravimetric relation can be determined from the atomic weights. The volumetric ratio is found by dividing the relative weight of each by its specific gravity. The following is a calculation for each of the above gases:

	C_2H_4	+	$6O$	=	$2CO_2$	+	$2H_2O$.
Relative weights, . . .	28	+	96	=	88	+	36
Ratio by weight, . . .	1	+	$\frac{24}{7}$	=	$\frac{22}{7}$	+	$\frac{9}{7}$
Specific gravity, . . .	1.25		1.43		1.97		0.80
Relative volume, . . .	0.8		2.4		1.6		1.6
Ratio by volume, . . .	1 c.m.		3 c.m.		2 c.m.		2 c.m.

	CH_4	+	$4O$	=	CO_2	+	$2H_2O$.
Relative weights, . . .	16	+	64	=	44	+	36
Ratio by weight, . . .	1	+	4	=	$\frac{11}{4}$	+	$\frac{9}{4}$
Specific gravity, . . .	0.716		1.43		1.97		0.80
Relative volume, . . .	1.4		2.8		1.4		2.8
Ratio by volume, . . .	1 c.m.		2 c.m.		1 c.m.		2 c.m.

	$2H$	+	O	=	H_2O.
Relative weights, ...	2	+	16	=	18
Ratio by weight, ...	1	+	8	=	9
Specific gravity, ...	0.0896		1.43		0.80
Relative volume, ...	11.2		5.6		11.2
Ratio by volume, ...	2 c.m.		1 c.m.		2 c.m.

	CO	+	O	=	CO_2.
Relative weights,	28	+	16	=	44
Ratio by weight,	1	+	$\frac{4}{7}$	=	$\frac{11}{7}$
Specific gravity,	1.25		1.43		1.97
Relative volume,	0.8		0.4		0.8
Ratio by volume,	2 c.m.		1 c.m.		2 c.m.

Sec. 27. *Tabulation of the Foregoing.* — These results may be tabulated as follows:

	By weight.		
	Kilos O required.	Kilos CO_2 formed.	Kilos H_2O formed.
1 kilo C_2H_4,	$\frac{24}{7}$	$\frac{22}{7}$	$\frac{2}{7}$
1 kilo CH_4,	4	$\frac{11}{4}$	$\frac{2}{4}$
1 kilo H,	8	—	9
1 kilo CO,	$\frac{4}{7}$	$\frac{11}{7}$	—

	By volume.		
	Cubic meters O required.	Cubic meters CO_2 formed.	Cubic meters H_2O formed.
1 c.m. C_2H_4,	3	2	2
1 c.m. CH_4,	2	1	2
1 c.m. H,	0.5	—	1
1 c.m. CO,	0.5	1	—

Sec. 28. *Examples of Combustion.*—The following are examples, showing the calculation of the combustion of certain gases:

(1) "Standard." By weight and by volume (see Sec. 19).

(2) A gas of high carbon-ratio. Gas No. 11 in tables, Sec. 14.

(3) A gas of low carbon-ratio. Actual sample taken.

(4) Marsh-gas.

(5) Carbonic oxide.

(6) Hydrogen.

(1) "Standard" (Sec. 19), by weight:

	(a)	(b)	(c)	CO_2.	H_2O.	N.
CO_2,	5.2	10.24	—	10.2	—	—
O,	0.4	0.57	—	—	—	—
C_2H_4,	0.4	0.50	1.71	1.6	0.6	
CO,	22.8	28.50	16.29	44.8	—	—
H,	8.5	0.77	6.16		6.9	—
CH_4,	2.4	1.73	6.92	4.8	3.9	—
N,	60.3	75.98				76.0
Oxygen necessary.			31.08			
Oxygen present,			0.57			
O needed from air,			30.51			
N from air, 3.31×30.51,						101.0
H_2O from air, 0.01×131.5,					1.3	—
				61.4	12.7	177.0

(a) Percentage by volume.
(b) Weight in 100 c.m. of gas, in kilogrammes.
(c) Oxygen required, in kilogrammes.

100 c.m. gas require 131.5 kilos air = 101.6 c.m.

giving { CO_2 — 61.4 kilos = 31.2 c.m.
{ H_2O — 12.7 kilos = 15.9 c.m.
{ N — 177.0 kilos = 140.5 c.m.

(1) "Standard" (Sec. 19), by volume:

	(a)	(b)	CO_2.	H_2O.	N.
CO_2,	5.2	—	5.2	—	—
O,	0.4	—	—	—	—
C_2H_4,	0.4	1.2	0.8	0.8	—
CO,	22.8	11.4	22.8	—	—
H,	8.5	4.3	—	8.5	—
CH_4,	2.4	4.8	2.4	4.8	—
N,	60.3	—			60.3
O necessary,		21.7			
O present,		0.4			
		—			
O from air,		21.3			
N from air, 3.77×21.3,					80.3
H_2O from air, 0.015×101.6,				1.5	
			31.2	15.6	140.6

(a) Percentage by volume.
(b) Cubic meters of oxygen required for 100 c.m. gas.

100 c.m. gas require 101.6 c.m. air,

giving { CO_2 — 31.2 c.m.
{ H_2O — 15.6 c.m.
{ N — 140.6 c.m.

In the same way we have:

(2) A gas of high carbon-ratio:

CO_2,	5.2
O,	0.6
C_2H_4,	0.4
CO,	24.0
H,	6.9
CH_4,	2.2
N,	60.7

100 c.m. gas require 98.4 c.m. air,

giving { CO_2 — 32.2 c.m.
{ H_2O — 13.6 c.m.
{ N — 138.0 c.m.

(3) A gas of low carbon-ratio:

CO_2,	2.6
O,	0.4
C_2H_4,	0.0
CO,	25.8
H,	12.2
CH_4,	3.2
N,	55.8

106 c.m. gas require 121.2 c.m. air,

giving { CO_2 — 31.6 c.m.
{ H_2O — 20.4 c.m.
{ N — 151.6 c.m.

(4) Marsh-gas:

CH_4 . . . . 100.0 — 100 c.m. gas require 954 c.m. air, giving { CO_2 — 100 c.m. / H_2O — 200 c.m. / 7N — 54 c.m.

(5) Carbonic oxide:

CO, ... 100.0 — 100 c. m. gas require 238.5 c.m. air, giving { CO_2 — 100 c.m. / N —188.5 c.m.

(6) Hydrogen:

H, ... 100.0 — 100 c.m. gas require 233.5 c.m. air, giving { H_2O — 100 c.m. / N — 188.5 c.m

SEC. 29. *A Comparison of the Foregoing Gases, with Relation to the Latent Heat of their Products of Combustion.*—In the comparison of the practical values of different gases, it is necessary to know the percentage of heat lost in the products of combustion when these escape at a higher temperature than the incoming gases. For this purpose there should be taken, not an equal volume nor an equal weight, but such a volume of each as will give an equal calorific power. The volume and composition of the products of combustion can then be determined and the amount and percentage of heat which is carried away for every degree in temperature. Taking as a basis the volume necessary to produce 1000 calories, we have as follows:

Gas.	"Standard"	(2)	(3)	(4)	(5)	(6)
Calorific power per cubic meter, .	1170	1146	1370	8480	3000	2660
Cubic meters to produce 1000 calories,	0.855	0.873	0.730	0.118	0.333	0.376

Calculating the products of combustion for these volumes which are required to produce 1000 calories, and taking the specific heat of CO_2, H_2O and N from Section 1, we have the following loss in the products for every 100° C.

	"Standard" (Sec. 19.)			(2)		
	Prod. combustion.		Absorbed.	Prod. combustion.		Absorbed.
	1 c.m.	0.855 c.m.	100° C.	1 c.m.	0.873 c.m.	100° C.
CO_2, .	0.312	0.267	11.37	0.322	0.281	11.97
H_2O, .	0.156	0.133	5.08	0.136	0.119	4.55
N, .	1.406	1.202	36.90	1.380	1.205	36.99
			53.35			53.51

	(3)			(4)		
	Prod. combustion.		Absorbed.	Prod. combustion,		Absorbed.
	1 c.m.	0.730 c.m.	100° C.	1 c.m.	0.118 c.m.	100° C.
CO_2, .	0.316	0.231	9.84	1.000	0.118	5.03
H_2O, .	0.204	0.149	5.69	2.000	0.236	9.02
N, .	1.516	1.107	33.98	7.540	0.890	27.32
			49.51			41.37

	(5)			(6)		
	Prod. combustion.		Absorbed.	Prod. combustion.		Absorbed.
	1 c.m.	0.333 c.m.	100° C.	1 c.m.	0.376 c.m.	100° C.
CO_2, .	1.000	0.333	14.2			
H_2O, .				1.000	0.376	14.36
N, .	1.885	0.628	19.3	1.885	0.709	26.66
			33.5			41.02

Sec. 30. *Summary of the Foregoing.*—Thus, for every 1000 calories produced, we have the following loss:

	Loss per 100° C		300° C.
	Calories.	Per cent.	Per cent.
(1) "Standard" (Sec. 19), . .	53.35	5.335	16 0
(2) Producer-gas, ...	53.51	5.351	16.1
(3) Producer-gas, ...	49.51	4.951	14.9
(4) CH_4, ...	41.37	4.137	12.4
(5) CO, ...	33.50	3.350	10.1
(6) H, ...	41.02	4.102	12.3

The Combustion of Gas—Practical.

Sec. 31. *The Composition of Products of Combustion as Determined by Ordinary Analysis.*—In the analysis of gases, under ordinary conditions, the steam is condensed and does not appear in the results as expressed in composition by volume; for the sake of comparison with analytical determinations, the theoretical data should be given upon the same basis. Omitting the H_2O in the results given in Section 29, we have the following volumes of the products of combustion in the creation of 1000 calories with the theoretical amount of air:

Composition by Volume.

	"Standard" (Sec. 19.)		(2)		(3)	
	C.m.	Per cent.	C.m.	Per cent.	C.m.	Per cent.
CO_2, . . .	0.267	18.2	0.281	18.9	0.231	17.3
N, . . .	1.202	81.8	1.205	81.1	1.107	82.7
	1.469	100.0	1.486	100.0	1.338	100.0

	(4)		(5)		(6)	
	C.m.	Per cent.	C.m.	Per cent.	C.m.	Per cent
CO_2, . . .	0.118	11.7	0.333	34.6	.....	.....
N, . . .	0.890	88.3	0.628	65.4	0.709	100.0
	1 008	100.0	0.961	100.0	0.709	100.0

Sec. 32. *Excess of Air in Combustion.*—In ordinary producer-gas there will usually be found about four-tenths of one per cent. of oxygen by volume. This is not altogether from the air remaining in the tube in which the sample is taken, but is actually present in the gas from the fire, and is probably due to the cessation of combustion when the oxygen is diluted to that degree.

If such can be the case with an enormous excess of combustible matter, it may be readily seen that to insure complete combustion of gas under ordinary circumstances a certain excess of oxygen must be present in the escaping gases. With the gas at 600° C. and the air at 50° C., from 20 to 100 per cent. of air in excess will be necessary to prevent the escape of an appreciable amount of combustible material. With the air and gas at 1000° C., 10 per cent. is a large excess and 5 per cent. is sufficient for ordinary producer-gas. These facts give rise to the following problems:

(a) Excess of air:
  (1) The composition of the products of combustion with different amounts in excess.
  (2) The heat absorbed thereby.

(b) Deficiency of air:
  (1) The composition of the products of combustion with different amounts of combustible matter escaping.
  (2) The potential heat lost thereby.

It will be seen on examination of Section 31 that the analyses and volumes of the products of combustion of the producer-gases (1), (2) and (3) are practically identical; in the following calculations, therefore, the only gases considered will be:

(1) As illustrating producer-gases in general.
(4) As illustrating natural gas.

Sec. 33. *Excess of Air—Composition of Products by Volume:*

"Standard" (Sec. 19).
One c.m. requires 1.016 c.m. air. (See Section 28.)
Ten per cent. excess of air = 0.1016 c.m. air per cubic meter of gas.

$$0.1016 \text{ c.m. air} = \begin{cases} \text{N} - 0.0803 \text{ c.m.} \\ \text{O} - 0.0213 \text{ c.m.} \end{cases}$$

		C.m.	Per cent.
Products, no excess (See Section 23)	CO_2 — 0.312	0.312	17.15
	N — 1.406		
Ten per cent. excess (see above),	N — 0.080	1.486	81.70
	O — 0.021	0.021	1.15
		1.819	100.0
Products, no excess of air, . .	CO_2 — 0.312	0.312	16.25
	N — 1.406		
Twenty per cent. excess of air, .	N — 0.160	1.566	81.56
	O — 0.042	0.042	2.19
		1.920	100.00

SEC. 34. *Excess of Air—Composition of Products by Volume:*

Gas (4), CH_4, natural gas.
One c.m. requires 9.54 c.m. air. (See Section 28.)
Ten per cent. excess of air = 0.954 c.m. air.

0.954 c m. air = N — 0.754 c.m.; O — 0.200 c.m.

Products, no excess (See Section 28)	CO_2 — 1.000	1.000	10.53
	N — 7.540	8.294	87.36
Ten per cent. excess (see above),	N — 0.754		
	O — 0.200	0.200	2.11
		9.494	100.00
Products, no excess of air, . .	CO_2 — 1.000	1.000	9.57
	N — 7.540	9.048	86.60
Twenty per cent. excess of air, .	N — 1.508		
	O — 0.400	0.400	3.83
		10.448	100.00

SEC. 35. *Excess of Air—Composition by Volume.*—It will be noticed that an excess of air does not increase the percentage of nitrogen in the waste gases. This is due to the theoretical percentage of nitrogen being higher than it is in air. This condition arises from the fact that both nitrogen and oxygen have been supplied as air to burn the hydrogen of the gas, while, owing to the condensation of the steam, only the nitrogen appears in the analysis.

The true guide by which to calculate the excess of air present is the content of oxygen; by comparing the results of analysis with the calculated composition, the excess of air may be ascertained.

SEC. 36. *Loss of Heat from Excess of Air.*—From the results obtained in the preceding section, and those in Section 29, we may find

the percentage of heat lost on account of the excess of air, when the products escape at a higher temperature than the incoming gases.

	"Standard" (Sec. 19).	Gas (4).
Air required for 1 cubic meter of gas,	1.016	9.540
Air for production of 1000 calories,	0.869	1.126
Ten per cent. excess of air,	0.0869	0.1126
Specific heat of air = 0.307.	—	—
Heat absorbed per 1° C. by 0.0869 c.m. air,	0.0267	—
Heat absorbed per 1° C. by 0.1126 c.m. air,	—	0.0346
Heat absorbed per 100° C.,	2.67	3.46
Per cent. of heat lost per 100° C.,	0.267	0.346

That is, if the products of combustion escape at a temperature 100° C. above that of the incoming gases, the heat lost, due to the presence of 10 per cent. excess of air, is less than one-half of one per cent. of the total heat produced.

Sec. 37. *Deficiency of Air.*—If the amount of air supplied be insufficient, the escaping gases will contain amounts of CO, H or CH_4. This loss may be equated most easily by assuming a certain loss of calorific power in the form of these gases and calculating the resultant products. Assuming a loss of 5 per cent. of the total calorific power in the case of each of these gases, the following calculations obtain :

	"Standard" (Sec. 19).	Gas (4).
(a) Calorific value per 100 c.m.,	117.000	848.000
(b) Five per cent. of (a),	5,800	42,400
(c) C.m. CO necessary to produce (b),	1.9	14.1
(d) " H " " "	2.2	15.9
(e) " CH_4 " " "	0.7	5.0

In Section 28, the products formed by the complete combustion of (1) and (4) have been given. It is only necessary to find the products resulting from the perfect combustion of the amounts of CO, H and CH_4 given in (c), (d) and (e), and to subtract these products from the total products, in order to find the results of the incomplete combustion. The following is the calculation :

Products of combustion of gases in (c), (d) and (e).

		Air.		Products.		
"Standard" (Sec. 19).	C.m. gas.	O	N	CO_2	H_2O	N
CO,	1.9	0.95	3.6	1.9	.....	3.6
H,	2.2	1.1	4.2	.....	2.2	4.2
CH_4,	0.7	2.1	7.9	1.4	1.4	7.9
Gas (4).						
CO,	14.1	7.05	26.6	14.1	.....	26.6
H,	15.9	7.95	30.0	.....	15.9	30.0
CH_4,	5.0	15.0	56.6	10.0	10.0	56.6

From these we are enabled to make the following calculation:

Column (a) = c.m. of products of perfect combustion of 100 c.m. gas.
(b) = products of (c), (d) and (e) as above.
(c) = difference between (a) and (b), neglecting H_2O and adding the amount of combustible matter assumed as escaping, giving the products of imperfect combustion under the assumed conditions.
(d) = composition of same in percentage by volume.

SEC. 38. "*Standard*" (Sec. 19); *Combustible Material Escaping:*

100 c.m. gas. 1.9 c.m. CO escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	31.2	1.9	29.3	17.42
H_2O,	15.6			
N,	140.6	3.6	137.0	81.45
CO,			1.9	1.13
			168.2	100.00

100 c.m. gas. 2.2 c.m. H escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	31.2		31.2	18.37
H_2O,	15.6	2.2		
N,	140.6	4.2	136.4	80.33
H,			2.2	1.30
			169.8	100.00

100 c.m. gas. 0.7 c.m. CH_4 escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	31.2	0.7	30.5	18 32
H_2O,	15.6	1.4		
N,	140.6	5.3	135.3	81.26
CH_4,			0.7	0.42
			166.5	100.00

SEC. 39. *Gas* (4), *Combustible Material Escaping:*

100 c.m. gas. 14.1 c.m. CO escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	100.0	14.1	85.9	10.38
H_2O,	200.0			
N,	754.0	26.6	727.4	87.92
CO,			14.1	1.70
			827.4	100.00

100 c.m. gas. 15.9 c.m. H escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	100.0		100.0	11.91
H_2O,	200.0	15.9		
N,	754.0	30.0	724.0	86.20
H,			15.9	1.89
			839.9	100.00

100 c.m. gas. 5.0 c.m. CH_4 escaping unburned.

	(a)	(b)	(c)	(d)
CO_2,	100.0	5.0	95.0	11.64
H_2O,	200.0	10.0		
N,	754.0	37.7	716.3	87.75
CH_4,			5.0	0.61
			816.3	100.00

SEC. 40. *Summary of Results of Sections* 37, 38 *and* 39.—Hence a loss of 5 per cent. of calorific power is indicated by the presence in the products of any one of the following percentages by volume of the combustible gases.

	"Standard" (Sec. 16).	Natural gas.
CO,	1.13	1.70
H,	1.30	1.89
CH_4,	0.42	0.61

SEC. 41. *Actual Results in Practice—Composition of Products of Combustion.*—The results of working producer-gases may be learned from the following analyses of products of combustion:

CO_2	O	CO	CH_4	H	N	
15.2	0.8	1.2	0.6	2.0	80.2	full flame.
15.2	1.2	0.2	0.0	0.1	83.3	
15.6	0.6	0.2	tr.	tr.	83.6	
16.8	1.6	0.4	0.0	0.0	81.2	
17.0	0.6	0.2	tr.	tr.	82.2	sharp flame.
17.2	0.2	0.0	tr.	tr.	82.6	" "
16.6	0.8	0.2	0.0	0.0	82.4	" "
16.6	0.6	2.0	0.4	0.3	80.1	smoky flame.
17.0	0.6	0.2	tr.	tr.	82.2	full flame.
17.2	0.4	tr.	tr.	tr.	82.4	
16.0	1.2	0.0	0.0	0.0	82.8	smoky, inward pressure.
15.6	0.4	0.4	tr.	tr.	83.6	smoky, outward pressure.
Av., 16.3	0.8	0.4	0.1	0.2	82.2	

SEC. 42. *Actual Results in Practice—Losses of Heat in Products of Combustion Due to Excess of Air and Escape of Unburned Gases.*—

In Section II we find that the average results of practice in the combustion of producer-gas give waste-gases containing:

	Per cent.
O_2	0.8
CH_4	0.1
H	0.2
CO	0.4

The loss from these causes may be equated from the data given in the preceding pages, as follows:

0.8 per cent. O = 6.4 per cent. excess air = a loss of 0.22 per cent. for every 100° C.

	Per cent.
For 300° C.,	0.7
0.1 per cent. CH_4,	1.2
0.2 " H,	0.8
0.4 " CO,	1.8
	4.5

Hence the average loss from imperfect combustion and from excess of air amounts to 4.5 per cent. of total heat supplied when the escaping gases pass off at a temperature of 300° C. in excess of the incoming gases.

Sec. 43. *The Comparison of the Results of Calculation and of Practice.*—In the comparison of theoretical and practical results, certain disturbing factors are encountered in the chemical action of the flame upon the stock. The chemical history of the operation may be divided into three general periods:

(1) The period of making the bottom and charging the furnace during which the only causes tending to disturb theoretical calculations are the reactions occurring in the checkers.

(2) The period of melting, in which CO_2 is formed by the action of the flame upon the carbon of the metal, and a large amount of oxygen is absorbed by the silicon and manganese.

(3) The period of adding ore, in which CO_2 is formed by the action of the flame upon the carbon of the metal, and a small amount of oxygen is absorbed by the silicon and manganese.

To write the equation of these periods it is necessary to know the amounts of gas burned, of CO_2 formed from the bath, and of oxygen absorbed during each period per unit of time.

Sec. 44. *The Estimation of the Amount of Gas Used During the*

Three Periods Mentioned.—Owing to irregularities of furnace-working, it is impracticable to measure accurately the consumption of gas per hour corresponding to any particular period of a charge. Neither is it necessary for our purpose to make more than an estimate, based on general considerations, for the practice herein examined.

With a knowledge of the amount of coal consumed in regular operations, including repairs, delays and. Sunday interruptions, it may safely be assumed that a consumption of 700 pounds of coal per ton of 2240 pounds of product would be sufficient for continuous working.

This supply would not be given regularly throughout the entire time of operation, but would be greater at certain periods. The furnace is supposed to complete its cycle every eleven hours. The following is an attempt to define the average length of time of the three periods given in Section 43, and the relative fuel-consumption :

SEC. 45. *Length of Periods and Relative Consumption :*

	(a)	(b)	(c)	(d)
First period, ...	90	3	270	9
Second period, ...	450	5	2250	75
Third period, ...	120	4	480	16
			3000	100

(a) Number of minutes in period (determined by experiment).
(b) Relative fuel-consumption (estimate).
(c) Relative consumption per period (a x b).
(d) Percentage of total consumption in each period.

SEC. 46. *The Gas Used per Minute in Each Period.*—By assumption, the total consumption is 700 pounds of coal per ton of product; for a charge of 24 gross tons the consumption for a complete cycle will be 16,800 pounds, and we shall have the following :

	(a)	(b)	(c)	(d)	(e)	(f)
First period, .	16,800 × .09 =	1,512	90	16.8	7.6	34.6
Second period, .	16,800 × .75 =	12,600	450	28.0	12.7	57.8
Third period, .	16,800 × .16 =	2,688	120	22.4	10.2	46.4

(a) = per cent. of total combustion in the period. (See Section 45.)
(b) = pounds of coal burned in the period.
(c) = minutes in the period.
(d) = pounds of coal burned per minute.
(e) = kilos of coal burned per minute.
(f) = cubic meters of gas burned per minute.
1 kilo of coal = 4.55 cubic meters of gas. (See Section 19.)

The amount of coal used while charge is in furnace will be:

Second period,	12,600 pounds.
Third period,	2,688 "
Total for 24 tons,	15,288 "
Total for 24 tons,	6,950 kilos.
Total per ton of charge,	290 "

SEC. 47. *Factors Influencing the Composition of the Products of Combustion.*—Having determined the amount of gas burned during the different periods, it is necessary to determine the effect upon the flame of the agents enumerated in Section 43.

The questions arising are:

(1) The effect upon the gas of its passage through the checkers.

(2) The chemical history of the bath.

SEC. 48. *The Effect Upon the Gas of its Passage Through the Checkers.*—The reactions which may be looked upon as possible in heating a producer-gas to a high temperature are as follows:

(1) A decomposition of hydrocarbons with deposition of carbon.

(2) A union of the carbon so deposited with the oxygen of steam with formation of CO or CO_2 and free H.

(3) A union of the carbon so deposited with the CO_2 with formation of CO.

Whatever changes ensue from internal rearrangement, the condition and content of nitrogen in weight will remain unchanged, and this element may be used as a basis of comparison.

The following are five sets of gases, the two samples in each set being taken simultaneously, one at the bottom, the other at the top of the checkers.

SEC. 49. *Determinations:*

At bottom of the checkers:

	CO_2	O	C_2H_4	CO	H	CH_4	N
(1)	4.0	0.4	0.4	23.2	6.4	3 6	62 0
(2)	5 2	0.4	0.4	22.0	5.3	3.4	63.3
(3)	5.6	0.8	0.4	22.8	8.3	2.8	59 3
(4)	7.2	0 4	0.4	20.0	7.7	3.6	60.7
(5)	6.0	0.6	0.4	21.4	7.5	2.4	61.7
Av.,	5.60	0.52	0.40	21 88	7.04	3.16	61.40

At top of the checkers, after passing through them:

	CO_2	O	C_2H_4	CO	H	CH_4	N
(1)	5.2	0.4	0.0	23.2	6.3	3.6	61.3
(2)	6.0	0.4	0.0	21.8	5.3	3.4	63.1
(3)	8.8	0.4	0.0	23.2	9.9	1 2	56.5
(4)	8.4	0 4	0.0	20.4	8.8	2.4	59.6
(5)	8.4	0.4	0.0	20.8	7 5	2.4	60.5
Av.,	7.36	0.40	0.00	21.88	7.56	2.60	60.20

	CO_2	O	C_2H_4	CO	H	CH_4	N	CH
Before, . .	5.60	0.52	0.40	21.88	7.04	3.16	61.40	13.32
After, . .	7.36	0.40	0.00	21.88	7.56	2.60	60.20	14.97

SEC. 50. *Relative Weights of the Elements.*—Comparing the weights of the elements by the method given in Section 6, we derive the following:

	N	C	H	O
Before passing through checkers, . .	859.60	188.64	14.16	264.64
After passing through checkers, . .	842.80	191.04	12.76	292.80

or reducing to N = 1000 (see Section 48) we have:

	N	C	H	O
Before passing through checkers, . .	1000	219 45	16.47	307.86
After passing through checkers, . .	1000	226.67	15.14	347.41

These figures show the following changes in the gas by its passage through the checkers:

(1) An increase of carbon of 3 per cent.

(2) A decrease of hydrogen of 8 per cent.

(3) An increase of oxygen of 11 per cent.

SEC. 51. *Considerations on the Foregoing.*—This increase of oxygen would, at first, seem to show a leakage of air; further consideration shows that air cannot possibly have entered. This may best be seen by supposing a leakage of air amounting to 100 per cent., as measured by the nitrogen already present. The calculation will stand as follows:

	C	H	O	N
Present, ...	219	16.5	308	1000
Supposed addition, ...			265	1000
Composition with supposed addition, .	219	16.5	573	2000
Composition on basis of N = 1000, .	110	8.3	287	1000

In other words, every unit of air added would reduce the percentage of carbon, hydrogen and oxygen. Hence no leakage of air is shown by the analysis; nor can the oxygen be supposed to come from the decomposition of steam, since there would in that case be an increase in hydrogen. The situation may be summed up as follows:

The C_2H_4 has been broken up completely by the high temperature, the CH_4 seriously encroached upon, while the CO remains unchanged.

The free hydrogen has received of its kind from the decomposed

hydrocarbons; the CO_2 has increased in quantity, probably deriving its carbon from the broken hydrocarbons and the oxygen from an unknown region.

A small amount of the free oxygen present has been induced to unite by the exaltation of temperature.

For practical purposes the gas may be looked upon as passing though the checkers unchanged.

Sec. 52. *The Chemical History of the Bath.*—The chemical history of the bath is essentially one of oxidation. Reduction by means of the flame alone does not obtain in practice. The rapidity of the oxidation varies with the character of the flame, the supply of combustible material, and the nature and disposition of the stock. The presence of a large proportion of hydrogen, either free or combined, in the gas, and a long exposure to a temperature just below the melting point, increase the amount of oxidation, while the absence of easily oxidizable elements in the metal concentrates this action upon the metallic iron. It follows that the data of one practice or one charge are worthless for general conclusions. Groups may be taken, however, under different conditions whose histories in the furnace have been characteristic, and the results assumed to be representative.

The following is an attempt to compare the results obtained from two groups:

Group I. Nineteen boiler-heats. Producer-gas.

Group II. Six boiler-heats. Oil-gas, much steam in gas.

Sec. 53. *Average per Heat:*

		Group. I.	Group. II.
(1) Pig used Si 1.72, C 3.50, pounds,		11,700	20,700
(2) Steel,	Rail Si 0.07, C 0.40, pounds,	39,580	29,600
	Boiler Si 0.02, C 0.13, pounds,	5,970	7,200
(3) Ore FeO = 81 per cent., free O = 9 per cent., by analysis, pounds,		1,020	850
(4) Metal when melted, by analysis,	Si per cent.,	0.02	0.05
	Mn, "	0.09	0.06
	C, "	0.54	0.64
(5) Metal before tapping, by analysis,	Si per cent.,	0.02	0.01
	Mn, "	0.04	0.02
	C, "	0.13	0.12
(6) Slag after melting, by analysis,	SiO_2 per cent.,	50.24	49.46
	MnO, "	21.67	13.16
	FeO, "	23.91	33.27

	Group. I.	Group. II.
(7) Slag before tapping, by analysis, SiO_2 per cent.,	49.40	49 36
(7) Slag before tapping, by analysis, MnO, "	16.50	11.30
(7) Slag before tapping, by analysis, FeO, "	29.79	34.11
(8) Time of melting, minutes,	433	480
(9) Time of oreing,	150	120
(10) Weight of slag under ladle, pounds,	4,050	5,670
(11) Weight before tapping (assumed), pounds,	3,900	5,520
(12) Amount MnO in slag before tapping, pounds (from 7 and 11),	644	624
(13) Amount MnO formed after melting, pounds (from 4 and 5),	36	28
(14) Amount MnO in slag after melting, pounds (12 — 13),	608	596
(15) Per cent. MnO in slag after melting (see 6),	21 67	13.16
(16) Weight of slag after melting, pounds (14 ÷ 15),	2,810	4,530

NOTE.—It will be seen that we have calculated the weight of the slag after melting and before adding the ore. The method of doing this is indicated by the sequence of the above figures. Since MnO must come entirely from the oxidation of Mn in the metal, and since, knowing the weight of the bath and the content of Mn at any two periods, we may determine the amount of Mn oxidized and absorbed by the slag, it follows that if we know the actual weight of MnO in the slag at one period, we may find the weight in the slag at the other period. Knowing thus the weight of MnO in the slag and having by analysis the percentage it constitutes, we may easily calculate the weight of the slag of which it forms a part.

It will be noticed that the weight before tapping is assumed ; the weight after tapping is definitely known. The only difference in weight between these two periods comes from the MnO formed in the recarbonizing, and from the incorporation of a little material from the tapping-hole and the ladle linings. The error made in a careful assumption is a very small proportion of the total weight.

Another method of obtaining the weight of slag before tapping would be to make allowance for the MnO given by the recarbonizer and calculate the slag from its percentage of MnO. The amount of MnO, however, given by the recarbonizer is so great in proportion to the total amount present that the probable error is quite large ; and since this error is multiplied several fold in calculating the slag, the method of estimating the weight as given above is far preferable.

SEC. 54. From the foregoing we derive the following results :

		Group. I. Lbs.	I. Oxygen needed.	II. Lbs.	II. Oxygen needed.
(17) During melting,	Si oxidized, . . .	219	250	378	432
	Mn, ...	473	135	464	132
	Fe, ...	523	149	1172	335
	C, ...	279	744	500	1333
			1278		2232
(18) Nitrogen with the oxygen, . . .			4230		7388
(19) During oreing,	Si oxidized, . .	—	—	22	25
	Mn, ...	28	8	22	6
	Fe, ...	—	—	—	—
	C, ...	230	613	286	763
			621		794

SEC. 55. From Section 54 we have:

	Group I. Lbs.	Kilos.	Kilos per min
(20) Oxygen required by Si, Mn and Fe during melting (see 17), ...	534	243	0.56
(21) CO_2 formed during melting (see 17), .	1023	465	1.07
(22) CO_2 formed during oreing (see 19), . .	843	383	2.55
	Group II.		
(23) Oxygen required by Si, Mn and Fe during melting (see 17), ...	899	409	0.85
(24) CO_2 formed during melting (see 17), .	1833	833	1.74
(25) CO_2 formed during oreing (see 19), . .	1049	477	3.98

SEC. 56. From Section 53 we may also derive:

In pounds.	Group. I.	II.
(26) FeO in slag after melting (see 6 and 16), . .	672	1507
(27) FeO in ore added (see 3), ...	826	689
(28) FeO which would be present in final slag if no reduction took place, (26 and 27),	1498	2196
(29) FeO in final slag (7 and 11), ...	1162	1683
(30) FeO reduced during oreing (28 and 29), . .	336	313
(31) Oxygen in reduced FeO, ...	75	70
(32) Free oxygen in ore (see 3), ...	92	77
(33) Total oxygen derived from the slag and ore during oreing, (31 and 32),	167	147
(34) Total oxygen required during oreing (see Section 54), ...	621	794
(35) Oxygen from flame during oreing (34 — 33), .	454	647
(36) Nitrogen associated with this oxygen, 3.31 × (35),	1503	2142

SEC. 57. *Additions from the Bath to the Products of Combustion.*—Hence the result of the chemical interactions of the flame, ore and metal add to the normal products of combustion the following factors:

		Lbs.	Kilos.	Kilos per min.	C.m. gas per min.
			Group I.		
During melting, (18 and 21)	N,	4230	1923	4.44	3.55
	CO_2,	1023	465	1.07	0.54
During oreing, (36 and 22)	N,	1503	683	4.55	3.64
	CO_2,	843	383	2 55	1.29
			Group II.		
During melting, (18 and 24)	N,	7388	3388	7.06	5.65
	CO_2,	1833	833	1.74	0.88
During oreing, (36 and 25)	N,	2142	974	8.12	6.50
	CO_2,	1049	477	3.98	2.62

SEC. 58. *Normal Products of Combustion of Coal-Gas.*—According to Section 46 we have as the volume of gas burned during the two periods, the following:

Coal-gas.	C.m. per min.
During melting,	57.8
During oreing,	46.4

According to Section 28, 100 c.m. gas =		C.m.
	CO_2,	31.2
	N,	140.6

Hence we have as the products of a furnace, neglecting the factors coming from the metal:

		C.m.
During melting, per minute,	CO_2,	18.0
	N,	81.3
During oreing, per minute,	CO_2,	14.5
	N,	65.2

SEC. 59. *Composition of Products of Combustion in the Two Periods of the Charge and with an Empty Furnace* (producer-gas).—From the preceding two sections we may derive the composition of the gases actually coming from a furnace in operation.

		Cubic Meters from gas.	Cubic Meters from metal.	Total.	Composition, per cent.
During melting, per minute	CO_2,	18.0	0.5	18.5	17.9
	N,	81.3	3.6	84.9	82.1
				103.4	100.0
During oreing, per minute	CO_2,	14.5	1.3	15.8	18.7
	N,	65·2	3.6	68.8	81.3
				84.6	100.0

Hence we have as the composition of the waste-gases in the several periods the following:

	Per cent. by volume.	
	CO_2.	N.
Empty furnace (see Section 31), ...	18.2	81.8
During melting, ...	17.9	82.1
During oreing, ...	18.7	81.3

From this it is evident that the chemical reactions in the bath do not materially affect the composition of the products of combustion.

The Use of Oil-Fuel.

Sec. 60. The oil-gas referred to in the following remarks was made by vaporizing crude petroleum by a jet of superheated steam, and heating the mixture to a temperature of 300° to 400° C. If such a mixture of vapors is put into the hot chambers of a regenerative furnace, the reactions caused by high temperature, thorough mixing and impact create permanent gases with deposition of carbon. The steam brings with it some oxygen and nitrogen which was dissolved in the water in the boilers, and this oxygen, together with that of the steam tends to pick up this deposit of carbon with liberation of hydrogen. As an agent in freeing the checkers of the deposit this action may be important. It would be more important if the work were more thoroughly done. As an element in thermal calculations it is even less necessary to consider it.

If the so-called gas is used in a non-regenerative furnace the steam may be entirely neglected; the liberated hydrogen must be oxidized again, or its heating power will be lost, while if such oxidization does take place, it is difficult to find where anything is gained.

The reaction

$$C + H_2O = CO + 2H$$

followed by

$$CO + 2H + 2O = CO_2 + H_2O$$

cannot possibly give more heat than the one reaction,

$$C + 2O + H_2O = CO_2 + H_2O.$$

It is therefore ridiculous to speak of anything being gained from the steam in those cases where all reactions must occur in one place.

In a regenerative furnace it is conceivable that dissociation may be effected by waste-heat, and the dissociated gases be burned in another place where heat is needed. This reasoning, however, can

apply only to simple dissociation. The transference of oxygen from hydrogen to carbon in the chamber, when viewed as the anticipation of the reactions in the furnace, seems a purely profitless proceeding. As a medium for transferring the oil-vapor from the producer to the furnace, the steam serves a useful purpose, but it may be entirely neglected as a thermal agent.

The following are analyses of various gases taken from furnaces using oil-fuel.

a.—Gas after passing through checkers:

	CO_2	C_2H_4	O	CO	H	CH_4	N.	Undecomposed steam.
1	4.6	0.0	1.8	9.6	51.6	8.4	24.0	240 grammes per c.m.
2	5.6	0.4	2.0	11.2	51.8	7.2	21.8	122 " "
3	4.4	0.0	4.0	6.4	42.2	7.0	36.0	
4	6.6	0.0	0.6	10.0	37.3	7.6	37.9	
5	4.0	6.0	1.4	5.4	44 1	22.1	17.0	
6	4.0	7.6	2.0	5.4	46.5	23.1	11.4	
7	4.4	3.6	2.4	6.0	44.4	19.0	20.2	

It will be noticed that the above analyses of gas show a wide variation in composition. The first four samples were taken after traversing about five feet of open-hearth checker-work at a temperature of about 1200° C. The last three samples were taken after passing through three feet of heating-furnace checker-work at a much lower temperature, and represent the oil-vapor in various stages of decomposition. The presence of so much free oxygen, the high CO_2 and the large amount of steam, when considered in connection with the carbonaceous components, illustrate plainly the fact that a long time and a high temperature are necessary for the completion of the reactions incident to the gasification of oil-fuel.

b.—Products of Combustion—Theoretical.

The perfect combustion of the first gas in the above table will give the following:

	CO_2	C_2H_4	O	CO	H	CH_4	N
Gas (1)	4.6	0.0	1.8	9 6	51.6	8.4	24.0

Volume of gas, 100 c.m. Air required, 217.5 c.m.

Products.	CO_2	22.6 c m.	22.6 c.m.	10.3 per cent.
	H_2O	71.7	.....	.....
	N	195.9	195.9 c.m.	89.7 per cent.
			218.5	100.0

This neglects steam in the gas which would not influence volumetric composition of products.

c.—Products of Combustion—Practical.

As before stated, the gas on entering the checkers deposits large amounts of carbon and some tarry matter from the dissociation of the complex hydrocarbons. The products of combustion which leave by way of the gas-chambers react upon this deposit, and the result may be seen in the following samples:

Products at top of gas-checkers:

	CO_2	C_2H_4	O	CO	H	CH_4	N	
(8)	2.4	0.0	0.8	6.0	9.9	0.6	80.3	Sharp flame.
(9)	2.8	0.0	1.0	3.0	4.3	0.8	88.1	" "
(10)	3.4	0.0	0.4	6.0	7.4	0.8	82.0	Soft "

Products in flue after passing through gas-checkers:

	CO_2	C_2H_4	O	CO	H	CH_4	N
(11)	9.4	0.0	0.6	2.6	5.6	0.4	81.4
(12)	13.2	0.0	0.4	0.4	1.1	0.4	84.5
(13)	14.0	0.0	1.0	1.4	4.5	0.8	78.3

Double samples. One taken at top and one at bottom of gas-checkers at same time:

	CO_2	C_2H_4	O	CO	H	CH_4	N	
(14)	2.8	0.0	0.8	5.2	7.8	0.8	82.6	Top.
	9.2	0.0	0.4	2.0	4.9	0.8	82.7	Bottom.
(15)	3.6	0.0	0.4	5.2	6.2	1.0	83.6	Top.
	10.8	0.0	0.6	1.8	6.0	1.0	79.8	Bottom.

A consideration of these figures gives rise to problems which are almost insoluble. Instead of carbonization a process of oxidation is indicated. For purposes of comparison, therefore, the products are given as taken from the air-chambers.

Double samples. One taken at the top and one at the bottom of air-checkers at the same time:

	CO_2	O	CO	H	N	
(16)	3 4	0.6	8.2	12.0	75.8	Top.
	11.8	4.6	0.0	0.0	83.6	Bottom.
(17)	2.8	0.8	6.0	11.2	79.2	Top.
	9.6	6.4	0.0	0.0	84.0	Bottom.

The samples from the bottom of the chamber are harder to take,

owing to an increased draught to overcome, and the use of tubes four feet long, extending down beyond the reach of air from the testing-orifice. The collecting tubes in the above instances were filled with water to exclude air, and the pipes reaching into the flues were washed out with waste gases before sample was taken. In the following instance extra care was used; the amount used to wash out the one-quarter inch pipe was over five hundred times its capacity in order to make certain that no air came from the apparatus:

Products from bottom of air-chamber.

(Extra precautions in taking sample.)

CO_2	O	CO	H	N
9.2	7.2	0.0.	0.0	83.6

This would indicate that the foregoing samples were properly taken. From the samples numbered 16 and 17 we have the following:

	CO_2	O	CO	H	N
Average at top of checkers, .	3.1	0.7	7.1	11.6	77.5
" at bottom of checkers,	10.7	5.5	0.0	0.0	83.8

To get the 5.5 per cent. of oxygen present in the gases from the bottom of the checkers, we must assume that the top-gases are burned with about 100 per cent. more air than is needed theoretically for perfect combustion. This gives as follows:

100 c.m. Gas plus 83.0 c.m. Air, gives

CO_2,	10.2 c.m.	10.2 c.m. = 6.3 per cent.	
O,	8.7 "	8.7 " = 5.4	"
H_2O,	11.6 "		
N,	143.1 "	143.1 " = 88.3	"
		162.0	100.0 "

To add less air than this will bring the percentage of CO_2 nearer to the amount found at the bottom, but will lower the oxygen and raise the nitrogen; while to assume that the leakage of chamber-walls supplies 83 c.m. of air to every 100 c.m. of products of combustion is rather bold. The problem is somewhat puzzling; but experiments on different furnaces, under different conditions, would probably locate the causes of error.

The Chemical Equation of the Acid Process.

Sec. 61. Sections 52–56 present data concerning a series of nineteen heats made with coal-gas. The amount of ore used varied

from 600 to 1500 lbs. per charge. It is desirable to know what conditions obtain with varying amounts of ore. The following are the results of analyses and calculations on charges classed according to the weight of ore used in decarbonizing:

Group	I.	No ore used.
"	II.	100 to 500 lbs.
"	III.	600 to 1000 "
"	IV.	1000 to 1500 "
"	V.	1600 to 2500 "

Each group embraces from two to five members. With them are given the results of the nineteen heats given in sections 52–56.

SEC. 62. *The Composition of the Slag.*—The following is a comparison of the composition of the slag, (1) after melting, (2) before tapping:

	Silica.		MnO.		FeO.	
Group.	(1)	(2)	(1)	(2)	(1)	(2)
I., . .	49.32	50.73	19.16	19.60	27.00	27.93
II., . .	49.18	46.17	15.06	13.03	30.90	35.23
III., . .	48.79	47.83	14.24	11.68	33.54	37.00
IV., . .	51.79	50.43	20.88	16.31	24.61	30.63
V., . .	50.29	49.82	20.91	16.50	26.66	30.22
Av. 19 heats, See Sec. 53.	50.24	49.40	21.67	16.50	23.91	29.79

Conclusions.—(1) The percentage of FeO increases, owing to absorption from the ore.

(2) The percentage of SiO_2 remains about the same, owing to absorption from the bottom.

(3) The percentage of MnO decreases, owing to dilution by the FeO and SiO_2 added.

SEC. 63. *The Composition of the Metal.*—The following is a comparison of the composition of the metal, (1) after melting, (2) before tapping:

	Si.		Mn.		C.
Group.	(1)	(2)	(1)	(2)	(1)
I., ...	.03	.02	.06	.04	.36
II., ...	.01	.01	.03	.02	.18
III., ...	.04	.03	.05	.02	.32
IV., ...	.07	.02	.15	.05	.61
V., ...	.09	.02	.15	.03	.57
Average 19 heats, See Sec. 53. .	.02	.02	.09	.04	.54

Conclusions.—(1) During the melting-period there is oxidized:

Nearly all the silicon;

Nearly all the manganese;

About one-half the carbon; (See Section 54.)
About 1 per cent. of metallic iron. (See Section 54.)

(2) During the oreing there is oxidized:
The remaining silicon to about 0.02 per cent.;
The remaining manganese to about 0.03 per cent.;
The carbon to the desired point.

SEC. 64. *The Amount of FeO Reduced.*—The following is a comparison of the groups with regard to the amount of FeO reduced during oreing:

(a) = pounds FeO added in the ore.
(b) = pounds FeO reduced during oreing.
(c) = percentage of the FeO addition that is reduced.
(d) = pounds FeO added to the slag by the ore addition = (a — b).

Group.	(a)	(b)	(c)	(d)
I., ...		7		
II., ...	223	58	27	165
III., ...	664	346	52	318
IV., ...	1215	793	65	422
V., ...	1499	1172	79	327
Average of 19 heats, See Sec. 53. . .	826	336	41	490

NOTE.—For method of calculation, see Section 53.

Conclusions.—(1) In the case of Group I., there appears to be a reduction of iron from the slag; the amount, however, is so slight that it is covered by the possible errors of determinations.

(2) The amount and percentage of reduced FeO rises with the amount added.

(3) The amount of FeO added to the slag by the ore does not increase proportionately, or even absolutely, with an increase of ore.

SEC. 65. The foregoing tables give the data of the various heats at two periods, (1) after melting, (2) before tapping. We are thus enabled to compare ultimate results. It remains to investigate the record of reactions between these two epochs. For this we select four heats, each using 1500 pounds of ore, and take samples at four different times, as follows: (1) After melting, no ore; (2) after adding 500 lbs. ore; (3) 1000 lbs.; (4) 1500 lbs. The following are analyses of metal at the periods mentioned.

No. of Charge.	Silicon. No ore.	500 lbs.	1000 lbs.	1500 lbs.	Manganese. No ore.	500 lbs.	1000 lbs.	1500 lbs.
7596	.07	.01	.01	.01	.10	.02	.02	.02
7598	.04	undet.	undet.	.01	.02	.02	.02	.02
7606	.04	.05	.03	.02	.08	.05	.03	tr.
7635	.13	.07	.05	.06	.19	.08	.09	.10

SEC. 66. *The Reduction of FeO During Oreing.*—The slags corresponding to the periods specified in Section 65 gave the following results:

Silica.

	No ore.	500 lbs.	1000 lbs.	1500 lbs.
7596	50.27	49.27	52.77	50.97
7598	51.96	51.10	50.30	51.48
7606	52.43	55.82	55.73	55.66
7635	52.94	51.72	52.28	52.90

MnO.

	No ore.	500 lbs.	1000 lbs	1500 lbs.
7596	14.91	15.20	14.70	14.22
7598	21.65	19.09	17.50	16.72
7606	15.61	15.31	13.89	12.40
7635	21.84	20.44	19.06	16.36

FeO.

	No ore.	500 lbs.	1000 lbs.	1500 lbs.
7596	31.23	30.68	26.96	31.70
7598	22.59	26 12	28.26	26.03
7606	27.14	25.11	26.20	26.96
7635	23.18	24.21	26.26	29.13

SEC. 67. *The Reduction of FeO During Oreing.*—The weight of slag will be as follows:

	No ore.	500 lbs.	1000 lbs.	1500 lbs.
7596	4020	4210	4350	4500
7598	2740	3110	3390	3550
7606	1940	2100	2400	2850
7635	1840	2320	2490	2900

From which we derive the amount of FeO in slag in pounds:

	No ore.	500 lbs.	1000 lbs.	1500 lbs.
7596	1255	1292	1173	1427
7598	619	812	958	924
7606	527	527	629	768
7635	427	562	654	845
Average,	707	798	854	991
Increment,		91	56	137

The amount added in the ore was a constant for the three periods —405 pounds.

Hence we have the following, as reduced:

First period,	314
Second period,	349
Third period,	268

NOTE.—For method of calculation, see Section 53.

SEC. 68. *The Reduction of FeO During Oreing.*—Owing to difficulty in taking a true sample of the slag at any time and the errors of determination, certain irregularities in the results are unavoidable. The use of the average of four heats tends to reduce the effect of these errors, but the results must not be translated too literally.

Practical observation makes it certain that after the carbon is reduced below 0.15 per cent., oxidation of iron ensues. The decrease in the reduction during the latter period, shown by the results given in Section 67, is consistent with this fact.

Taken as a whole the data would indicate that the rate of reduction is practically uniform for the charge.

THE CHEMICAL EQUATION OF THE BASIC PROCESS.

SEC. 69. *Experimental Data.*—The following are the data of three series of heats made by the basic open-hearth process:

Conditions: Five-ton furnace; dolomite bottom; lime charged with metal; metal charged cold and all at one time; ore added as needed. Melted with Archer oil-gas.

Recarbonization: First series in ladle; second series in furnace; third series in furnace.

In the first two series no bottom was made between the charges, but at the end of each series enough dolomite was applied to restore the bottom to its original condition, and the wear was thus determined. After the third series, when the slag-line and other small patches had been repaired by the use of 1500 lbs. of dolomite, it was found that the level of the next charge was lower than before the series, indicating a general scorification, which was estimated at 4200 lbs. of lining. This scorification, however, had not encroached upon the original lines of the bottom.

SEC. 70. In running a basic furnace it often happens that the bottom becomes partially filled or built up with a thick viscous material. This arises from non-homogeneity in the slag. In charges which do not stay long in the furnace after melting, the lime-additions do not have sufficient time to unite with the acid ingredients and form a homogeneous mass. The result is a slag made up of a thin homo-

geneous liquid, containing practically all the SiO_2, P_2O_5, MnO and FeO and part of the CaO and MgO, surrounding and enclosing globules and masses of lime. These masses, owing to absorption of the surrounding liquid, are very gummy; and when the furnace is tapped much of this material sticks to the sides and bottom.

Such a set of conditions may be succeeded by one or more charges having a thin homogeneous slag and a high temperature. The result will be a scorification of this deposit.

Previous to the third series, such a building up had been in progress. During the series, such a wearing-away ensued. It is impossible to state the exact composition of the material thus taken up, but it will be nearly correct to assume that one-half of it was lime that had escaped fluxing and the other half was slag absorbed by this lime. It will also be supposed that the lime and slag were of the same composition as those found in this series.

The following table shows the materials used in making these three series of heats, with the analysis as far as known. All estimated values are enclosed in brackets. Everything is thus marked where a sample was not taken of the material actually used. Every practical man will know that with regular stock the composition is known within quite narrow limits.

SEC. 71.—*Table of Materials.*

FIRST SERIES—SIX HEATS. ANALYSIS.

Material.	Weight.	Si.	Mn.	P.	SiO_2.	CaO.	MgO.	FeO.
Bessemer Pig	37,400	0.45	0.92	.076	[0.50]	.....	.....	.....
Basic Pig	3,000	0.36	1.68	[3.00]	[2.00]	.....	.....	.....
Scrap	38,000	[0.05]	[0.90]	[.09]	.....	.....	.....	.....
Ore	750	.....	.....	[.04]	6.73	.....	.....	82.08
Lime	3,600	.....	.....	.....	6.19	80.80	3.60	.....
Dolomite	[1,000]	.....	.....	.....	[4.00]	[58.00]	[38.00]	.....
Ferro	470	.....	[80.00]	.....	.....	.....	.....	.....

Wear of roof assumed to be 15 lbs. per heat; of ladle-linings, 40 lbs.

SECOND SERIES—SEVEN HEATS. ANALYSIS.

Material.	Weight.	Si.	Mn.	P.	SiO_2.	CaO.	MgO.	FeO.
Bessemer Pig	34,000	0.43	0.38	.095	[0.50]	.....	.....	.....
Basic Pig	3,500	0.36	1.23	3.186	[2.00]	.....	.....	.....
Scrap	53,000	[0.05]	0.91	.092	.....	.....	.....	.....
Pit Scrap	500	.....	0.50	[.030]	[5.00]	.....	.....	.....
Ore	1,700	.....	.....	.040	[6.73	.....	.....	[82.08]
Lime (partially slaked)	3,975	.....	.....	.....	[5.40	70.07	4.54	.....
Dolomite	2,000	.....	.....	.....	[4.00	[58.00]	[38.00]	.....
Ferro	1,220	.....	[80.00]	.....	.....	.....	.....	.....
Spiegel	300	.....	[20.00]	.....	.....	.....	.....	.....

Wear of roof assumed to be 15 lbs. per heat; of ladle-linings, 40 lbs.

THIRD SERIES—SEVENTEEN HEATS. ANALYSIS.

Material.	Weight.	Si.	Mn.	P.	SiO_2.	CaO.	MgO.	FeO.
Bessemer Pig	109,500	0.78	0.58	.082	.....	.....	.....	.....
Basic Pig	8,500	0.22	1.03	3.089	.....	.....	.....	.....
Scrap	102,000	0.05	1.11	.094	.....	.....	.....	.....
Fit scrap	1,000	0.01	0.50	0.02	[5.00]	.....	.....	.....
Ore	2,740	.....	.....	0.034	1.00	.....	.....	88.38
Lime	10,200	.....	.....	.....	2.5	78.1	2.8	0.60
Dolomite	1,500	.....	.....	.....	1.37	51.69	33.75	.....
Wear of bottom	[4,200]	.....	.....	.....	.....	.....	.....	.....
Ferro	1,275	.....	79.76	.....	.....	.....	.....	.....
Spiegel	5,100	.....	18.04	.....	.....	.....	.....	.....

Wear of roof in 17 heats estimated to be 360 lbs. SiO_2.
Wear of ladle-linings, 1,200 lbs., of which the SiO_2 will be about 750 lbs.
Sand on pig-iron estimated to be 400 lbs.
NOTE.—Figures enclosed in brackets [ ] are estimates.

SEC. 72. *Chemical Data of Slags and Metals.*—The samples of metal and slag were taken as follows:

A. After complete fusion of the metal.

B. After beginning of the boil.

C. Metal ready for the recarbonizer. When the recarbonizer was added in the ladle, this test was taken within 5 minutes of tapping.

D. Final product leaving the steel ladle.

NOTE.—All data given below refer to the total weights and average samples of all the heats included in the series. Such average samples were made by taking a sample from each separate heat and mixing them together. The same method was employed in taking samples of material used. A small amount of pig, steel, ore, or lime, was taken from that used on each heat, and these lots were mixed together to get an average sample.

The following is a table showing the analysis of the tests of slag and metal:

Table of Analyses of Slag and Metal.

First Series.	Metal.					Slag.						
	Weight.	C.	Si.	Mn.	P.	Weight.	SiO_2.	MnO.	CaO.	MgO.	FeO.	P_2O_5.
Test A	.....	.....	.01	.27	.013	.....	12.90	12.84	40.76	4.43	21.96	2.48
" B	Not taken	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
" C	.....	.....	.01	.19	.011	.....	12.96	10.71	41.57	6.07	22.23	1.71
" D	73,600	.....	.....	.40	.015	9,350	13.97	12.14	40.26	6.44	20.61	1.83
Second Series.												
Test A	.....	.36	.01	.27	.035	.....	15.45	14.90	36.93	6.72	18.54	3.195
" B	.....	.19	.01	.24	.020	.....	13.09	14.29	36.65	7.00	22.32	2.853
" C	.....	.10	.01	.14	.013	.....	9.66	9.87	34.98	7.19	33.39	2.264
" D	93,620	.13	.....	.44	.014	13,140	10.31	14.85	33.42	7.44	27.72	1.559
Third Series.												
Test A	.....	.71	.06	.33	.046	.....	19.21	11.12	42.16	6.64	13.68	5.149
" B	.....	.34	.01	.25	.022	.....	16.37	10.36	42.78	7.87	16.29	4.848
" C	192,750	.12	.01	.22	.013	.....	15.08	9.01	42.16	8.45	20.34	3.850
" D	197,720	.16	.01	.49	.018	30,780	15.75	14.11	39.05	10.40	16.65	2.961

In estimating weight of C metal, allowance is made for addition of recarbonizer, less its oxidized manganese.

Sec. 73. *Calculations on Slags* D.—From the composition and weight of slags D we may compute the actual amount of the various components present, while from the table of materials in Sec. 71, by allowing for the elements contained in the final metal, we may determine the amount of each component which we should expect to find; this latter we will call the "theoretical amount." Following is the result of such calculation in pounds:

	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
First series.						
Actual, . . .	1331	1157	3837	614	1964	174
Theoretical, . .	1214	1037	3489	510	—	323
Second series.						
Actual, . . .	1355	1951	4391	978	3642	205
Theoretical, . .	1348	1624	3945	940	—	414
Third series.						
Actual, . . .	4848	4343	12020	3201	5125	911
Theoretical, . .	3383	3946	11201	1069	—	1092

Sec. 74.—In the results of the first two series, it will be noticed, that the amounts actually found in the slag of SiO_2, MnO, CaO, and MgO agree reasonably well with the amounts supposed to be added. In the case of the P_2O_5 a considerable loss is indicated. It was with the intention to verify or discredit this result that the third set of charges was investigated. In this series care was taken to sample all material with extra precautions. The Bessemer pig-iron was sampled by taking two pieces of the lot charged in each heat. The basic pig was also sampled each heat, and the analysis agrees with every determination ever made of the lot in question. Both the Bessemer and basic pig-irons came from large piles which were made up of metal of similar composition, and were well mixed when unloaded from cars. The steel-scrap all came from one rolling-mill; this mill was supplied entirely from one Bessemer plant; the product of this Bessemer plant was sampled by taking drillings from every sixth heat throughout the day for the four consecutive days in which its product was used in the experiments. The regularity of composition, as far as phosphorus is concerned, will be seen from the following list of determinations of this element from such daily samples: .088, .089, .092, .099—average, .092. The scrap was also sampled by taking a piece from the lot charged in each heat. The average phosphorus obtained was .096.

The value of .094 for the phosphorus content of the steel-scrap may, therefore, be looked upon as reliable.

The slag-tests were made by taking a sample in a small ladle from three different parts of the bath after thorough stirring, thereby avoiding, as far as possible, the effect of the non-homogeneity as described in Sec. 70.

The metal-tests were taken at the same time as the slag-tests.

The following is the full calculation of the third series, the general principles being the same as explained in the note to Sec. 53, viz.: Knowing the weight of slag D, we subtract the weight of material which entered the slag between tests C and D; said material is as follows:

(1). The MnO from recarbonizer.

(2). The clay from ladle-lining.

(3). The SiO_2 from roof.

(4). The SiO_2, CaO, MgO, etc., coming from bottom.

(1). MnO from recarbonizer.

Mn in metal before recarbonizer,	435 pounds.
Mn added in recarbonizer,	1937 "
Total,	2372 "
Mn in final steel,	969 "
Mn entering slag,	1403 "
MnO formed,	1811 "

(2). Clay from ladle-lining.

A careful estimate of the wear of ladle-linings, stoppers, etc., gave for the 17 heats a total of 1200 pounds, of which about 750 pounds would be SiO_2.

(3) and (4). As before stated, the wear of the roof during the series was estimated at 360 pounds. The wear of the bottom, as explained in Sections 69 and 70, was estimated at:

Dolomite,	1500 pounds.
Lime,	2100 "
Old slag,	2100 "
Total,	5700 "

All of the above reactions, however, do not fall between tests C and D. It is impossible to apportion to each period its proper share, but as a wrong estimate is better than none, it will be assumed that an equal amount of wear of roof and bottom occurs in each of the four intervals. The shortness of the later periods is offset by the greater intensity of heat and increased violence of the reaction.

The material, therefore, which was added to the slag between tests C and D from the roof amounted to 90 pounds, and from the bottom, 1425 pounds.

SEC. 75.—We have, then, the following:

		Pounds.	Pounds.
Weight, Slag D (scale weight),			30,780
Added to slag between Tests C and D,	from recarbonizer,	1811	
	from ladle,	1200	
	from roof,	90	
	from bottom,	1425	4,526
Weight, Slag C,			26,254

NOTE.—Although the errors in above assumptions may be a large proportion of the amounts to which they apply, the proportion which they bear to the slag C is very small. The total weight of this slag C must be very nearly correct, and, since the analysis is known, we may find the weights of the components as follows:

Slag C.

	Weight.	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
Analysis, per cent.,	—	15.08	9.01	42.16	8.45	20.34	3.850
Pounds,	26,254	3959	2365	11,069	2218	5340	1011

SEC. 76. *Calculation of Slag B.*—In the calculation of slag C, it must not be forgotten that very little depends upon estimates. The weight of the slag before tapping, must be approximately the same as after it, and thus, from the analysis, we may calculate accurately the weights of the constituents, undisturbed by the large quantity of MnO introduced by the recarbonizer and the reactions incident thereto. In the less valid determinations of slags B and A, the data of slag C, as given above, will be taken as the basis of operations. It is necessary to select also a component which exists in nearly equal amount in both slags B and C. The following is a synopsis of the factors operating to change the quantity and composition of the slag between the periods B and C:

(1) The Mn oxidized in the metal.
(2) The P oxidized in the metal.
(3) The ore added to decarbonize the metal = 775 pounds.
(4) The droppings from the roof.
(5) The wear of the bottom.

The MnO formed from oxidation of bath, is found as follows:

		Per cent.		Pounds.
Weight C metal (see Table, Section 72),				192,750
Mn in B test,	=	0.25		
Mn in C test,	=	0.22		
Oxidized,		0.03	=	58
MnO produced,			=	75

Similarly, we have the phosphorus oxidized

= 0.009 per cent. = 17 pounds = 39 pounds P_2O_5

The wear of the roof and bottom has been assumed to be one-fourth of the total in each of the intervals (see Section 74).

We can, therefore, make the following summary:

Added to Slag between Tests B and C.

	Pounds.					
	SiO_2.	MnO.	CaO.	MgO.	FeO.	P_2O_5.
Oxidation of Mn	.....	75	.....	.....	.....	.....
Oxidation of P	.....	.....	.....	.....	.....	39
Ore.....775 lbs.	8	.....	.....	.....	685	1
Roof-droppings	90	.....	.....	.....	.....	.....
Dolomite from bottom, 375 lbs.	5	.....	194	127	.....	.....
Lime " " 525 lbs.	13	.....	410	15	.....	.....
Old slag " " 525 lbs.	83	74	205	55	87	16
Total,	199	149	809	197	772	56

Selecting from these factors the individual, MnO, and calculating the total weight of slag B, we have the following:

MnO as a basis.

Weight of MnO in Slag C (see Section 75),	2365	pounds
Added between B and C,	149	"
MnO present in Slag B,	2216	"
Per cent. MnO in Slag B (see Section 72),	10.36	

Weight Slag B = 2216 ÷ 0.1036 = 21,390 pounds.

We then have

Slag B.

	Weight.	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
Analysis, per cent.,		16.37	10.36	42.78	7.87	16.29	4.848
Pounds,	21,390	3502	2216	9151	1683	3484	1037

NOTE.—775 pounds ore added between B and C.

In the above calculation, the MnO is chosen because the amount added from the bath during the interval between B and C is definitely known, and the amount from the wear of the bottom is a small proportion of the whole. It is also one of the components which as explained in Section 70, will, in the early stages of the operation, be a constituent of the fluid and homogeneous part of the slag.

SEC. 77. *Calculation of Slag A.*—By exactly the same method as exemplified in the calculation of slag B, we have:

MnO as a basis.

Weight of MnO in slag C, ...	2365 pounds.
Added between A and C, ...	422 "
MnO in slag A, ...	1943 "
Per cent. MnO in slag A (see Sec. 72), . . .	11.12
Weight slag A = 1943 ÷ 0.1112, ...	17,470 pounds.

And from this we have:

Slag A.

	Weight.	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
Analysis, per cent.,	—	19.21	11.12	42.16	6.64	13.68	5.149
Pounds, . . .	17,470	3356	1943	7365	1160	2390	900

Note.—1965 pounds of ore added between A and B.

SEC. 78.—Summary of results:

	Pounds.						
Slag.	Weight.	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
A, . .	17,470	3356	1943	7365	1160	2390	900
B, . .	21,390	3502	2216	9151	1683	3484	1037
C, . .	26,254	3959	2365	11,069	2218	5340	1011
D, . .	30,780	4848	4343	12,020	3201	5125	911

The small amount of CaO and MgO in slags A and B indicates the existence of the conditions described in Sec. 70. The lime and dolomite are not wholly incorporated in the liquid portion of the slag, which is made up of all the SiO_2, MnO, FeO and P_2O_5.

SEC. 79. *The P_2O_5.*—We have found in Section 76 that the amount of P_2O_5 which is added to slag between tests B and C amounts to 56 pounds. In the same manner, we find that the amount added between A and B is 121 pounds. Comparing the results of this calculation with the values of the summary we have:

Slag.	Pounds P_2O_5 Calculation.	Summary.
A,	834	900
B,	955	1037
C,	1011	1011
D (see Sec. 73),	1092	911

We thus find that no loss of phosphorus is indicated by the results of this series, but that variations are included within the limits of error.

Sec. 80. *The FeO.*—The amount of FeO in the various slags is dependent upon the rate of oxidation of the iron in the bath, and upon the reactions between the bath and the ore which is added.

(1). Interval during melting:

Amount of FeO in slag A,	2390	pounds.
Of which the wear of bottom supplied,	87	"
Leaving as product of oxidation of the metal,	2303	"

Representing 1791 pounds of Fe, or about 0.9 per cent. of the charge.

(2). Interval between A and B:

Amount of FeO in slag A, as above,		2390	pounds.
Amount added in ore between A and B,	1737		"
Amount from wear of bottom,	87		"
Total additions of FeO,		1824	"
Total, which would be in slag B if no reactions occurred,		4214	"
FeO in slag B (see summary),		3484	"
FeO reduced between A and B,		730	"

Representing 40 per cent. of the FeO added.

(3). Interval between B and C:

Amount of FeO in slag B,		3484	pounds.
Added in ore between B and C,	685		"
Amount from wear of bottom,	87		"
Total FeO added,		772	"
Amount which would be in C if no reactions occurred,		4256	"
FeO in slag C,		5340	"
Increment between B and C,		1084	"

Allowing for the 772 pounds coming from the ore and bottom, we have 312 pounds from the bath, representing 243 pounds of Fe, or $\frac{1}{3}$ of 1 per cent. of bath.

(4). Interval between C and D:

Amount of FeO in slag C,	5340 pounds.
Amount from wear of bottom,	87 "
Amount which would be in D if no reactions occurred,	5427 "
FeO in slag D,	5125 "
FeO reduced between C and D,	302 "

These figures indicate that, as in the case of acid work, reduction of iron from the slag obtains while the bath is high in carbon:

The total FeO in slag D is,	5125 pounds.
Representing 3986 pounds Fe, or about 2 per cent. of metal charged.	
The amount of FeO from ore and bottom is,	2770 "
Hence, the amount from oxidation of bath,	2355 "
Representing 1832 pounds of Fe, or about 1.0 per cent. of the bath.	

Sec. 81.—The full calculations of the first two series are not given here, but all the data are recorded. If future experiments should indicate the volatilization of any portion of the phosphorus of the bath, or should show that the chemical methods are incorrect, these results may serve as corroborative evidence. If not, they remain as monuments to error.

The Thermal Equation.

Sec. 82. *The Thermal Equation of the Furnace—Determination of Constants.*—In the determination of the amount of heat required theoretically to melt and superheat an open-hearth charge, it is necessary to know the thermal data of the pig-iron and steel used. Unfortunately different experimenters give different results for both the critical temperatures and the determination of the values of specific heat. Taking, however, two sets of values given by good authorities, it will be found that the variations balance, to some extent, and the totals show an approximate agreement. We quote the figures given by Åkerman and those by Troilius. The latter author does not distinctly state his authority for all the values, but mentions the work of Ledebur, Favre and Silverman.

Sec. 82a. *Melting-Points:*

	Degrees C.	
	Åkerman.	Troilius.
Melting-point, pig-iron,	1200	
" " white pig,	—	1050–1100
" " gray pig,	—	1200–1300
" " steel (probably high carbon),	—	1300–1400
" " soft iron,	—	1600
Highest heat Bessemer converter,	2000	

From which we may assume as the average temperature of

Tapping an open-hearth charge,	1800 degrees C.
Melting rail-steel,	1450 " "

SEC. 82b. *Latent Heat* (calories per kilo).

	Åkerman.	Troilius.
Latent heat of fusion, pig-iron,	46	
" " " gray pig,		33
" " " white pig,		23

From which we may assume that the truth is about:

Steel,	20
Pig-iron,	30

This factor does not play an important part, and the error of this supposition will but slightly influence the result.

SEC. 82c. *Specific Heat* (calories per kilo).

	Åkerman.	Troilius.
Specific heat, pig-iron 0° to 1200°,	0.16	
" " " 1200° to 1800°,	0.21	
" " " 0° to 1500°,		0.18
" " " 1500° to 1800°,		0.20

Calculating by both sets of data we have:

	Åkerman.	Troilius.
Heating from 0° to 1800°,	318	330
Hence probable value is about		325

SEC. 82d.

	Troilius.
Specific heat, steel (probably high-carbon),	.1175
" " soft iron,	.1081
Hence probable value, solid rail steel,	.1125
" " " melted (compare Section 82c),	.1275

SEC. 83. *Calories Absorbed by Bath.*—From the foregoing data we may calculate roughly the heat absorbed in the melting and superheating of an open-hearth charge consisting of one-sixth pig-iron and five-sixths rail-steel scrap.

One kilo of mixture.	Calories.
Heating pig-iron from 0° to 1800° $= \frac{1}{6} \times 325$, ...	54
Latent heat of fusion, ...	5
Heating steel 0° to 1450° $= \frac{5}{9} \times 1450 \times .1125$, . . .	163
" " 1450° to 1800° $= \frac{5}{6} \times 350 \times .1275$, . . .	45
Latent heat of fusion, ...	17
Total per kilo, ...	284

One ton = 1020 kilos.

Hence heating and melting to 1800° C. one ton of mixture would require

$$1020 \times 284 = 290{,}000 \text{ calories.}$$

SEC. 84. *Composition and Calorific Power of Coal.*—In the *Journal of the Iron and Steel Institute* for 1887, there are given data concerning the calorific power of different coals as determined by experiment, compared with the heating power as determined by calculation. The results show that there is a field of chemical physics in which ignorance still resides. Selecting from the samples given the specimens which run between 80 and 90 per cent. carbon, as approximating to the coal used in the foregoing determinations, we have the following:

	C.	H.	C/H.	Calories per kilo. Experimental.	Theoretical.
(1)	88.5	4.41	20.0	9623	8431
(2)	87.0	4.72	18.11	9111	8404
(3)	87.5	5.10	17.1	8964	8552
(4)	82.7	5.07	16.3	8021	8154
Av.,	86.4	4.82	17.9	8930	8385

The coal used in the work given in the foregoing pages ran as follows:

C.	H.	Ash.	S+O+N.	C/H.
82.5	4.7	8.5	4.5	17.6

Hence we may assume that the heating-power of this coal is about 8500 calories per kilogramme—experimental value.

The composition would indicate a theoretical value of about 8000 calories as determined by the usual formulæ.

The method used in the foregoing pages, as shown in Section 19, gives the value 8198 calories, which will be used as a basis.

SEC. 85. *The Heat Produced by Internal Combustion.*—From the data given in Section 54 we may derive the following:

Weight 24 tons.

During melting:

	Lbs. oxidized.	Kilos oxidized.	Calorific power per kilo.	Calories produced.
Si,	219	100	7830	783,000
Mn,	473	215	2000	430,000
Fe,	523	238	1350	321,000
C,	279	127	8080	1,026,000
Total during melting,				2,560,000

During oreing:

	Lbs. oxidized.	Kilos. oxidized.	Caloric Power. per kilo.	Calories. produced.
Mn,	28	13	2000	26,000
Fe,				
C,	230	105	8080	848,000
Total during oreing,				874,000
Total during melting,				2,560,000
" during oreing,				874,000
" during heat,				3,434,000
Per ton,				143,000

SEC. 86. *A Summary of the Data Determined.*—In the foregoing pages the following facts have been determined:

Section 19.—Calorific value of coal, 8198 calories per kilo.
Section 46.—Kilos of coal per ton of product while charge is in furnace = 290.
Hence while charge is in furnace the heat produced from coal = 2,377,420.
Of this amount there is the following distribution:
Section 19.—Lost in ash as carbon, 5.9 per cent. = 140.650 calories.
Section 20.—Lost in producer, 29.2 per cent. = 694,840 calories.
Section 19.—Potential in gas, 64.9 per cent. = 1,541,930 calories.
Of these 1,541,930 calories given to the furnace, supposing that the waste-gases escape at a temperature of 300° C. in excess of the incoming gases, the following losses occur:
Section 30.—Due to specific heat of products of combustion, 16 per cent. of potential heat of gas = 246,709 calories.
Section 42.—Due to imperfect combustion, 4.5 per cent. = 69,390 calories.
Section 83.—The heat necessary for melting and heating = 290,000 calories.
Section 85.—The heat produced by internal combustion = 143,000 calories.

SEC. 87. *The General Thermal Equation of the Furnace.*—From the foregoing data we may construct the following:

Per ton of product:

	Calories.
Heat generated from coal,	2,377,420
Heat generated from internal combustion,	143,000
Total,	2,520,420

	Calories.	Per cent.
Heat lost as carbon in ash, ...	140,650	5.6
Heat lost in producer, ...	694,840	27.6
Heat lost in waste-gases, ...	246,710	9.8
Heat lost from imperfect combustion, . .	69,390	2.8
Heat lost in melting and heating, . . .	290,000	11.5
Heat lost in radiation (by difference), . .	1,078,830	42.7
	2,520,420	100.0

Sec. 88. *The Fuel Consumed in an Empty Furnace.*—From Section 87 we see that the heat produced by the coal alone for a certain production is 2,377,420 calories; the losses due to causes not connected with the bath amount to 2,230,420 calories, or 94 per cent. of the total. This ratio applies to a given time as well as to a given production. Hence the amount of coal necessary to keep the furnace continuously at a working-temperature is, by our calculation, 94 per cent of the usual consumption. But as the furnace is never kept at the working-temperature when empty for any considerable period, as, for instance, over Sunday, the radiation decreases and the consumption varies according to the length of time and the temperature maintained. When economy is carried too far in the use of coal, it will be found that the first charge will take an abnormal amount of fuel. This excess belongs to the period of stoppage, as it goes to restore the lost heat of the chambers and to build up the fuel-bed of the producer to a proper thickness. The following are records of coal consumption showing the relative amounts of coal used:

(1) During the stop, Saturday night and Sunday.

(2) During Sunday night and Monday, when the furnace was in operation, and during which time the fires and furnaces were being brought up to their normal condition:

(3) During regular and normal work.

Without attempting to reduce these data to a formula, it will be evident that the consumption of coal in keeping a furnace at a dull yellow temperature is over three-quarters as much as is necessary in melting-operations; more than this, the abnormal amount charged to Sunday night and Monday shows that a part of the fuel of this period belongs to the time of stopping. It is reasonable to suppose that to maintain a furnace at the usual working-temperature would require the full 94 per cent. called for by the previous calculations as summarized in Section 87.

	FIRST TRIAL.			SECOND TRIAL.		
	Hours.	Hoppers.	Average Per Hour.	Hours.	Hoppers.	Average Per Hour.
Saturday night	14	189	13.5	14	210	15.0
Sunday.....	11	186	16.9	11	208	18.9
	25	375	15.0	25	418	16.7
Sunday night.....	13	286	22.0	13	305	23.5
Monday	11	180	16.4	11	198	18.0
	24	466	19.4	24	503	21.0
Monday night	13	237	18.2	13	231	17.8
Tuesday.....	11	200	18.2	.....	.....	.....
Tuesday night.....	13	259	19.9	.....	.....	.....
Wednesday.....	11	177	16.1	.....	.....	.....
Wednesday night.....	13	245	18.8	.....	.....	.....
	61	1118	18.33	.....	.....	.....

The Dynamical Equation.

Sec. 89. Of an ordinary heating-furnace not using forced blast, it is common to hear it said that the chimney draws the air into the fire-box. This is a complete mistake, and since the same fallacious conception is carried into the more complicated dynamics of the regenerative furnace, it may be well to consider the simpler problem. Every piece of matter which is hotter than its immediate environment tends to warm the gases in contact with it and cause them to rise. This necessarily tends to produce a vacuum beneath and a plenum above. Such is the situation in the fire-box of a furnace. The fire-bed may be looked upon as a blower, drawing in the air at one end of a cylinder and forcing it out at the other.

If the outlet is ample the pressure will be slight; if it is small the pressure will be high. If it is blocked altogether the blower will be unable to work, and there will be a backing-down of the gases coming from the distillation of the volatile hydrocarbons.

We may put another blower on the opposite end of the furnace, to take the products of our first blower and force them elsewhere.

This we do when we put a chimney there. If it has ample inlet and outlet, if it is high enough and hot enough, it will be more powerful than the blower from which it derives its supply and there will be a vacuum above the fire. These conditions we try to produce when we start the fire.

But if we reduce the outlet sufficiently we may reach that point where the second blower is not quite equal to the first, and the result will be a slight plenum in the furnace, which tries to relieve itself by squeezing out of the doors and other openings.

As long as there is an outward pressure at the doors (and a furnace is usually run in that condition) it is an absurdity to speak of the stack drawing the air into the fire. The chimney takes away from the furnace what is put in by the fire. Like every blower or pump the fire will force more against a diminished pressure, and since increased draught of the chimney will cause diminished pressure, we may carelessly, but always incorrectly, speak of the direct influence of the damper upon the grate.

A producer is a heating-furnace with an abnormally thick bed of coal in the fire-box. The products of combustion are forced into the tube and finally escape through the valves into the furnace. It sometimes happens, owing to an insufficient producer-power, or to obstructions in the tubes, or other causes, that there is a partial vacuum, but there should be a plenum in all parts of the tube.

In order that a strong outward pressure may be had at the valve and a lower pressure obstruct the draught of the producer, it is well to make the conducting pipe considerably above the level of the producer. This will give an uptake for the hot producer-gases and a downtake for them after they have partially cooled, and we shall obtain a syphonic action which will facilitate, but is not necessary to good work.

As soon as the gas goes through the valves it meets with the pulling power exerted by the regenerative chamber. By this action it is drawn into the chamber and pushed upward into the furnace. In this process the mass of checkers acts precisely like the bed of coal in a heating furnace. Whether a plenum is caused in the furnace-chamber above depends upon the conditions of the opposite end of the furnace construction. The chamber on the outgoing end is also trying to force gases into the upper room, but the connected stack is supposed to have sufficient draught to overcome this tendency and also to be able to handle the gases pushed in by the chamber on the

incoming end. By regulating the draught-area in the stack, a plenum or a vacuum may be created in the upper room.

It should hardly be necessary to repeat that an action which *per se* causes a vacuum, can by no alteration short of complete subversion create a plenum.

It follows, therefore, that the gases entering the furnace come in through influences entirely extraneous to the action of the draught.

The above remarks apply to a furnace in running order with hot chambers. If the furnace is just starting, the conditions are different, and the pumping action of the chamber on the incoming end may be almost entirely absent. Under these circumstances it is perfectly possible, and sometimes necessary, to have the gas pulled in by the draught of the stack.

THE BALL-NORTON ELECTRO-MAGNETIC SEPARATOR.

BY C. M. BALL, TROY, N. Y.

(New York Meeting, September, 1890.)

THE magnetic concentration of iron-ores has been so often and so widely studied and discussed among the members of the Institute that any remarks concerning its general importance, from an economic standpoint, would be superfluous; but it is thought that a description of the Ball and Norton ore-separator, designed for cobbing and separating magnetic iron-ores, indicating its essential differences from other machines and the advantages claimed for it, together with some account of recent results obtained in its application to the concentration of such ores, will prove of interest.

This machine is the joint invention of Mr. Sheldon Norton, of Hokendauqua, Pa., and the writer, and has been patented in the United States, Canada, and other countries.

Of the accompanying illustrations, Fig. 1 represents a longitudinal vertical section of the perfected ore-separator, adapted for separating fine ore, and Fig. 2 the same, adapted for cobbing ore of the size of stove- or chestnut-coal. The two designs are identical in principle, and vary from each other in minor details only.

The apparatus consists of a partially-closed chest, having an opening at *f*, from the feed-hopper, *h*, through which the ore is delivered to the machine from an ore-pocket or storage-bin provided with means for regulating the flow of ore, so that, when the machine

is in operation, the hopper is kept always full. Other openings are provided for the discharge, at *t*, of tailings; at *m*, of middlings; and at *c*, of concentrates; also, at *e*, for allowing free ingress of air to the chest at that point, and at *s*, where a powerful exhaust-fan is connected. The openings at *t* and *m* are kept sealed against ingress of air at those points by means of the hinged and weighted valves, *v*, *v*, which discharge the products from the hoppers, *p* and *k*, continuously, and in the same proportion as received from above, when a sufficient weight has accumulated upon the inside to cause the contents of the hoppers to leak by the valves.

The machine is also provided, as shown, with two drums, Nos. 1

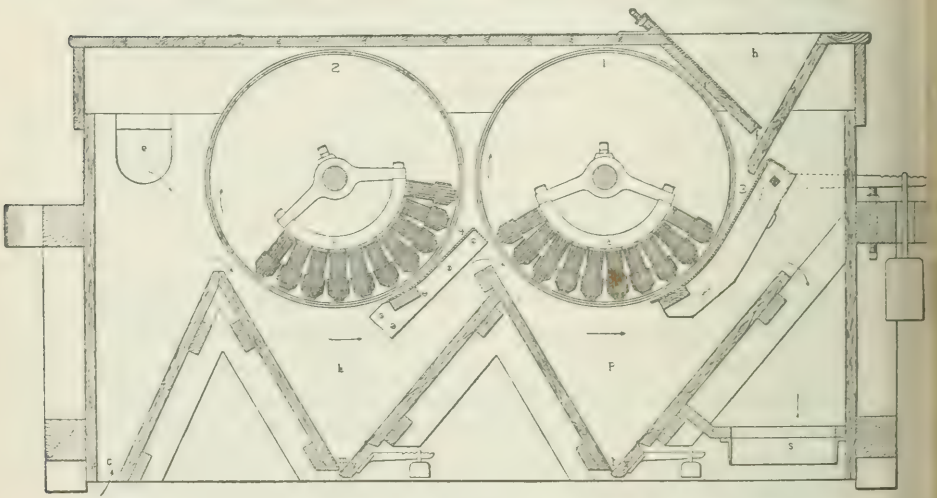


Fig 1.

Ball Norton & Parker
Magnetic Separator

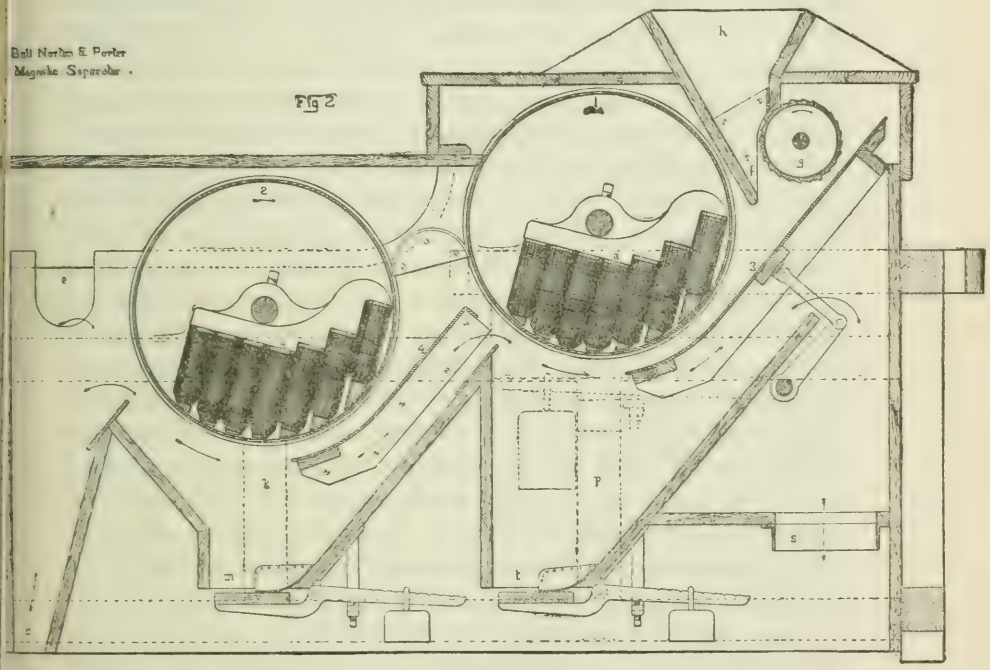
and 2, turning upon the shafts, *i* and *j*. These shafts, together with the magnets, *a* and *b*, which they also serve to support, stand still, while the drums may be rapidly revolved around the magnets and out of contact therewith.

It will be noticed that the magnet occupies a sector of the drum, the proportions being such that, approximately, one-third of the periphery of the drum is within the influence of the magnetic field, while the upper two-thirds is outside of the field and removed from the magnetic influence. The magnet is so constructed as to present a series of poles of alternately opposite polarity near the inner surface of the drum. In accordance with the well-known phenomena of magnetic attraction, which in the case of powerful magnets is exerted at a considerable distance from the magnetic poles, any magnetizable

matter brought near the outer surface of the drum, within the arc covered by the magnet, will be powerfully attracted and drawn into firm contact with the outer surface of the drum.

These drums are composed of a non-metallic and neutral material, such as wood, paper, etc., and they turn in the direction indicated by the arrows.

Just below the feed-hopper an apron of neutral metal, No. 3, is arranged, curving downward and forward in the direction of the rotation of the drum, its lower portion describing a short arc con-



centric to the surface of the drum. This serves as a chute to direct the stream of ore falling from the feed-hopper within the influence of the first two or three poles of the magnet. A similar but somewhat shorter apron, No. 4, is arranged in like relation to the second drum and magnet, *b*.

Attention is called to the provisions for applying and directing the air-current, which fulfils an important function in the process of separating ore as developed in this apparatus. The air may enter freely at the openings *c* and *e*, and is drawn through and out of the chest by the action of a powerful exhaust-fan connected at *s*. The air which passes through the chest containing the drums and

magnets must, of necessity, follow the course indicated by the arrows in the space below the drums.

When the machine is put in operation the magnets are excited, the drums are revolved in the direction before indicated, the air-current is established through the machine in a direction opposite to that of the rotation of the drums, and ore is supplied through the feed-hopper which is kept always full. The ore passes down the chute under the first drum, and, as soon as it comes within the influence of the magnet, the magnetizable portions are drawn into contact with the drum and through friction upon its surface take on the forward movement of the drum. At this stage, a curious and important result takes place. In accordance with the well-known laws of magnetic induction, a particle of iron brought near a magnet itself becomes a magnet by induction, and the magnetic force tends to bring the longer axis of this induced magnet into a position as nearly as possible parallel to the direction of the magnetic force. So a single particle, or many particles, brought near one pole of a magnet stand on end, as it were; and, in the case of many particles simultaneously influenced by the same pole, they form tufts standing out from the pole, their outer ends repelling each other, but all pointing in the direction of the lines of force towards some focus of opposite magnetic polarity. In conformity with this law, the particles of ore in contact with the drum opposite one of the poles of the magnet stand on end, forming tufts, spreading away from each other at their outer ends. As they are drawn along, however, by friction against the moving drum, when they get to a point midway between two poles they lie down flat against the surface of the drum, and, as they are drawn still further along, they again stand on end—but this time the other end out. So, in passing through the magnetic field, they are tumbled end over end as many times as there are poles in the field. The result is, that every time they are reversed in position opportunity is afforded for any non-magnetic particles of gangue, which may have been entangled with the ore, to fall away from the tufts of magnetite; and this result is still further facilitated by the centrifugal tendency and by the counter-current of air.

When the ore reaches the limit of the arc covered by the magnetic field it is no longer attracted, and takes on a tangential movement, which carries it away from the drum. It has now, however, passed the edge of the second apron, and, on leaving the first drum, comes within the influence of the magnet of the second drum, where similar

operations are repeated, a portion being finally discharged as concentrate at *c*. The function of the second drum and magnet is to differentiate the product from the first drum into two portions, which may be conveniently designated as middlings, discharged at *m*, and concentrate, discharged at *c*. The middlings consist of particles of ore with adhering portions of gangue, which may require a little finer crushing to effect their mechanical liberation; or, they may consist in part of iron compounds having a smaller degree of magnetic susceptibility than the pure magnetite. The separation of the middlings from the mass delivered to the second drum may be effected in two ways: If the drums have the same speed of rotation, a weaker magnetism in the second magnet will allow these less magnetic particles to drop away; or, if the magnets have approximately the same force in the two drums, a higher speed of rotation of the second drum will throw these particles off by reason of the centrifugal force overpowering the centripetal magnetic attraction, the magnet having the smallest influence upon the leaner portions of the mass.

Modifications of this apparatus have been designed, which adapt it for use in conjunction with a stream of water mingled with the ore, so constituting a wet process.

In designing this apparatus, it has been assumed that, in order to separate effectively a mass of commingled ore and gangue, containing a large percentage of fine dust, positive and contending forces must necessarily be brought into active play.

The importance and special efficiency of the means provided for suppressing the movement and discharge, along with the ore, of the finer particles of gangue, and of the means for differentiating the product into several portions graded according to the percentage of iron in each portion, and its magnetic susceptibility, should not be overlooked. It is upon the application of these means that the ability depends to perform the critical operation of converting non-Bessemer into Bessemer ores by the elimination of phosphorus-, sulphur-, and titanium-bearing compounds.

In the concentration of ores by the dry process the control of the dust developed in the rapid movement of large quantities of ore, is generally difficult. When the ore reaches this machine, the finer portions of the material are as readily and perfectly dealt with and separated as the coarser, the concentrates, however finely pulverized, being delivered from the machine in an absolutely clean condition, and free from non-magnetic dust and the dust carried away by the air-current being also as wholly free from any magnetic material.

The peculiar action of the ore in passing through the complex or multipolar magnetic field has already been described. Upon the mode of arrangement of the poles the efficiency of the apparatus largely depends. In this machine the magnetism of the field of attraction performs a two-fold function, *viz.*, to attract the magnetizable matter of the ore and, as it is moved through the machine by friction upon the revolving drum, to turn it over and over so as to allow the gangue to fall away, and also to permit the air-current to take effect on all sides of every particle of the ore.

If the importance of this operation is once recognized, it will be readily understood that there can be only one possible efficient arrangement of the poles, and that it is not a matter of indifference whether they conform to to this or some other plan.

The positive character of the functions above enumerated renders it possible to make an effective separation under the conditions of a very heavy supply of ore to the machine. The easy working-capacity of a machine having drums of 24 inches diameter and 24 inches working-face is from 15 to 20 tons per hour of ore granulated to pass 16- to 20-mesh screens.

The power required is from 1 to $1\frac{1}{2}$ horse-power in electricity for each drum, and $\frac{1}{2}$ to $\frac{3}{4}$ horse-power to drive the machine.

These machines have been applied to the treatment of a considerable number of ores with results shown in the appended tables. The most remarkable of these was the conversion of Port Henry Old Bed ore into a Bessemer ore, carrying Fe, 71.10; P, 0.037. This concentration was made from the crude ore, carrying Fe, 58.7; P, 2.25; the Bessemer concentrate representing about 65 per cent. of the original mass.

An earlier type of this machine was in use during nearly the whole of the year 1889, at Benson Mines, St. Lawrence county, N. Y., concentrating the lean magnetite mined there, which in the crude ore carries about 40 per cent. Fe, and is above the Bessemer limit in both phosphorus and sulphur. The concentrated product found a market at Pittsburgh, Pa. The analyses in Table A show the general character of the results; those in Table B, the results obtained there with the improved machines of the later type; and the analyses in Table C, the results of latest concentrations.

The new works at this mine (a detailed description of which it is hoped may be offered at a future meeting) are provided with crushing-machinery having sufficient capacity to crush daily 800 to 1000 tons of ore, sized to 16-mesh and finer, and the separation of this

quantity of ore is to be effected at the outset with three separators of the type represented in Fig. 1, having a working-face on the drums of 24 inches.

In the minds of some persons who have been studying the general subject of ore-concentration, and employing the apparatus hitherto available, inability to convert some of the well-known ores high in phosphorus and sulphur, such as Port Henry Old Bed, Little River, Croton, etc., from non-Bessemer into Bessemer ores, has led to the inference that some as yet undiscovered refinement of preparation was essential to their successful treatment. The entirely successful and economical management of these ores with this machine has proved that the essential matter in the problem was to provide a separator that would separate ore that had been properly prepared. The preparation of any particular ore being wholly dependent upon its physical and chemical characteristics, its final separation would depend upon the efficiency of the separator, which would constitute the last and supporting link in the chain of operations. It is believed that this machine offers a practical solution of the problem of concentrating magnetic ore such as has not been hitherto provided.

Table A.

Analyses from Tests with the Ball-Norton Electro-Magnetic Separator, Earlier Form.

Mine.	Crude Ore.		Concentrates.		Tailings.		
	Fe.	P.	Fe.	P.	Fe.	P.	Per ct. of Original Weight.
Benson Mines, Little River....	45.48	0.158	61.40	0.042	5.60	1.25	29.0
	41.24	0.25	63.31	0.042	5.47	.....	.....
	40.51	0.19	63.68	0.045	3.87	.....	.....
	*43.02	0.19	62.10	0.022	.....	.....	.....
Port Henry, New Bed.....	47.70	0.025	70.40	0.011	4.40	0.061	38.0
" " Old Bed, 21.....	58.30	2.18	71.10	0.11	8.70	10.61	19.3
Flagler, Vt., ore	23.74	0.408	64.87	0.084	4.13	0.465	68.44
Chateaugay, N. Y. 16-mesh.....	44.56	.....	66.69	.....	1.26	.....	44.0
Chateaugay, N. Y. 4-mesh.....	33.99	.....	59.58	.....	10.64	.....	52.0
Chateaugay, N. Y. 8-mesh.....	31.23	.....	62.11	.....	7.32	.....	61.11
Chateaugay, N. Y. 16-mesh.....	36.23	.....	66.98	.....	8.12	.....	59.0
Chateaugay, N. Y. 16-mesh.....	†19.60	.....	60.48	.....	8.96	.....	84.375
Tilly Foster.....	50.25	.....	61.12	.....	7.88	.....	20.0
Beach Glen, N. J., Ore.....	‡50.81	0.042	64.88	0.02	7.06	0.082	25.0

* Average of eight lots shipped from mine. The crude ore contained also: SiO_2, 28.31; S 0.856; the concentrates, SiO_2, 5.94; S, 0.127.

† Tailings from Conkling jigs.

‡ Sulphur as follows: in crude, 0.031; in concentrates, .022; in tailings, 0.02.

Table B.

Analyses from Tests with the Ball-Norton Electro-Magnetic Separator, Latest Form.

Mine.	Crude Ore.		Concentrates.				Tailings.		Analyst.
	Fe.	P.	Fe.	P.	SiO_2	S.	Fe.	Per ct. of Original Weight.	
Benson Mines, Little River, 16-mesh	45.48	.....	68.45	0.022	4.78	.....	.....	.....	Donohue.
Benson Mines, Little River, 16-mesh		.....	67.49	0.016	5.64	0.32	.....	.....	
Benson Mines, Little River, 20-mesh		.....	68.62	0.0106	4.18	0.34	.....	.....	
Benson Mines, Little River, 20-mesh		.....	67.81	0.013	5.19	0.27	3.39*	.....	
Chateaugay tailings	11.80	.....	68.365	0.008	4.31	.....	4.33	89.0	Wuth.
Port Henry, New Bed	47.70	.....	70.9	0.0089	1.48	.....	.....	38.0	Wuth.
" " Old Bed, 21	58.70	2.25	71.1	0.037	.....	.....	.....	20.0	Wuth.
Croton, N. Y., ore	†42.99	0.153	69.86	0.021	1.71	0.04	7.95	47.0	Wuth.

Table C.

Analyses from Latest Concentrations with the Ball-Norton Electro-Magnetic Separator, Latest Form.

Mine.	Crude Ore.			Concentrates.				Middlings.	Tailings.	Analyst.
	Fe.	P.	S.	Fe.	P.	S.	SiO_2	Fe.	Fe.	
Republic Magnetic	69.8	.031	.....	71.12	.015	.035	.540	.....	.....	Wuth.
Gellivare, Swedish	65.76	.954	.....	71.44	.032	.020	.390	.....	.....	Wuth.
Orchard, Port Oram, N. J.‡	57.13	1.482	.097	70.11	.068	.....	.720	.....	.....	Moore.
Sea Sand	13.04	.....	.....	71.60	Trace	.....	.....	.....	.....	Sherrerd.
Benson Mines, Lean Ore	36.80	.093	.....	67.22	.016	.....	2.54	.....	.....	Moore.
Tilly Foster, Lean Ore	31.88	.....	.....	64.1	.010	.....	4.40	.....	.....	Riotte.
Tilly Foster, Concentrate from Conkling Separators	43.30	.....	.....	67.4	.010	.....	.....	.....	.....	Riotte.
Tilly Foster, Tailings from Conkling Separators	15.80	.....	.....	65.1	.010	.....	.....	.....	.....	Riotte.
Humboldt, Mich., Concentrate from Edison-Dickson Magnetic Separator.	61.60	.046	.....	68.06	.031	.....	.....	.....	.....	Wuth.

* The sulphides and hornblende account for more than 2 out of the 3.39 iron in these tailings.

† Sulphur in crude, 0.30.

‡ Dr. Otto Wuth, of Pittsburgh, reports that this ore contains some phosphate of iron.

AMALGAMATION AT THE COMSTOCK LODE, NEVADA: A HISTORICAL SKETCH OF MILLING OPERATIONS AT WASHOE, AND AN ACCOUNT OF THE TREATMENT OF TAILINGS AT THE LYON MILL, DAYTON.

BY A. D. HODGES, JR., SAN FRANCISCO, CAL.

(New York Meeting, September, 1890.)

I. Early Workings of Placers.

In May, 1850, the first gold from the Comstock lode was discovered in the sands near the Carson river. It had been washed down by natural agencies, through Gold Cañon, from the decomposed rock of the ledge above. Within a few weeks, the precious metal was traced more than three miles in shallow deposits along the bed of the cañon, from the river to the Devil's Gate—a narrow, rocky cleft, notable to the teamsters of a later day chiefly as the place where "Uncle Jimmy Fair" collected high tolls, and could not reduce the rates, because "this was all he had for the education of his poor children."

From 1850 to 1857, a fluctuating band of miners, whose number varied from twenty to two hundred, washed the Gold Cañon placers with rockers and long-toms, making four to five dollars a day each until the last-named year, when the deposits were in great measure exhausted. The chief settlement of these miners was at Johntown. During only a third of the year was the water-supply sufficient for their work. The remainder of the time they utilized in desultory prospecting of the hills and in occasional forays for supplies among the Mormons in Carson valley.

In 1857, the Grosch brothers discovered the Comstock lode at some point, now unknown, in Gold Hill, but their discovery perished with them. Early in 1858, the "Pioneer Quartz Company," having located several of the small surface-veins around Silver City (whose connection with the main lode has never been definitely shown), built an arrastra, and worked some of their ore; but the returns were so small that they soon abandoned this business. The following winter, James Finney and his partners found gold in place in

crumbling surface-rock at Gold Hill, located claims there, and worked the auriferous material in rockers with good results. Other claims were speedily taken up, and in the spring of 1859 half a dozen arrastras were in profitable operation. In the following summer arrastras were in use also at the new discoveries in Virginia City, the number at Gold Hill was largely increased, and a few were started down on the Carson river.

II. The Discovery of the Washoe Process.

Up to this time the miners had sought for gold only, with no thoughts of any other metal than that which the California mines were yielding. But the assays made in June, 1859, by Melville Atwood, showed that the Comstock lode was a rich auriferous *silver*-deposit, and the treatment of silver sulphurets became an important question. Samples of the ores were sent to be tested in the various metallurgical works of California, and some of these samples came into the possession of Almarin B. Paul, an able and energetic millman of Nevada City. Paul was fully posted on all the details of gold-milling as practiced in California, and had read various works on the metallurgy of silver. It occurred to him to treat the sulphurets with the chemicals of the patio process in the pans then used to some extent in the gold-mills; and the results of his tests, made in the autumn of 1859, convinced him that this treatment could be applied with success to the Comstock ores. With characteristic energy, he organized the Washoe Gold and Silver Mining Company, No. 1, and on May 25, 1860, began work on a mill-site near the Devil's Gate, in Gold Cañon. On June 7th, he ordered his machinery at the Miners' Foundry, San Francisco, his enthusiasm and confidence being undisturbed by the general distrust and even ridicule with which his plans had been received. On June 12th, he signed contracts to work several thousand tons of ore from the Gold Hill claims (at $30 per ton), the contracts being made conditional on the completion of his mill within 60 days from date—a condition which only a bold man would accept in those times, when material for construction was so far away and transportation was so difficult and uncertain. On August 11th, just within the limits of his contracts, the 24 stamps of his Pioneer mill commenced to drop. Three hours later, Charles S. Coover and Dr. E. B. Harris, who had been inspired to rivalry by Paul's example, set in motion the machinery of the little 8-stamp mill which they had built near by, in Gold Hill.

Paul crushed the ore dry in his batteries, and amalgamated it in Knox pans, which were 4 feet in diameter by 14 inches deep, and held charges of about 300 pounds each. The light iron muller, revolving twelve or thirteen times per minute, did not grind, but served merely to stir the pulp for the purpose of effecting a distribution of the quicksilver throughout the mass. Copper plates were fastened in the pans in order to hasten the reduction of the silver. A part of the amalgam adhered to these plates and was scraped off twice a day. The rest was drawn off at intervals through a discharge-hole.

Each charge was treated with 40 pounds of quicksilver, a pint of salt and a few ounces of copper filings or copper sulphate. The pans were set on brick furnaces and heated by wood fires, and the water used was warmed by the exhaust from the engines. At a later date, in place of the brick furnaces, iron stoves were used, with circular flues, through which the heat and smoke were led beneath the pans. Then Paul suggested steam-chambers fitted to the pan-bottoms, a suggestion adopted and perfected by Howland.

Thus Paul invented the Washoe Process. With due acknowledgment of what others had done, to him should be given the credit of being the first to work out, and apply successfully on a large scale, the method of treating silver-ores by pan-amalgamation.

III. The Building of Mills.

Paul's first mill-runs, and also those of Coover and Harris, gave satisfactory results, and increased materially the mining rush and excitement which had now set in. Claims were taken up all over the country in the neighborhood of the lode, and mill-sites were located wherever there was any available water. Quite a number of mills were built in 1860, Paul's company (the Washoe Gold and Silver Mining Company, No. 1) at once putting up a large establishment at Gold Hill. In 1861 there were already seventy-six mills erected for the purpose of treating Comstock ore, with 1153 stamps and an estimated daily crushing-capacity of 1200 tons; and twenty more were planned or in process of construction. There were also forty or fifty arrastras and several patio-yards. The mills lined Seven-Mile, Six-Mile and Gold Cañons, from Virginia City to the Carson river, and the Carson river, from above Empire to below Dayton; and several were placed more than a dozen miles away, at Ophir and Franktown, on the borders of Washoe Lake. The mill-building mania kept pace with the mining-excitement which culminated in 1863, and was followed by a stock-panic in 1864,

although the bullion-production of the lode continued to increase. In this time some seven hundred mining companies had been incorporated to operate on the Comstock; but of these only a hundred possessed prospected mines, and only fourteen had paid dividends. The number of mills built approximated one hundred and fifty, the exact figures being unobtainable, on account of the indefinitenes and variance of the published statistics. Their capacity was always in advance of the ore-supply. Many were erected for mines which never produced anything except litigation and assessments. A large number were hastily and poorly built and, as rapid improvements in machinery and construction were made, soon became antiquated; but some, notably the Ophir and the Gould and Curry, were erected in a most gorgeously expensive manner. The majority were pecuniary failures from the start; but, in the more fortunate cases, where there was a constant supply of rich ore, with charges varying from $20 to $30 per ton, large profits were obtained.

The first published list of Washoe Mills, with full details, so far as I know, was that given in the *Mining and Scientific Press*, of September 29, 1866, and this is avowedly incomplete, it being stated that there were "a few small mills, running irregularly, not enumerated." The total number given is considerably less than that contained in the "Surveyor-General's Report," for 1865, where eighty mills are mentioned. The Report of J. Ross Browne, United States Commissioner of Mining Statistics, under date of March 5, 1868, states (pp. 324–328) that there were in Storey, Lyon, Ormsby and Washoe counties, in 1867, 122 mills, with 1921 stamps. The same report (pp. 352–360) gives a detailed list of the mills in the same counties which were crushing ore from the Comstock Lode in 1866. Here, the numbers are 77 mills and 1462 stamps. According to this last list, there were 7 mills with 40 to 80 stamps each, 2 mills with 30 stamps each, 20 mills with 20 to 25 stamps each, 39 mills with 9 to 18 stamps each, 6 mills with 8 stamps each, 3 mills with 5 stamps each.

The seven largest mills mentioned were the following:

1. The Imperial mill, 44 stamps, at Lower Gold Hill, built in 1860. I have always understood that this was the second mill erected by Almarin B. Paul, who, in different publications, is said to have put in 48 and 64 stamps. The mill came into the possession of the Imperial Mining Company, and ran for many years.

2. The Gould and Curry mill, 80 stamps, at the junction of Six-Mile and Seven-Mile Cañons, on the site occupied later by the

Omega Tailings mill, was built in 1861 by the Gould and Curry Mining company. Its costly construction and brief career have been described frequently.

3. The Ophir reduction-works, 72 stamps, with furnaces, barrels, patio-yards, etc., erected in 1861 at Ophir City, in Washoe valley, by the Ophir Mining Company, rivalled the Gould and Curry mill in expense of building and shortness of existence.

4. The Rock Point mill was built in 1861 on the Carson river, near Dayton. It had 40 stamps, which were increased to 56 in 1864. It belonged to the Imperial Mining Company and had a successful career until the building of the Virginia and Truckee railroad diverted its supplies to more accessible establishments.

5. The Mexican mill, 44 stamps, on the Carson river, at Empire City, was built in 1862 and was quite noted in early days. It was owned originally by private parties and employed educated metallurgists. Here experiments were made and processes tried which aided essentially in the development of silver-amalgamation at the Comstock. In 1868 the mill was bought by the Yellow Jacket Mining Company. Subsequently, it passed into the hands of the "bonanza firm." It was running at last accounts.

6. The Washoe reduction-works, better known as Dall's mill, 60 stamps, were built in 1863 at Franktown, in Washoe valley, by Captain J. H. Dall, after the general model of the Ophir reduction-works, and ran as a custom-mill. They were destroyed by fire in December, 1866, but rebuilt the next year and maintained a precarious existence until June, 1871, when they were again burned down.

7. The Morgan mill, afterwards known as the Yellow Jacket, on the Carson river, at Empire City, was built as a custom-mill, with 20 stamps, by James Morgan, in 1864. It was sold to the Yellow Jacket Mining Company in 1865, and enlarged to 40 stamps, 20 of which weighed 1050 pounds each. In 1871 it passed into the hands of the Nevada Mill Company and worked ores for this company until 1875, when it was purchased by the Pacific Mill Company, which still owns and operates it.

IV. Milling Combinations.

A new era in the mill-business may be dated from the advent, in 1864, of William Sharon as agent, in Virginia City, of the Bank of California. Sharon was a shrewd financier and keen speculator, was backed by William Ralston, manager of the parent bank in San Francisco, and had an enthusiastic faith in the Comstock

lode. When he arrived, although stocks were low and mining depressed, he unhesitatingly loaned money to mining companies and mill-owners at much lower rates than had previously prevailed. The rate of interest had been from 3 to 5 per cent. per month. Sharon asked 2 per cent., and soon had put out large sums at this figure.

Worked-out mines and idle mills being worse than valueless as securities, and no new bonanzas coming to light, while assessments continued frequent, the bank seemed before long to be on the point of sustaining heavy losses, and many of its stockholders became frightened. But Sharon possessed the confidence of those holding the largest interests in the bank, and with their aid continued to carry out the plans he had formed. He and his friends, known popularly as the "Bank Ring," secured control of the principal mines, buying the shares at low figures, and by able stock-manipulation not only guarded against loss, but made very large profits. The next step was to acquire mills and form a milling company under the corporate name of the Union Mining and Milling Company. The mills were obtained, first by foreclosing mortgages and then by a sort of "freezing-out" process. The number of these establishments in Washoe had been steadily decreasing since 1863, but the capacity of those standing had been enlarged, so that competition was sharp and milling-charges were falling. In 1865, when the effects of Sharon's plans commenced to be felt (and these plans included a business-like working of the mines), the price of crushing fell from $25 and $22 per ton to $17 and $18. This year, according to the statements of Samuel Bowles, in the Springfield *Republican*, there were 77 quartz-mills, with 1019 stamps, working Comstock ore, capable of crushing daily about 1850 tons, but running at only two-thirds capacity. Of these mills, 22 were connected with mines and 55 were doing custom-work. The troubles of the custom-mills were soon greatly increased by the refusal of the mines controlled by Sharon and his associates to give them ores at any price; and thus many mill-owners were obliged to sell their properties to the Union Mill Company (or to join it) on such terms as this company would allow. The Union Mining and Milling Company was formally incorporated in 1867, after 7 mills had been taken by foreclosure by the Bank. These 7 mills were at once transferred to the new company, whereupon the price of milling asked by outsiders fell to $15, and then to $10 per ton. Within a couple of years more the company had acquired "upwards

of 30 mills" (*Min. and Sci. Press*, August 24, 1869), and for a time had almost a monopoly of the milling business at the Comstock. Possessing ample capital and doing a profitable business, it made many improvements in the construction of mills, and in 1871, built the largest and best establishment of the kind in Washoe,*—the Eureka mill, on the Carson river, with 60 stamps of 940 pounds each, and 24 improved pans. The motive power was furnished by a 52-inch Leffel turbine placed with its shaft horizontal,—the first time, it was said, this had ever been done.

Sharon aimed at complete sovereignty over the Comstock, but the kingdom was too large and the fortunes of the various mines were too fluctuating for him to be able to hold everything against his rivals. Not being a miner, he had to depend on others for information concerning the interior conditions of the mines, and could not always be sure of the correctness of the information thus derived. He openly accused Jones of deceiving him in 1871, concerning the Crown Point mine, and a bitter controversy ensued. He also had a public discussion with John Mackey as to whether this gentleman had not induced one of Sharon's experts to give deceptive reports, and is said to have declared that he would "have John packing his dinner-pail yet"—a prophecy unfulfilled at last accounts.

The first formidable competitor to come into general notice was the Nevada Mill Company, formed by Alvinza Hayward, of San Francisco, and John P. Jones, of Gold Hill, who, in 1871, wrested from Sharon the control of the Crown Point mine and diverted large supplies of rich ores to the mills which they acquired. This mill-company controlled by purchase and lease as many as 15 mills at one period. Among these, the most noted were the Morgan, the Mexican, and the Brunswick. The original Brunswick mill, built in 1863, was an 8-stamp establishment. Passing into Sharon's hands, it was rebuilt and improved. As remodelled in 1871 by the Nevada Mill Company, it had 56 stamps of 850 pounds, 26 Horn pans with settlers, etc., and vied with the Eureka mill in excellence of construction. Its last runs were for account of the bonanza firm.

Jones and Hayward did not secure many Comstock mines. When the Crown Point bonanza was exhausted, their controlling influence was diverted to other districts, and their most important mills were taken over by the next company to be named.

* The old Gould and Curry mill had 80 stamps, the Ophir reduction-works had 72, and Dall's mill had 60; but none of these were so well equipped or had the capacity of the Eureka, and the first two had been destroyed at this date.

In 1869, James G. Fair and John W. Mackey, of Virginia City, had secured control of the Hale and Norcross mine, then in bonanza, and associated themselves with James C. Flood and William O'Brien, of San Francisco. These associates formed the Pacific Mill Company, and purchased several mills. They next gained possession of the Savage and the Bullion mines; but they obtained no ore from the latter, and only low-grade material from the former; and before long the Hale and Norcross ore-body was exhausted. While in search of new properties, in 1871, they acquired the Virginia Consolidated (afterwards divided into the California and the Consolidated Virginia mines), where the Big Bonanza was developed in 1873. The history of the Bonanza Firm is well known. The Pacific Mill Company built the famous Consolidated Virginia and California mills, the most noted quartz mills in the world, and acquired a supremacy surpassing that of the Union Mining and Milling Company.

V. Various Amalgamation Processes Tried.

Paul's results in 1860 were accepted by the great majority of miners as fully proving the sufficiency of his process. But the largest companies, the Ophir and the Gould and Curry, with exaggerated ideas of the richness of their ores, and under the influence of conservative ideas, preferred to adopt methods which had the advantage of greater age.

In 1860, the Maldonado Brothers, who were Mexicans, put up extensive yards in Virginia City to work the ores of the Mexican mine by the patio process. Their example was followed by a few others, among them the Ophir company, which tried this method at its large establishment at Washoe valley. The patio process is slow, imperfect, and expensive. Brought into competition at the Comstock with other methods, it proved a failure, and was soon abandoned.

The "Freiberg process," as it was commonly called—chloridizing-roasting, and amalgamation in barrels—was adopted at the Ophir works and at Capt. Dall's mill, both in Washoe valley, at the Mexican mill in Empire, and at the Central mill in Virginia City. Küstel & Co., and Sutro & Co., in their Dayton mills, roasted their ores, and then amalgamated in pans. Roasting was very expensive at the Comstock. Partly on account of the cost, and partly from the desire to get bullion-returns as speedily as possible, the Freiberg process was given up. Dall alone retained it to the end; but his mill received only the richest grades of ores, which could

afford to pay high transportation-rates and a milling-charge of $45 per ton; and these ores became very rare.

The Gould and Curry mill adopted the Veatch process, which was about the same as the last, except that steam-tubs were substituted for barrels. The Central mill used this process in the beginning. In neither mill was the method a financial success. The Gould and Curry mill changed to pan-amalgamation, and the Central made trial of the Freiberg process.

About 1866, a smelting-furnace was built near Galena, in Washoe valley, and run fitfully for a year or two.

Electrical and other methods, most of them fearfully and wonderfully conceived, have had brief trials at the Comstock. But all yielded to the Washoe process, which, working rapidly and neatly, and being so well adapted to local conditions, soon was in universal use at the lode.

VI. Mechanical Improvements.

Paul's Pioneer mill, so hastily planned and built, was naturally very imperfect. In the structures which followed it, great mechanical improvements were rapidly and continuously made. The stamp, which early in the '60's crushed only 1 ton per day, in 1878 was crushing 4 tons daily through No. 5 slotted screen. In his second mill, Paul used the Howland rotary battery; but this contrivance found only very limited acceptance at the Comstock. Steam-stamps were tried once at Silver City, but were adjudged a failure after a very short run. The ultimate Comstock verdict has been in favor of 850- to 900-pound stamps, dropping about 100 times a minute, crushing wet through a slotted Russia-iron screen and working on a very solid foundation. The latest mills have mortar-blocks 16 feet long.

The pan received more attention than any other part of the mill, and numberless patterns were invented and tested. It received a variety of dimensions and proportions, was used to do all the crushing, none of the crushing, only a part of the crushing; was employed to effect the complete process of amalgamation, or was supplemented with settlers, agitators, concentrators, etc. It had flat bottoms, and curved bottoms; and the wings, mullers, shoes, and dies, in short all its component parts, were varied in shape, in accordance with mathematical theories or personal idiosyncrasies. To describe each style of pan made and tried in practical work would demand large space. Some idea may be obtained of the diversity of opinion among mill-men concerning the construction

and proper work of the pan from the following list of the amalgamating apparatus in use at the Comstock in 1866:

In the 62 mills, described in the *Mining and Scientific Press* of September 29, 1866, there were (besides 25 roasting-furnaces and 61 barrels) 10 rock-breakers, 1280 stamps, 1032 pans and tubs, 227 settlers, 44 agitators and 2 concentrators.

The 1032 pans and tubs were thus classified: 83 tubs, 13 grinding pans, 337 Knox pans, 226 Wheeler pans, 188 Hepburn pans, 66 Varney pans, 24 Wakelee pans, 56 "plain" pans and 39 pans not specified.

According to the best Comstock practice of the present day, the mills are given full stamp-capacity, and only a comparatively small amount of grinding is done in the pans. The pan-construction is solid and simple. The bottom is flat and the muller is plain, with ample spaces between it and the pan-sides and pan-cone to allow free circulation of the pulp. The wings are shaped like inverted ploughshares, and the angle of draft for the shoes and dies has been determined with more or less exactness for the usual speed, but variations within moderate limits are allowed. The usual pan-diameter is 5 feet, and the usual pan-charge is about $1\frac{1}{2}$ tons of ore. The gearing underneath is plain and open. The shaft is often made with a downward taper for ease in babbiting, and the bearing is at least 18 inches long. The pulp is generally quite thick, and the muller-speed rapid—90 revolutions per minute for a 5-foot pan. The settlers have preferably inclined bottoms, are about 8 feet in diameter and make 15 revolutions per minute. A couple of agitators are used in each mill, but with fine grinding and careful running of the settlers, they are not absolutely necessary. It is advisable, however, to have them.

VII. Deficiency of Metallurgical Knowledge.

While with much ingenuity and thought the mechanical details of the Comstock mills were elaborated and brought to a high degree of perfection, the chemistry of the Washoe process received inadequate attention. The list of educated metallurgists who have been employed at the lode, is exceedingly small. Of those who have held important positions, I can now recall only three: Louis Janin, Guido Küstel and Melville Atwood. Küstel and Atwood left Washoe about the time of the practical abandonment of the "Freiberg process," Küstel's influence there being afterwards limited to whatever

may have resulted from his writings. Janin remained for years, and his work (with the assistance of his brothers) produced important effects.

In explanation of the scarcity of metallurgical talent, it may be well to recall certain customs and conditions which have prevailed.

Mining operations at the Comstock have always been conducted simply as a basis for stock speculations. The mass of the shareholders never inquired into the details of the mining-work. As to milling, it is a question if the majority ever gave a serious thought to it. They knew, in an indefinite sort of way, that it was a profitable business which was a perquisite of the management. They felt powerless to get this plum away from the mill companies, and "let the matter slide" as something too small for them to bother with. With stock liable to jump up hundreds of dollars per share, they cared little about mill-profits of only a few dollars per ton. The business of the mills, they considered, whenever they thought anything about it, was to rush through as much material as possible in order to pile up dividends and bull the stock market. The average stockholder never intended to hold his stocks long. He bought with the expectation of realizing a fortune by selling out in a month or so. All this made matters comparatively easy for the mill companies, which were close corporations and never sold a share of their stock to the general public. The mill, within certain limits, could make any terms it pleased with the mine, when the mill-owners formed the mine-directory.

The mills were popularly supposed to guarantee a yield of 65 per cent. of the precious metal, but the precise form of guarantee has never been made public. Exactly what percentage they extracted, has never been discovered. No one ever knew it, neither the superintendent of the mill, nor the manager of the mine, nor any other person. Figures have been given, but these were merely approximations at the best.* There never was time enough to sample carefully and assay properly the mine-products. Such a procedure, moreover, if accepted as a rule, would have interfered materially with stock manipulations.

After Sharon's system had been adopted, none of the mills, and only a few of the mines, had an assay-office. It grew to be a custom

* At least one mill has extracted 114 per cent. of the gold contained in the ore. See "Report of the United States Commissioner of Mining Statistics" (dated November 24, 1866), p. 80. Other similar instances were not unknown, but were not usually put into print.

that there should be a single assay-establishment for all the mines and mills controlled by each "ring." Great pains were taken to conceal, from all except the favored few, the results of such assays as were made. It is a well-known fact that, in the earlier years, worthless material was often mined and milled. This must have been done to a very large extent; for the tailings of many mills where part of the ore worked was certainly rich, were found afterwards to average only a couple of dollars per ton.

A peculiar custom which existed at Washoe may be mentioned here, although it does not bear directly on the subject under consideration: Reclamations on bullion which had been melted and assayed in a custom-assay office were paid by the assayer. It may be necessary to explain that the bulk of the bullion-product was purchased by a few parties who paid in full (to responsible persons) on delivery of the bullion. The bars were stamped with their assay-values (*i.e.*, the silver reckoned at $1.29 and the gold at $20.67 per fine ounce), and were sold at a discount therefrom (calculated in per cent.), which was made to cover charges, depreciation in price of silver, etc. If, for example, a bar was stamped with a value of $20,000, and sold on this basis, and the purchaser subsequently decided the true assay-value to be only $19,500, he made a "reclamation" of (about) $500, which was paid, not by the seller, but by the assayer. The custom-assayer, therefore, was careful not to *over*-stamp a bar. This system had its value for the buyer, but by what process of reasoning the bullion-producer could find in it any advantage to himself, it is difficult to understand.

The system of mill-returns has been much criticized of late years. To show how it originated, a brief account of the usual mill-process is necessary.

The ore is raised from the mine in cars, the weight of whose contents is known with more or less exactness, and transported to the mill. Here, after the larger pieces have been reduced to a suitable size by rock-breakers, it is crushed with water under stamps. The stamped material is run into a series of tanks inside the mill, and settled, the number of tanks varying with the size of the mill. The finest and most clayey particles of ore, called "slimes," are carried by the current through the tanks, where only a part is caught, while a considerable amount, in proportions dependent on the character of the ore, the fineness of crushing, and the settling capacity of the mill, escapes in suspension in the water to the outside of the building. The coarser and heavier material, which collects in the tanks, and

is usually called "sand," is removed to the pan-floor, where, in the best-conducted establishments, it is sampled for assay. It is then amalgamated in the pans, whence it is discharged into settlers, where the pulp is thinned with additional water, and the amalgam and quicksilver are separated out and caught. The earthy residues, now known as "tailings,"* after passing through agitators for the collection of any amalgam and quicksilver which may have escaped from the settlers, are washed through sluices to the outside of the mill.

The mills work the ore at a given charge per ton crushed, and return the bullion which is extracted.

In former years, the mill-man was unable to work the "slimes" and "tailings" at a profit, and they were allowed to go to waste. Hence, as a matter of fact, the mine received only the bullion obtained from the "sand." When, at a later period, it was found profitable to re-work the tailings, and still later to work the slimes, —a matter of gradual development,—no change was made in the system of returns. The mine was still credited only with the results of treating the sand; the slimes and tailings were considered the perquisite of the mill. In other words, that which was caught inside the building belonged to the mine, whatever was caught outside belonged to the mill.

Under such conditions as those above mentioned,—mining conducted simply as a gamble; the great mass of the stockholders (anxious to realize sudden fortunes) demanding that every effort should be made to increase production without regard to cost, and caring little how much metal was wasted by rapid treatment, if only dividends could be increased temporarily; with no exact knowledge of the amounts of gold and and silver in the ores; and with mills receiving a fixed rate per ton whether the material was rich or poor, and sure of supplies whatever the yield might be,—under such conditions, it naturally came to pass that the main object of the mill-man was to work the ore as rapidly and cheaply as possible, and that close amalgamation was secondary in importance. Hence it was, that, while all the machinery was improved and brought to a high state of efficiency, the chemistry of the process received comparatively little attention.

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* More exactly, "pan-tailings." The shorter term, "tailings," is often used as including slimes, pan-tailings, and concentrates, all the ore-residues, in brief, passing outside of the mill, as in the word "tailings-mill." It is used, almost of necessity, with both significations in this article; but the exact meaning in each case can be understood from the context.

The Gold Hill surface-ores first worked by Paul contained more gold than silver. Much of the gold was free, and much of the silver was apparently in the state of a simple sulphide, which is reducible by quicksilver alone, the action being hastened by contact with iron under the favorable conditions existing in the pans. It is very possible that Paul might have obtained as good results without his chemicals as he did with them. However this may be, certain mill-men found that they could get a high yield without any chemicals (as happened to Küstel when treating the Ophir surface-ores), and others were unable to extract a satisfactory percentage either with or without them. There has always been a popular impression that all the ores from the lode have substantially the same chemical composition. Concerning the constitution of these ores we know surprisingly little. A few analyses have been published, but these have been of specimens rather than samples. We do know, however, from milling-experience, that important differences exist between the materials extracted from different spots on this huge vein, which has been opened for a length of four miles, and to an extreme depth of about two-thirds of a mile.

The early mill-men, deficient in metallurgical knowledge, and ignorant of what sort of material they were treating, worked on blindly, as best they could. When they fell into difficulties, a host of process-vendors sprang up to confuse them. All sorts of substances were tried, nearly all of them of no possible virtue, and all tested by most unsatisfactory methods. As the quacks pretended to be scientific men, science soon fell into disrepute among the workers, who were unable to distinguish between the genuine and the false article; and the "practical" men, after the failure of numberless "processes," settled down to the results of their individual experiences. Bluestone and salt, either separately or jointly, always had their adherents, although, for many years, they were used in such homœopathic doses (in order not to "base."* the bullion, against which there was a strong prejudice) that no appreciable effect could be produced. But the amalgamator who happened to make a good run when using one of these reagents, was apt to remain thereafter a sturdy believer in the merits of the particular chemical employed. As each mill-man backed his belief with statements of percentages extracted—or supposed to be extracted—there was for a long time a confused complication of ideas concerning the values of these chemicals, which

* That is, to reduce the fineness of the bullion.

continued until the experiments with tailings, made in 1867 and 1868 by the Janin brothers, solved the problem beyond dispute. Even then, the stamp-mills did not adopt the intelligent use of bluestone and salt until about 1875, some five years after the successful introduction of the method at the Meadow Valley mill, near Pioche. Before this date, the Washoe mills, after employing various chemicals in most irrational ways, had gradually come to use merely what was deemed essential for keeping the quicksilver clean.

VIII. Early Attempts to Work Pan-Tailings.

The slimes often assayed as high as the original ore; but for years they were run to waste, as no one knew how to work them with profit. The pan-tailings, holding considerable amalgam, were treated more or less successfully from about the year 1864. The usual treatment consisted of concentration on blanket-sluices, followed by amalgamation in pans. The blanket-sluices were shallow troughs, with sides an inch or two high, covered with coarse blankets, and set at a small inclination—6 to 12 inches in 12 feet. Two or more troughs were placed side by side; or, in the improved form, one wide blanket-table was divided into several narrow longitudinal compartments by parallel strips of wood. The stream of tailings from the mill, with an additional supply of water, was allowed to flow over the blankets, which caught and retained the heavier particles of ore and the amalgam and quicksilver, while the lighter and poorer material was washed away. The operation was aided by a man with a broom, who lightly swept the sluices, keeping the material evenly distributed and exposed to the action of the current. Two or three times a day the blankets were removed and washed out in tanks. A later and less laborious method was to leave the blankets in place and, with the aid of large amounts of clear water, to sweep the concentrates into the tanks through connecting troughs placed at the lower ends of the sluices and covered over except when in use.

There was no particular process of amalgamating the concentrates, but the procedure was similar to that employed for the ore. The richness of the concentrates consisted largely of amalgam, which fact made their successful treatment an easy matter.

The land in the main cañons leading from Virginia City to the Carson river was divided up into small properties which originally were taken up as separate mill-sites. Many of the mills were located in these narrow cañons, close to the water-courses. Owing to the smallness of the sites, these mills, as a rule, could have only short

lines of sluices; and when the treatment of blanket-tailings was found profitable, the owners of land commenced to pick up the material discharged from the mill-sluices above them for re-concentration in the same manner. This was done by successive land-owners, the residues being enriched by fresh supplies from the quartz-mills interspersed at intervals, until finally there were almost continuous series of blanket-sluices lining Seven-mile, Six-mile and Gold cañons. Many of the small owners put up little amalgamating-works, consisting of a pan or two, or an arrastra, or, in a few instances, of a barrel, using small water-wheels as motors. Others merely collected concentrates and sold them.

The yield of blanket-tailings, in 1867, was from $18 to $25 per ton and aggregated over $150,000. The cost of treatment was light, especially for the small operators who did all the work themselves and had their material delivered to them for nothing. These increased and prospered until the gradual decrease in value of the tailings and conflicts with the Water Company in time drove many out of the business.

The troubles with the Water Company arose from the insufficiency of the natural supply of water. To remedy this, water-companies (afterwards consolidated into one—the Virginia and Gold Hill Water Company) collected all the available supplies in the district and brought in additional quantities from outside sources, distributing and selling the water at high rates. The Water Company claimed that it sold to mills (and sluices) only the *use* of the water. Acting on this theory, the company caught up the water when discharged from the sluices of purchasers, and carried it in flumes around the property of those who refused to pay for it. The small sluice-owners claimed that it was impossible to distinguish between the natural and the artificial supply, and that, moreover, all waters once in a creek were free for the use of the riparian owners. Many refused to pay, and tore down the company's flumes. The dispute lasted for years, and was attended with violent acts and even bloodshed. The victory finally remained substantially with the Water Company, which built a small dam near Silver City and led the water and tailings coming down the creek (or the larger part of them) through a long line of flumes to the Woodworth mill, which was located on the Carson river, at the mouth of Daney cañon. Here the tailings were passed over a blanket-table, 1700 feet long and 19 feet wide, divided into compartments each 19 inches wide by 130 feet long. This was, undoubtedly, the largest structure of the kind ever built.

After the use of the blanket-sluices had been well established, reservoirs for impounding the slimes and tailings were built here and there. Most of those in the cañons were necessarily small, but those on the flats below the ravines were large. The impounding of these materials did not occur on an extensive scale until the process for working them had been brought to a point where success seemed certain. How this process was developed, will now be told.

IX. Discovery of the Tailings-Process.

In 1862–63 Louis Janin was metallurgist of the Mexican mill at Empire City, where the ores were worked by wet-stamping and amalgamation in the batteries and in Mitchell's screw-amalgamators. The tailings were dried, roasted in reverberatory furnaces, and amalgamated in barrels. Raw amalgamation in barrels was also tried—with partial success. In 1864, Janin was called to the old Gould and Curry mill. This establishment had made a failure with the Veatch process, which was thrown out that year, wet crushing and pan-amalgamation being substituted. The mill had a large amount of tailings on hand, including slimes, which assayed as high as $130 per ton, and blanket-concentrations. The richest slimes were hauled to Franktown and amalgamated in Dall's mill by the barrel-process. Janin put up experimental pans and worked the remaining tailings in various ways, generally trying to imitate the patio-process. Experiments were also made in patio-yards, but these were a complete failure, while those carried out in pans were attended with a fair measure of success. They all indicated strongly the utility of bluestone and salt, but the full appreciation of these re-agents was delayed by the difficulty of disposing, without very heavy discounts, of the base bullion produced. Bankers were afraid of it, and everybody was trying to get fine bullion.

In 1866 the Gould and Curry Company built the Reservoir mill, a short distance below its large quartz-mill, for the purpose of working its tailings. This was the first mill built expressly for the treatment of tailings, with the possible exception of one or two crude, miniature establishments of sluice-owners in Six-Mile cañon. It had 14 flat-bottomed pans, and was operated by the Gould and Curry until that company went out of the milling business and, in 1870, sold it to Parke and Bowie, who enlarged it and named it the Railroad mill.

Janin seems to have been confident of success as early as 1866, when he, with Charles Bonner and Ira S. Parke, built a large

reservoir in Six-Mile cañon, a short distance below the Reservoir mill, to collect the tailings and slimes from both Six-Mile and Seven-Mile cañons. Within a year about 100,000 tons were collected, assaying from $8 to $24 per ton; and in September, 1867, Janin commenced to work the material in the Cañon tailings-mill, which he had built near by for this purpose. The first runs were most encouraging, but a freshet in the following December swept away about half of the contents of the reservoir; and as the flood carried off the greater part of the rich slimes, from the treatment of which large profits were expected, the result was disheartening, although by no means disastrous.

The possession of a mill of his own enabled Janin to push his researches (which, with the aid of subordinates, he had kept up constantly) with more independence than before. He tried various chemicals, but always obtained the best results with sulphate of copper and salt, although these reagents produced a poorer amalgam and, with the apparatus then used, the quicksilver was fouled and its consumption was large. Up to this time the usual method was to grind the pulp in small flat-bottomed pans, then thin it, and discharge it without settlers. Increasing the size of the pans and using settlers remedied in great measure the flouring and loss of quicksilver. Especial trouble was found in treating slimes, which, when amalgamated fresh from the reservoirs, would remain in lumps in the pans, robbing the quicksilver and refusing to yield up the precious metals. But this difficulty was overcome by drying the slimes, the lumps then readily dissolving in the pans. The ever-varying quantity of gold and silver in the batches worked, and the presence of amalgam caught with the tailings, augmented the troubles encountered in the experiments, by introducing an element of uncertainty as to the actual results obtained. Furthermore, the ores from the different mines did not work alike, the percentages of bullion-yield varying greatly under precisely the same conditions of treatment. Particular difficulty was found with the Chollar-Potosi slimes which, notwithstanding an excess of bluestone, produced only fine bullion. This result was attributed by some persons to decomposition of the bluestone by the oxide of manganese which was present in the ore. Others have thought that probably the injurious agent was an alkali or alkaline earth.

Henry Janin, having become financially interested with Louis, visited Washoe in 1868; and the two brothers (aided later by Alexis Janin) made a systematic, extensive and costly series of ex-

periments, which thoroughly demonstrated the efficacy of bluestone and salt. Henry Janin came to the conclusion that dichloride of copper was the active reagent in the reduction of sulphurets of silver, and thought its direct use in the pans would be greatly superior to the use of salt and bluestone. He obtained a patent for this process; but the brothers never did more than experiment with the method, confining themselves to the question of the proper proportions of copper sulphate and salt. This question they solved to their satisfaction; and when they accepted the fact that base bullion was a necessity, and ceased trying to make fine bullion, their success was complete. They proceeded to secure slimes and tailings from all sources and to engage extensively in the business. They had apparently a fortune in their grasp. But it was impossible to keep their process secret; and when the ore-supply from the mines became scarce and concentrated in the hands of a few millmen (as before described) all the mills began to save their tailings and slimes and to work them up for themselves. Moreover, other tailing-mills were built, and the competition became keen.

X. Development of the Tailings-Business.

At the end of 1865 or the beginning of 1866, J. R. Andrews & Co. built large reservoirs on the flats at Dayton, and caught the tailings coming down Gold cañon. The amount collected here in 1869 was estimated at 400,000 tons, assaying from $15 to $30. Quite a large quantity was much richer; one reservoir, which was afterwards purchased by Birdsall & Co., assaying about $60. The mill here, a small establishment, originally built in 1864 to work ore, and known later as the Dayton Reservoir mill, was purchased and started in the fall of 1866 by E. Wertheman and Dr. Briarly, who sold out to Louis Janin and Leon Baldwin. This firm altered it, and put in five McCone pans, each holding 4000 to 5000 pounds of tailings to the charge, and treating four or five charges daily. Baldwin sold his interest to Janin, who, in 1871, tore down the old building, and erected a new structure on the same site. The mill worked a part of the tailings collected in the Dayton reservoirs; but the main portion of these passed into the possession of Birdsall & Co., and the Dayton Reservoir mill finally was obliged to close down.

About 1867, Janin and Parke started a large reservoir on the banks of the Carson river, 3 miles from the mouth of Six-Mile cañon, the flow of water and tailings in this cañon being caught up and led in a ditch to the reservoir. At one time (about 1869) the contents of

this reservoir were estimated at 200,000 tons. The quality was much poorer than that of the material impounded at sites nearer the stamp-mills. The average assay-value was only $7 or $8, according to samples taken in 1875; but much poor material and even waste (brought down in seasons of abundant water) had been allowed to run in. No attempt to work the tailings here was made on a large scale until 1870, when the Carson Valley Tailings Company put up a mill with ten large Parke pans, holding 8 tons to the charge. It was generally understood that the profits of the company were meager.

In 1869, Augustine, of Silver City, put up a mill with four large pans on American Flat, to work the tailings from a reservoir below the old Bay State mill. This tailings-mill ran irregularly, and with results which were apparently unsatisfactory.

D. E. Avery had an establishment in Washoe valley called the Back Action (if I remember aright), which worked tailings on a small scale. It was built in 1866, and then had eight arrastras. Concerning this mill I have only scant information. I imagine that it was not a great pecuniary success.

In Six-Mile cañon there were a number of mills, some of them curious little home-made affairs, which worked blanket-concentrates principally. Bassett's mill, originally built with 16 stamps, was using a barrel-process for tailings in 1865, but was credited with 6 pans in the statistics of 1866. Jennings, Proctor, Evans, and one or two others, had establishments which ran more or less intermittently. Jennings, I think, outlasted the others.

The Atlanta mill, at Johntown in Gold cañon, erected in 1865 with 10 stamps, was owned by W. Hill & Co., who altered it to a tailings mill. In 1871 two large Parke pans, 8 feet in diameter, were put in. The owners worked tailings for a number of years from a reservoir near by.

Not far from the last mill was the Keystone, built in 1865 with 5 stamps, and changed subsequently to work tailings. It was owned by Rulison and Lykens, who afterwards moved it nearer Dayton. It was operated whenever pay-material could be obtained.

Concerning the mill of Birdsall & Co. (afterwards the Lyon Mining and Milling Co.), which was built originally for crushing ore, detailed information will be given in a subsequent part of this article.

Besides the tailings-mills above mentioned, quite a number of stamp-mills engaged in the tailings-business when supplies of ore

from the mines were unobtainable,—as the Woodworth mill, on the Carson river at the foot of Daney cañon, where large amounts were treated.

The result of the increase in the number of mills working tailings was that in time rich tailings were not to be bought; even low-grade material was very difficult to get at paying prices, and the Janins were gradually forced out of the business. When they had retired there were only two large tailings-companies in successful operation at the Comstock: Parke & Bowie in Six-Mile cañon, and the Lyon M. and M. Co. at Dayton.

Ira S. Parke and David Bowie owned two mills—the Railroad mill, formerly called the Reservoir mill, which they had bought from the Gould and Curry Company in 1870; and the Cañon mill, which they purchased from Janin about 1872, enlarging it and changing the name to Express mill. Parke was a good millwright and an able and energetic worker, who had learned the tailings-business from the Janins. The firm secured large reserves of tailings and operated for several years, apparently with great success. They did not, however, appreciate the importance of close assaying and of chemical studies, but sought only for improvements in mechanical details. They had no assay-office connected with their mills—in this respect, however, merely following the usual Comstock custom—and were content to work without any exact knowledge of the values of the material which they treated, values which were varying constantly. Hence they had only hazy ideas of the accuracy of their work, and were liable to make (and did make) costly mistakes.

Parke was constantly changing the size and form of his pans and settlers, and seemed to have a mania for large pans. He undoubtedly introduced several improvements, but eventually over-did the thing. He increased the size of the pans to 6, 8 and 10 tons per charge, and in this his example was followed very generally. The extreme dimensions reached were 12 feet diameter by 8 feet height. Increasing the size was found beneficial to a certain extent, but the largest pans were complete failures. He then began to increase the number of mullers in each pan. He invented a 2-muller pan which was 11 feet long, 5½ feet wide and 5 feet deep, held 5-ton charges and was provided with 2 settlers. At first the mullers ran in opposite directions, later in the same direction. He had a 6-muller pan, also in the shape of a trough, but with very small mullers. He built 3-muller and 4-muller pans. No one, so far as I know, copied these multiple-muller pans, nor was it easy to find out the exact results obtained with them, owing to the loose manner in which all

the experiments were made. It was plain, however, that large motive power was demanded, and that the construction was expensive.

In 1874 quicksilver had risen in price from 60 cents to (at least) $1.30 per pound, and the price of silver had fallen greatly. This added essentially to the troubles of the tailings-mills, which were then struggling with low-grade material, and proved fatal to Parke and Bowie, who accordingly failed. In fact all the tailings-mills in Washoe, except the Lyon mill, closed down this year, some temporarily, others permanently; and none (with the exception noted) made any long runs thereafter. The Express and Railroad mills were sold. Ira Parke ran them subsequently for account of his creditors, but he died before long. His place was taken by his brother, Frank Parke, who a few years after died in the same manner as Ira—being run over by the steam-cars. The success of the mills during these last years was very limited.

The best arranged, best equipped and most successfully conducted tailings-mill ever operated was the Lyon mill at Dayton. This was built in 1865 by Birdsall and Carpenter to crush ore. Carpenter soon sold his interest to Birdsall, who was invited by Sharon to join the Union Mill Company, but declined. The mill then found it impossible to obtain adequate ore-supplies, although its charges for hauling and working were as low as $8 per ton; and for several years it ran intermittently and without much profit.

When the treatment of tailings began to attract attention, George Langtry, a most energetic and able millwright, induced Birdsall to undertake this branch of the business. Birdsall owned land adjoining the Andrews property at the mouth of Gold cañon, and here Langtry put up concentrating apparatus, receiving in return a small interest in the mill, of which he was soon after made superintendent. When the Janins perfected their process, Langtry found out the details and copied them with success. In 1869 the company purchased the rich Andrews reservoir, added 20 pans to the mill and worked on a very large scale, at one time amalgamating 250 tons daily. But in order to keep up this rate of treatment, it was found necessary to work the slimes "green" from the reservoir, and the company gave up this wasteful procedure and reduced the daily amount treated to about 150 tons.

George Langtry was, all things considered, the best tailings-millman of Washoe. He differed from the ordinary "practical" man in many respects, perhaps in none more remarkably than by recognizing and acknowledging the fact that his want of chemical knowledge was a serious deficiency. To remedy this deficiency, he sought

always to have a capable metallurgist connected with the establishment. Under his management, and the subsequent guidance of his pupil, the Lyon mill achieved a success unparalleled in the history of tailings-mills. From 1869 to 1878, notwithstanding the decline in the value of the material worked and of the silver produced, and in spite of the great increase in the price of quicksilver, the mill steadily yielded large dividends. The company had a working-capital sufficient for the purchase of large amounts of rich slimes and tailings whenever such came into the market. The mechanical skill and great executive ability of Langtry, supplemented by the chemical knowledge of his assistants, were sufficient to overcome each new difficulty in amalgamation as it arose, and enabled the mill to run on low-grade material which no other establishment could treat with profit. Thus, when the price of sulphate of copper was raised to an excessive height (it touched 24 or 26 cents per pound), the company's officers were found competent to put up and run acid-chambers and bluestone-works with such success that the mill obtained its own supplies at low figures and large revenues were derived from sales to others. When the discount on base bullion reached 27 per cent. in 1874, a process for refining this bullion was promptly devised, whereby not only were fine silver and fine dorée bars turned out, reducing the discounts, but also, through the recovery of the copper previously lost, in the place of additional expenses, additional profits ensued. This process, it may be noted, solving the difficulty which so long delayed the financial success of the Janin brothers, rendered their method practically complete. Moreover, as no one else ever succeeded in refining bullion so rich in copper, other mills sent their product to the Dayton refinery, to the financial advantage of the Lyon mill.

Langtry died in harness in 1875, but so long as his methods prevailed the mill was operated with financial success, and although the tailings treated sank in assay-value to about $5 per ton, it was found possible to earn and pay continuous dividends. But when his pupil resigned in 1878, and a new management imbued with radically different ideas was established, dividends became painfully rare.

In 1877 the bonanza firm built their large tailings-mill on the site of the old Gould and Curry mill. It was the last successful mill of the kind erected at the Comstock, and was appropriately named the Omega. It had a daily capacity of about 170 tons and ran a number of years on slimes and tailings from the Consolidated Virginia and California ores. It was provided with a couple of

long cylinder-furnaces for drying the slimes, and with a refinery, built in 1878 after the plans of the one at Dayton. It was operated successfully, I am told, until it exhausted its supplies.

Early in the '80's a small mill was built by Conrad Wiegand and others, east of Virginia City, near the railroad tunnel. It was erected after my departure from Washoe, and I have no detailed information concerning it. The intention was to work the tailings of the old Lynch reservoir, which were supposed to be very rich. These tailings had been offered previously to the Lyon mill, which sampled and assayed them, finding the material to average only about $6, if I remember correctly. The Lyon mill gave its assay-results to the owners, who refused to believe the figures. Conrad Wiegand, the well-known assayer, made some assays, obtaining much higher results, and put about all the capital he had into the company which built the mill. I have been told by a person who was connected with the establishment that the mill-runs verified the Lyon mill assays, and that the mill was a complete failure from the start. This failure is supposed to have been one of the principal causes of poor Wiegand's tragic death.

XI. Treatment of Tailings at the Lyon Mill.

The plant of the Lyon mill was slowly wrought out by experience. The original mill had 30 stamps, 20 Wheeler pans, 10 settlers, and 3 "lavaderos" or washing-tubs. The power was furnished by a 50-foot overshot water-wheel. The pans first used for tailings held 1500 to 1800 pounds to the charge. When the Andrews reservoir was bought, in 1869, 20 Wheeler pans and 10 settlers were added, each pan holding charges of 2500 to 2800 pounds. In 1870 and 1871 very large pans for tailings came into vogue. These were "combination" pans, with iron bottoms and wooden sides. Janin was using McCone pans, which held 2 to 2½ tons of tailings (or 1¼ to 1½ tons of pure slimes), and Parke was introducing much larger sizes. Influenced by the reports of improved results obtained with the largest pans, the management, in 1872, took out 20 Wheeler pans, substituting 2 10-ton pans of the Parke pattern, with 2 large settlers. These pans were 9 or 10 feet in diameter by about 6 feet high, and the settlers were about 12 feet in diameter by 8 feet high. Their use was attended by such a large loss of quicksilver and low yield, that they were soon thrown out, and the mill was run with the remaining 20 Wheeler pans. Finally these last were replaced by 8 combination pans, of about 3 tons capacity, for tailings, and 2 somewhat smaller pans for slimes, each pan being provided with a settler. The

experience of the Lyon mill indicates strongly that, for the material worked there, the best results are to be obtained within the limits of 2-ton and 3-ton charges.

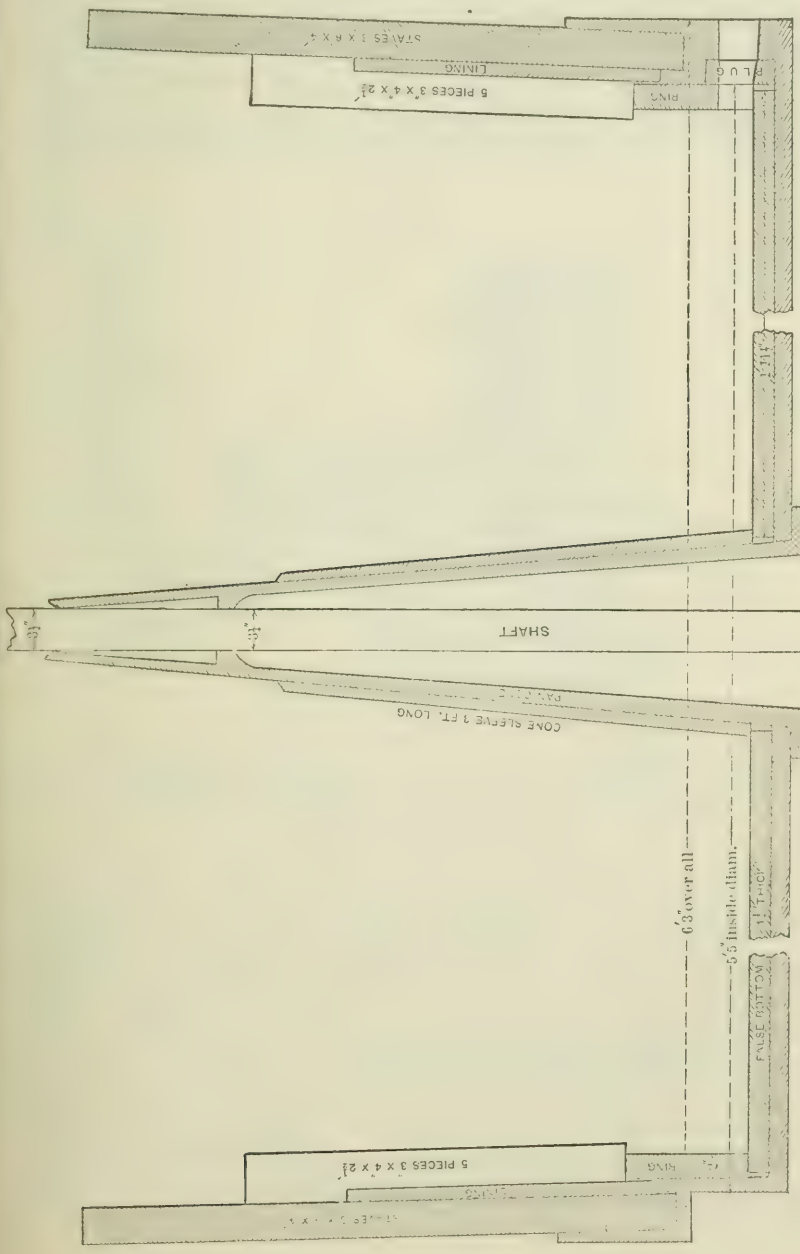


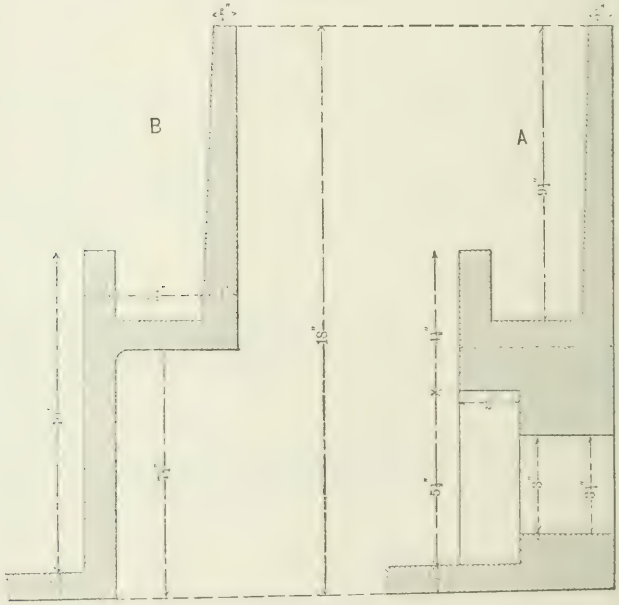
FIG. 1.

Tailings-Pan, Lyon Mill, Dayton, Nevada.

The perfected arrangement of the mill was as follows:

The mill-building was located on a steep side-hill with abundance of fall. The tailings were hauled from the dumps and reservoirs in cars which entered on the upper floor directly over the bins. These cars had sheet-iron bodies supported on 4-wheeled trucks, modeled after the ordinary railroad pattern. They held 3300 pounds of pan-tailings, dry weight, and dumped through the bottom of the body and between the rails. The track was of railroad-iron, much of it old 65-pound iron rails, bought from the Virginia and Truckee Rail-

FIG. 2.



Flange of Pan-Bottom.

A. Section at discharge-hole. B. Section at any other point of circumference.

road Company when this road substituted steel rails. Care was taken to have a solid track and well-constructed rolling-gear, and hence the repairs were only nominal and the expense of hauling was light.

The bins held enough to supply the mill for several days. Their spouts discharged directly into the pans.

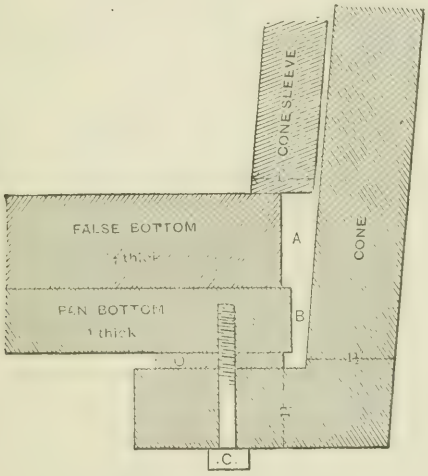
There were eight pans for treating the tailings caught in the company's reservoirs and locally called "sand," by which term they will be designated hereafter in this paper, and two pans for the slimes, which were all purchased from other mills. The pans

were supported on an open framework, admitting light and giving free access to the shafting and gearing below.

There was a settler to each pan. The sand-settlers were 9 feet in diameter by 7 feet high. The slime-settlers were eight feet in diameter. The settlers discharged over blanket-sluices which extended from the mill to the Carson river.

Besides the apparatus named, the mill contained a small boiler for heating the pans with live steam, a Chili mill for grinding copper-ore and another for crushing the roasted bullion from the refinery,

FIG. 3.



Mode of Fastening Cone to Bottom.

A. Space between false bottom and cone, calked with pine. B. Space between bottom and cone, calked with thick red-lead and hard-wood wedges. C. Eight $\frac{3}{8}$-inch bolts. O. Rubber gasket, with thick red-lead and oil.

an extra pan or two for treating sweeps and sulphurets, and a special dissolving tub for making bluestone-solution. All the machinery was run by a Leffel turbine, placed with its axis horizontal, under a 52-foot head.

The construction of the pans is shown in the accompanying sketches, the dimensions given being those of the larger pans. Concerning the details of construction, the following points may be noted:

The height of the flanges of the pan-bottom was gradually increased until it reached the figures given. This prevented all leak-

age of quicksilver between the staves and the iron—a leakage which occurred, to a greater or less extent, with low flanges, despite all methods of calking which could be devised.

Iron rings were used to protect the flanges. The greatest wear on the pan was at the discharge-hole, where, in the older construction, the iron was rapidly destroyed until the hole became enlarged to such an extent that it could no longer be plugged securely; and thus the whole bottom was, of necessity, thrown away, although sound in all other respects. By casting the bottom with an outside boss at this place, having a square socket on the inside, $7\frac{1}{2}$ inches wide by $5\frac{1}{2}$ inches high by 2 inches deep, this socket being filled with an oaken plug bored for the discharge-hole, and by using false bottoms, the life of the pan was prolonged for years. An oak plug lasted as long as three false bottoms and rings. A false bottom $1\frac{1}{2}$ inches thick wore out a 2-inch ring and a muller.

The wooden staves forming the pan-sides were protected by a lining of inch-stuff, 2 feet long, nailed on the inside. Instead of wings, five pieces of timber, 3 inches by 4 inches by $2\frac{1}{2}$ feet long, spiked vertically to the sides, were used with satisfactory results.

The muller and shoes were cast in one piece, as shown in Fig. 4 (p. 223). This simple form was adopted after trial of many varieties. By leaving ample spaces, both on the outside and the inside of the muller, with a proper speed, there was no difficulty in securing a good circulation of the pulp.

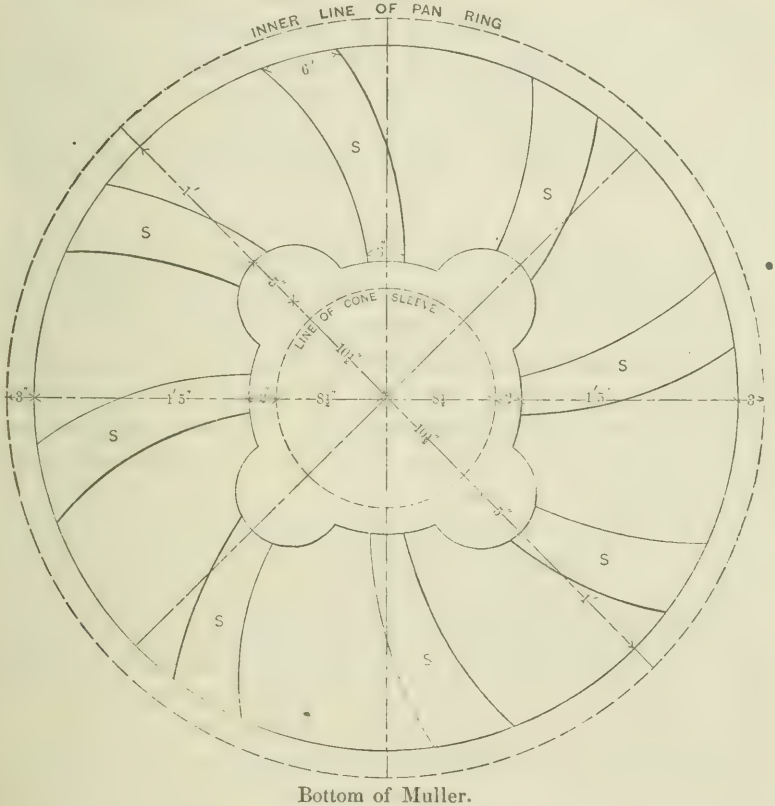
Besides the usual appendages of retort-house, blacksmith-shop, carpenter-shop, etc., the mill was provided with a bullion-refinery (which has been described in a previous volume of these *Transactions*),* with sulphuric-acid chambers, and with bluestone-works. Moreover, at one time, there was a borax-factory attached, which was a short-lived affair. This last was in operation in 1872, when the borax-excitement raged in California and Nevada. The managers of the company were attacked by the fever, and by their order a couple of dissolving-tubs with crystallizing-vats were put up in the mill. Borate of lime was hauled some 60 miles, in wagons, from Rhodes's marsh, near Columbus, Nevada, and treated with carbonate of soda (also hauled a long distance) and steam. The borax produced was of the finest quality. A considerable amount was shipped to England, and there sold at the highest market-prices. But the price of borax fell from 36 to 8 cents per pound (local quotations);

* *Trans.*, xiv., 731.

and as the latter sum was insufficient to pay expenses of manufacture at such a point as Dayton, which was far removed from the deposits, the borax-department was closed.

The acid-factory held two lead-chambers, one 15 feet high by 20 feet wide by 45 feet long, the other of the same height and width, but 90 feet long; besides concentrating-pans, where acid of 62°

FIG. 4.



Bottom of Muller.

S, S. Shoes cast in one piece with muller.

B. was made for sale. The two chambers were run, either separately or (by preference, when business demands permitted) as a system. They produced chamber-acid of 45° to 48° B., and tower-acid of 12° to 18° B., the latter article coming from the "bumbaroon-towers," which were cheap local substitutes for the more expensive Gay-Lussac apparatus. The raw materials used were Japanese or Nevada sulphur and niter. Owing principally to the uncertain

and fluctuating nature of the market, which, for a couple of months, would demand an immediate supply that taxed the extreme capacity of the works, and then suddenly fell off to almost nothing, it was impossible to run the chambers continuously, and therefore to the best advantage. Nevertheless, good results were often made, in one instance, in 1877, the chambers producing 290⅔ pounds of monohydrated acid per 100 pounds of sulphur consumed. As the maximum *theoretical* production is 306¼ pounds of monohydrate for this amount of sulphur, the loss was only 5.1 per cent.

A large part of the chamber-acid was concentrated in lead pans to 62° B. and sold. The rest, except small quantities sold to mills, was used without concentration in the bluestone-works. The weak tower-acid was utilized in making copper-sulphate solution for the mill.

The raw material for the bluestone-factory consisted of copper-ore, principally carbonates from Walker river, copper-matte and precipitate from California, and base bullion. The bluestone made from the bullion was of the highest grade, and equal to the best English imported. The product of the factory was sold to amalgamating mills, principally; some went to California for the use of the farmers. During several years there was a branch establishment at Carson, which utilized the copper-sulphate solution from the mint refinery.

The Lyon mill used only the refuse from the factory, that is, the scrapings and portions which did not look well enough to be put on the market, or the mother-liquor which was too weak for profitable concentration. Usually it employed uncrystallized sulphate of copper made from ore in a special tub. In either case, the copper-sulphate held an appreciable amount of iron-sulphate, and, at times, an excess of acid designedly introduced.

Sulphate of iron was manufactured in small amounts to satisfy a local demand.

The tailings worked at the mill from 1873 to 1877 inclusive were for the most part of very low grade. They were classified under the local designations of "sulphurets," "slimes" and "sand."

The sulphurets were obtained by purchase from the blanket-sluices so extensively used at the Comstock. They were principally iron-sulphurets. Those caught near the mills which crushed the ores whence they were derived, oxidized with comparative facility on exposure to the air, and yielded by amalgamation as much as 80 per cent. of their contents of precious metal. Those caught at greater

distances from the quartz-mills showed no signs of oxidation even after long exposure, and it appeared impossible to extract from them by pan-amalgamation more than 50 per cent. of their assay-values. The reason for this difference in behavior was not known positively. Sulphurets were obtained only in small amounts, were usually amalgamated by themselves, and made no figure in the production of the mill.

The slimes, during the years mentioned, were all purchased from the stamp-mills whenever these had any to sell. The silver contained in them existed apparently in part as a simple sulphuret (such as argentite), and in much larger measure as a multiple sulphuret (such as stephanite or polybasite). Their chemical composition varied materially. Their chief constituent, clay, was the main cause of the difficulty in treating them.

The slimes were spread out on the ground, all lumps being broken up, and allowed to dry thoroughly before being worked. Long experience proved beyond question that this preparatory drying was essential for good amalgamation. In common parlance, the slimes were "oxidized," but the change effected was mechanical rather than chemical.

At Dayton, two pans were reserved for treating this material, which was mixed in the pan with certain amounts of sand. The usual charge was composed of 60 to 65 per cent. of slime and 40 to 35 per cent. of sand. The sand seemed to act beneficially by cutting up any lumps of slime which might exist, and by clearing the globules of quicksilver from any adhering slimy envelope which, unless removed, would keep the quicksilver from contact with the precious metals, prevent the globules from uniting, and carry them off in the stream of tailings.

The slimes purchased by the mill varied greatly in value, ranging from $13 to $30 per ton, with occasional higher values. The slime-charges (slime and sand mixed) during the years 1875 to 1877 assayed from $13 to $18 per ton.

The sand was that part of the tailings which had escaped from the stamp-mills, and after passing through numberless blanket-sluices and other concentrators, finally had been caught in reservoirs. Some of this material was purchased. By far the larger part came from the company's large reservoirs on the flat at the mouth of Gold cañon, 3 miles or more from the nearest quartz-mill. Into these reservoirs were led the waters of the creek, carrying not only the

residues from the blanket-sluices above, but also various impurities discharged into the cañon or washed down by the rains. The inflowing stream was watched night and day by men who decided from its appearance whether it held a sufficient amount of pay-material to be allowed to settle in the reservoirs, or whether the contents were too poor for profitable treatment, and therefore were to be turned back into the creek-bed. When one reservoir was filled with the water, the stream was led into another, and the material in the first was allowed to settle, the cleared water being then drawn off. In this manner the reservoirs were gradually filled with sand, which was allowed to remain undisturbed for a long time, and then was ploughed up and more completely dried under exposure to the air. During the dry, hot summers at Dayton, evaporation proceeded at a rapid rate.

The material deposited at the upper end of each reservoir, near the inlet, was left untouched, being too coarse and poor. The remainder, which held a little slime and assayed from $5 to $7 per ton, averaging only a shade over $5 during the years 1875 to 1877, was scraped up and hauled into the mill.

The sand formed 90 to 95 per cent. of the whole amount amalgamated in the period mentioned above. It consisted principally of fine quartz sand, whence its local name, and held limited but varying amounts of clay. Iron and copper pyrites were distinguishable, but beyond this the mineral composition was doubtful. The quicksilver used for amalgamation by the mills crushing the original ore had been thoroughly removed by the repeated concentrations to which the material had been subjected before reaching the reservoirs.

Although the Janins had discovered and made known the proper method of amalgamating tailings, it was necessary for each mill to work out certain details best suited to its own special conditions. At the Lyon mill tests were being made continually to determine what modifications were required by the varying character of the tailings, and no trouble was spared to make these tests accurate. No experiment was considered satisfactory unless sufficient material had been treated to give results *in bullion*. Hence, a week's run at least was made in each instance, and often a month was taken to complete a test. Moreover, as an exact knowledge of the results which were being obtained in actual practice was considered a necessary basis for all efforts at improvements, great pains were taken to determine accurately the amounts and values of the material worked.

All the slimes, being purchased, were of course weighed. The sand was measured by car-loads. To decide the weight of a load, in the beginning 1200 weighed tons were hauled, and check-tests were made subsequently from time to time.

To determine the values, a scoopful of material was taken from each car and put in a box. At the end of each day the sample thus collected, marked with the date and the number of carloads, was thoroughly mixed and its moisture determined. Sometimes, these samples were assayed each day for gold and silver, but, usually, amounts proportioned to the number of loads were weighed out daily and the assays made once a week.

Four samples were taken daily from the mill—one, of the sand worked in the large pans; another, from the large settlers; a third, of the slime-charges of the small pans; and a fourth, from the small settlers.

In getting the settler-samples, a dipperful was taken from near the surface of the charge just before each plug was pulled, and poured into large buckets. At the end of the day, after thorough settling, the supernatant water was siphoned off carefully, and the residue dried for assay. This method seems preferable to the more usual way of taking the sample from the material as it runs out of the plug-holes. At Dayton, the percentage of yield, calculated from the settler-assays, did not usually differ from that obtained by the bullion-product by more than one-hundredth of the assay-value of the material milled.

Ten one-ounce assays were made of each sample. The silver beads obtained by cupellation were parted in two sets of 5 beads each, giving a check-assay of the gold, in weighable quantities. As the assay-tables used (Taylor's) were calculated for half-ounce assays, on averaging the results, any error was divided by 20. This gave most accurate results, but entailed a great amount of work. Work, however, was not spared at the Lyon mill.

The pans all ran four hours. With the low-grade material treated, any increased product arising from longer amalgamation was more than counterbalanced by the increased expense.

While the pans were being charged, live steam was introduced until the pulp was heated to a degree just bearable to the hand. No advantage was derived from greater heat; on the contrary, a higher temperature was apt to volatilize the mercury, which occasionally was found condensed in the caps of the drivers. This was taken as proof that the pans had been run too hot.

The salt was introduced at once with the tailings, and the bluestone was added about half an hour later. After the pans had run nearly an hour, and the pulp had been brought into proper condition, the quicksilver was charged.

The amount of sulphate of copper used, while varying with the different materials treated, averaged about 10 pounds per ton for the sand-pans, and 20 pounds per ton for the slime-pans, calculated in terms of crystallized bluestone. When the loss of quicksilver commenced to increase, either more bluestone or more acid was charged, according as outside conditions made it more convenient, and always with improved results. The useful effect in such cases, I am inclined to think, was due to the excess of acid, which might act chemically or mechanically on the clay in the mass. It is possible that the acid may have combined with the clay to form alum, which, according to Malaguti and Durocher, acts beneficially in amalgamation.

From 2 to 2½ pounds of salt was used to each pound of bluestone. Careful experiments determined decisively that no better results were obtained with greater proportions of salt, and that it was not safe to use less.

The quicksilver-charge was 50 pounds to the ton for all tailings (sand or slime) usually coming to the mill. For uncommonly rich slimes the amount was increased.

Much stress was laid on having the pulp in the pan of proper consistency. Repeated tests had shown that better results were obtained with a thick than with a thin pulp. The pulp was considered thick when it would drop slowly (not run) from a stick which had been dipped into the charge. With the right consistency, the mass had a slow, rolling motion, and the quicksilver could be seen in small globules thoroughly disseminated throughout it.

In 1877, a long series of exact experiments was made to determine this question afresh. The results are given below. The column headed "tons per charge," shows the relative thickness of the pulp. Seven tests were made with sand, and six with slimes. One of each (No. 3) miscarried. More than 19,000 tons of sand, and nearly 2200 tons of slime-charges were used in the experiments.

In every test a much larger percentage of the silver was extracted with thick pulp than with thin charges. The evidence here seems to be conclusive. With reference to the gold, the results, although not decisive, tend in the same direction. With the richer material (slime-charges) a gain is evident, while with the poorer (sand) the loss per ton of this metal seems to have been reduced to a minimum with both thick and thin pulp.

Tests of Consistency of Pulp.

Number of Test.	Tons Worked.	Pulp.	Tons per Charge.	Assay per Ton.			Yield per Ton in Bullion.					Loss per Ton.		
				$ Ag.	$ Au.	$ Total.	$ Ag.	$ Au.	$ Total.	Per ct. Ag.	Per ct. Au.	Per ct. Total.	$ Ag.	$ Au.	$ Total.
SAND-PANS.															
1 and 2	5877.3	Thin.	2.78	4.40	1.10	5.50	2.71	0.25	2.96	61.6	22.7	53.8	1.69	0.85	2.54
7	1095.6	"	2.77	4.40	1.05	5.45	2.70	0.26	2.96	61.4	24.5	54.4	1.70	0 79	2.49
1, 2 and 7...	6972.9	Thin.	2.78	4.40	1.09	5.49	2.71	0.25	2.96	61.6	23.0	53.9	1.69	0.84	2 53
4	4831 2	Thick.	3.18	3.62	1.10	4.72	2.54	0.26	2.80	70.1	23.2	59.2	1.08	0.84	1.92
5	4595 2	"	3.25	3.64	1.15	4.79	2.87	0.29	3.16	80.0	25.0	66.0	0.77	0.86	1.63
6	2805.0	"	3.22	3.78	1.15	4.93	3.03	0.32	3.35	80.1	28.0	68.0	0.75	0.83	1.58
4, 5 and 6...	12,231.4	Thick.	3.22	3.67	1.13	4.80	2.78	0.28	3.06	75.8	25.0	63.8	0.89	0.85	1.74
SLIME-PANS.															
1 and 2.....	554.3	Thin.	1.08	20.15	3.51	23.66	15.00	1.39	16.39	74.4	39.8	69.3	5.15	} 2.12	7.27
4	558.4	Thick.	1.45	11.78	1.76	13.54	10 06	0.78	10.84	85.3	44.1	80.0	1.72	0.98	2.70
5	531.1	"	1.45	13.96	2.31	16.24	12.00	0.93	12.93	86.1	40.0	79.6	1.96	1.38	3.31
6	535.9	"	1.45	14.15	2.76	16.91	12.16	1.20	13.36	85.9	43.6	79.0	1.99	1.56	3.55
4, 5 and 6...	1,625.4	Thick.	1.45	13.26	2.27	15.53	11.38	0.97	12.35	85.8	42.5	79.5	1.88	1.30	3.18

An additional argument in favor of thick pulp is, that in a given time larger amounts of material can be treated, and, therefore, an increased production obtained, with the same plant, without materially increasing the working-expenses.

One possible cause of the greater yield of thick over thin charges

is the less violent agitation of the quicksilver, which is easily floured by excessive churning. Such churning does not occur with what I consider a proper consistency and movement of the pulp, but the quicksilver is subjected to a slow rubbing, which tends to keep it clean and perhaps to remedy fouling—just as the amalgamator when "horning" a charge is often able to clean and collect the amalgam by judicious rubbing with the thumb or finger.

An important detail of amalgamation is the management of the settlers. The usual plan in silver-mills is to have only one settler to a pair of pans, and only too often the settler is left to take care of itself. The Lyon mill had a settler for each pan, an arrangement conducive to better extraction of the precious metals and especially to the saving of quicksilver. When a charge is first run off into the settler, quite a large amount of the quicksilver appears to be floured, but this quantity constantly decreases as time passes—a fact observable by the aid of the horn. Moreover, in this apparatus the process of amalgamation is carried on to a considerable extent, and of course the longer the time the more opportunity is given to the quicksilver to settle.

The settlers should be watched as carefully as the pans. With intelligent care and proper construction they can be run off clean, and agitators can be dispensed with in treating tailings.

I consider that a larger diameter than 8 feet is not desirable. If greater capacity is needed, it should be obtained by increasing the height.

The quicksilver needs to be kept clean. For this, with proper precautions against grease and similar substances, moderate doses of sodium or cyanide of potassium, put in the quicksilver in the tanks, with careful and repeated sponging of the surface of the bath, generally prove sufficient. The main requisite is cleanliness, which includes careful handling. Efforts to substitute for this the use of various chemicals put in the pan have always proved failures. The proper principle is to keep the quicksilver from getting dirty, instead of trying to clean it after it has been allowed to foul.

At Dayton the loss of quicksilver (from all causes) averaged only one-half pound to the ton, being somewhat less than this figure for the sand, and from 0.75 to 1 pound for the slime-charges.

The fineness of the bullion did not average much over 150. It may be said, with some limitations, that the best results were obtained from sand when the bullion was about 150 fine; and from the (richer) slime-charges, when it was from 200 to 250. This

standard of fineness was less than that accepted by the other mills, more particularly when treating richer tailings. But after the Dayton refinery was established, nearly all the mills sent their base bullion to it for treatment and lowered their standard, which they were then able to do without any loss resulting from the increased baseness of their product. In the cases of the two mills (the Lyon and the Omega) which had refineries, the recovery of the copper in the state of sulphate and the increased yield of the tailings much more than compensated for the increased amounts of bluestone used in the pans. The Omega refinery paid for itself within two months after starting, and afterwards, as was the case at Dayton, yielded a respectable revenue; for, essentially, the whole product of base bullion at the Comstock was sent to the two establishments for refining, which was done without any direct charges.

The following table gives the cost per ton of working 46,500 tons of tailings in 1876. These expenses include the expenditures of all descriptions. In one item (hauling) I have substituted the average cost incurred in the following year, for the reason that the transportation-expenses in 1876, owing chiefly to a great mortality among the horses, were much above the general average.

Cost per Ton of Treating 46,500 *Tons of Tailings.*

A.—Preparing and Hauling Tailings:			
Labor on reservoirs, Chinamen	0.113 shifts at $1.51,...	.....	.....
" " whites	0.012 " at 3.003..	.....	.....
	0.125 shifts at $1.65...	$0 206	.....
Hauling, teams.....	.....	0.102	.....
Sundries.....	.....	0.005	$0.313
B.—Milling:			
Labor.....	0 113 shifts at $3 42...	0.385	.....
Quicksilver	0.45 lbs. at 62½ cts.....	0.282	.....
Salt.....	23.7 lbs. at 1.245 cts....	0.286	.....
Bluestone.....	12.3 lbs. at 4 cts.....	0.493	.....
Iron	3 1 lbs. at 6¼ cts	0.205	.....
Fuel	0.276 cords at $6.26....	0.173	.....
Sundry supplies.....	.....	0.121	1.945
C.—General:			
Assaying	.....	0.021	.....
Office-expenses.....	.....	0.013	.....
Legal exp., taxes and insur'ce	.....	0.065	.....
Bullion express-charges.....	.....	0.016	.....
Sundries.....	.....	0.033	.....
Superintendence.....	.....	0.069	0.217
Total per ton.....	.....	.....	$2.475

THE DEPARTMENT OF METALLURGY AND ECONOMIC GEOLOGY IN THE UNITED STATES NATIONAL MUSEUM.

BY F. P. DEWEY, WASHINGTON, D. C.

(New York Meeting, September, 1890.)

THE first systematic attempt of the National Museum to gather material representing the economic geology and metallurgy of the country was made in connection with the Centennial Exhibition. The department may be considered as having originated then, although it was not formally inaugurated until December, 1882. Previous to the Centennial, all the illustrations of economic geology had been included in the mineral collection, and the illustrations of metallurgy were so very few that they had also been included with the minerals.

When the museum began its collections for the Centennial, the subject of the mineral resources of the United States was placed in the hands of Professor Wm. P. Blake, who was eminently fitted to undertake the work by his experience at various international exhibitions and his extensive acquaintance with the mining industry of the country. Professor Blake prepared a broad and exhaustive plan for this purpose, and the very large collection shown at Philadelphia was well calculated to set forth the vast mineral wealth of the country.

The history of the department proper may conveniently be divided into three periods: first, the formative period, during which the Centennial material was mainly administered upon, ending July 1, 1884, with the beginning of the preparation for the New Orleans Exposition; second, the New Orleans Exposition period, from July 1, 1884, to June 30, 1887, during which the collections for the New Orleans Exposition were made, exhibited at New Orleans, returned to Washington, and installed permanently in the museum; third, the southwest-court period, from July 1, 1887, to October 1, 1889, during which the collections were arranged in the southwest court of the museum.

The first attempt to bring the collections systematically into condition for exhibition was made in 1882. In July of that year the writer was detailed to make a preliminary examination of the material, and on the first of December was formally appointed curator.

On account of the lack of proper facilities in Washington for the exhibition of these collections, they were, for the most part, stored in boxes after the close of the Centennial until the new building was so far advanced as to permit their being unpacked and arranged therein.

Attention was first given to the arrangement of the vast collection of ores. This was found to be very full and complete, illustrating every mining locality in the country of any prominence in 1876, and being especially rich in the ores of precious metals from the western states.

The illustrations of metallurgy exhibited at Philadelphia were very elaborate; but, owing to the fact that many of the specimens were examples of final products finished in the highest manner possible, and were, in most cases, examples of metal-working rather than metal-production, the collections were not at all systematic or complete. The specimens were just such as would be naturally expected at an exposition, being noteworthy rather for their beautiful appearance than as showing the steps in the process by which they had been produced from the raw material.

Much time and attention was absorbed in preparing a suitable card-catalogue of the collections. In the case of ores, besides the usual information as to locality, the mineralogical composition of each specimen was recorded. The general rule followed in this work was to determine and name every mineral plainly visible upon the specimen, and while in some cases other minerals might be (and, indeed, have been since) discovered, yet no name was put upon the card unless the mineral was undoubtedly present.

The coal-collections were entirely rearranged and consolidated. These had at one time been the most satisfactory part of the department; but many of the soft and easily decomposed coals had disintegrated, and were no longer fit for exhibition. Fully one-half of the material had to be thrown away.

A plan was suggested for carrying on a series of tests of the steaming power of the different fuels of the country, which should also include the formation of a large systematic collection for exhibition.

At the close of the Centennial, many of the foreign governments

presented their entire exhibits to the museum, while others presented portions of their exhibits. Many private exhibitors did the same. This department received a considerable amount of material from such sources, among which should be mentioned the large and complete collection of the ores and metallurgical products of the Krupp Works at Essen, Germany; the large and complete illustration of the mineral resources of Spain; a full illustration of the mineral resources of Portugal, especially rich in tin and antimony material; a small but interesting collection from Sweden, containing much systematic metallurgical material; large and complete illustrations of the mineral resources of the British Colonies, especially Australia, which are very rich in gold and tin specimens; a complete and interesting illustration of the mineral resources and metallurgy of Japan; and from Russia a considerable amount of material, representing both the mineral resources of the country and some of the metallurgical operations carried on there.

This brings us to July 1, 1884, the close of what might properly be called the first or formative period of the department; with regard to which the following statistics may be of interest: There had been made upon the book-record 9670 entries, covering 15,087 specimens. There had been 8235 entries upon the card-catalogue covering 13,041 specimens; and upon these the writer had made 9265 determinations of mineral species. The exhibition series embraced 9500 specimens. There had been received 202 accessions containing 1642 specimens.

During this period the following notable accessions were received: An interesting collection, representing the recent deposition of sulphur and sulphide of mercury at Sulphur Bank, California,* as observed by Prof. Joseph Leconte; a large series of cokes from nearly every coke-making center of the United States and England; two series of specimens representing the production of pig-iron of remarkable strength, one from the Wassaic furnace,† New York, the other from the Muirkirk furnace,‡ Maryland; a complete illustration of the operations of the Colorado Coal and Iron Company at South Pueblo, Colorado, in the manufacture of steel rails, nails and other articles; a collection of the iron-ores of California, and also a collection of copper and other ores from the Lake Superior region, the latter being particularly valuable from a historical standpoint; a

* See *Am. Jour. Sci.*, 3d series, vol. xxiv., p. 23; vol. xxv., p. 424; vol. xxvi., p. 1.

† See *Trans.*, xii., 91 and 520. ‡ See *Trans.*, xvii., 460.

collection of about 600 specimens, representing the process of manufacture of bolts, nuts and washers by machinery, together with the various commercial sizes of these articles; and a very interesting collection, representing the manufacture of busheled or refined iron from scrap-material.

By far the most interesting and valuable single collection received up to this time is the extensive collection of iron-ores made in connection with the Tenth Census,* under the direction of Prof. R. Pumpelly. This collection represented the occurrence of iron-ore at every mine of importance that was worked during the census year. Carefully selected samples were also taken for analysis, and the powdered material for this purpose accompanied the hand-specimens. The whole collection comprises about 4000 specimens.

As pointed out in the annual reports of the department, the representations of metallurgical processes were the weakest portions of the Centennial collections. This is a matter requiring the collection of systematic series, in which the various specimens are interdependent. The collections need not necessarily be at all showy, and it is therefore not surprising that the Centennial material should have been deficient in this respect. Fortunately, however, the necessity of collecting new material for the New Orleans Exposition furnished the much-desired opportunity of strengthening the collections in this direction.

It is not easy to get collections of this kind. Few owners of mines or works will take the necessary trouble to make up complete series, and those who do so, are in many cases unwilling to furnish the information necessary to confer upon the specimens the highest educational and scientific value.

The design in collecting for the New Orleans Exposition was to commence first with the ore and take series of specimens in the mine at various points, which should be referred to some central starting-point, so that the position might be known and traced up afterwards if desirable. These specimens were taken to show the various characters of ore produced by a given mine. Again, series of specimens were taken to represent the changes in the character of ore encountered in going up and down the mine, and also in going through it, both lengthwise and crosswise. The enclosing wall-rocks were also thoroughly represented.

In connection with this collection as much information as possible

* See vol. xv., Tenth Census Reports.

was obtained as to the methods of mining, and the customs of the miners.

After the ore was extracted from the mine it was followed through any process of concentration to which it was subjected, such as simple washing, or crushing and jigging. Each step in these operations was represented, when practicable, by specimens of the material forming the basis of the operation, and by the valuable and the waste products. To the material as prepared for smelting were added illustrations of all other materials entering into the smelting-operation, such as fuels and fluxes. Each step in the smelting was illustrated by the different materials entering into the operation, and by all its products. This was followed through a given mine or works, and, if practicable, on through other works, until the material reached the final, merchantable form. These collections were supplemented with all the information which could be got in the way of statistics, plans, drawings, and views of mines and works. In some cases the collectors themselves took photographs, and in other cases local photographers were engaged to do so. As the result of much effort in this direction, a very large collection of photographs illustrating mining and metallurgy has been secured, including many drawings, sketches, and specimens of tools. I need hardly say that, aside from the exhibition-value of such systematic collections in metallurgy, they open a wide field for research in the laboratory.

In selecting specimens it was designed to take, not specially handsome or rich ores, but rather such as represented the actual character, occurrence, and value of the ores. Collections made in this way have far more value for the museum than the hap-hazard exhibits of showy specimens usually found in such institutions.

The collections made for the New Orleans Exposition could not possibly cover the whole ground. In fact, the work was mainly preliminary; but it was hoped that the results obtained would be of sufficient value to warrant an extension of the number of such illustrations, and the inauguration of a system of collecting which would add largely to the material on hand, and in time make the collections in metallurgy compare favorably with those now in the museum in the line of economic geology.

The collections sent to the New Orleans Exposition were briefly described in Circular No. 31, Proceedings United States National Museum, 1884; and more fully in the annual report of the department for 1884.* They formed the basis of the present systematic

* *Smithsonian Report*, 1884, Pt. II., p. 248.

collections in economic geology and metallurgy, and will be mentioned in describing these collections.

In this connection the first serious attempt was made at preparing standard sets of printed labels for the exhibition-series of the department, and about 1500 were prepared. The forms adopted at that time have been more or less adhered to in the preparation of labels since, although the system has been developed to include several classes, which were not called for at that time. After getting the material of the department under tolerably good control, the preparations of labels became one of the most pressing needs, and at the same time one of the most difficult to fill.

At the close of the New Orleans Exposition much new material was added to the museum collections from the exhibits shown there. In this way several gaps in the collections were filled, particularly representations of regions that had come into prominence since the Centennial. Much very valuable material was secured from the Mexican exhibits.

At the close of the Centennial in 1876, the gratitude of foreign governments and private exhibitors for courtesies received from the American Institute of Mining Engineers, found expression in the donation to the Institute of many valuable collections. These were for some time kept in Philadelphia, and finally transfered to the National Museum.* These collections were particularly rich in illustrations of foreign practice in metallurgy.

Upon the return of the exhibits shown at the New Orleans Exposition it was decided to expand these collections, and install them in the West Hall of the Museum. During the work of arrangement a complete series of labels was prepared, and a preliminary descriptive catalogue of the collections was begun. The ore-series was completely reorganized and greatly enlarged by the addition of many specimens. The metallurgical series received some valuable additions, especially a collection from the Colorado Smelting Company, South Pueblo, Colorado.

This brings us to the close of the New Orleans period, June 30, 1887. During this period there were made upon the book-catalogue 12,690 entries, covering 16,565 specimens; there were made upon the card-catalogue 8175 entries, covering 10,500 specimens; and upon these the writer made 5000, and Mr. H. J. Biddle 2400, determinations of mineral species in the ores.

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* See *Trans.*, v., 25, 37; viii., 6, 280, 284; x., 243.

There had been received 581 accessions containing 16,338 specimens. Besides those received in connection with the New Orleans Exposition, two valuable accessions were received from the Boston and Colorado Smelting Works at Argo, Colorado, the first being a complete illustration of the Ziervogel process* for the extraction of silver from copper-ores, which contains a large series of the ores treated, including, besides the strictly Ziervogel ores, a variety of smelting-ores, the various mattes and slags produced, and the silver as precipitated by metallic copper; and the second collection being a very interesting series of gold and bismuth alloys.†

The total number of specimens in the collection was about 49,000, of which 18,000 were on exhibition, 12,000 being duplicates and 7000 reserves. The remainder consisted of specimens as yet unadministered upon, which were classed with the reserve material.

When it was decided to remove the entire exhibit of the department into the southwest court of the museum, the systematic collections, now in very good condition, were moved bodily and without any material changes, into the western half of the court.

For the purpose of installing the American geographical ore-series a long case was constructed running around the whole of the outside of the court. The ore-series was thoroughly overhauled and installed in this case.

After the opening of the court, the descriptive catalogue of the systematic collections was revised, and the collections were somewhat enlarged.

This brings us to the close of the writer's connection with the department, October 1, 1889.

During this last period there were made upon the book-catalogue 2274 entries, covering 3348 specimens, and upon the card-catalogue 7253 entries, covering 8053 specimens; 225 accessions, containing 3203 specimens, were received. Specially noteworthy among these were the three following:

Two of them are collections of petroleums. The first was made for the museum by Prof. S. F. Peckham, of Providence, Rhode Island, who prepared the Tenth Census report upon petroleum.‡ By the connections formed while making this report he was enabled to gather many specimens, especially those of historical value, which otherwise it would have been impossible to obtain. The collection

* See *Trans.*, iv., 276.

† See *Trans*, xiii., 738.

‡ See vol. x., Tenth Census Reports.

numbers 350 specimens, and illustrates very fully the occurrence of crude petroleum in this country, besides containing some representatives of foreign petroleums. A very complete illustration of the technology of the subject is included in this collection.

The second was a collection of 129 specimens received from Prof. J. J. Stevenson, of New York. While the specimens in this collection are mostly small, it constitutes a valuable complement to the Peckham collection, in which it fills several gaps.

The third was a complete illustration of the quicksilver industry at New Almaden, California, comprising a full series of the different characters of ores, together with the associated minerals and rocks of the deposit; a complete illustration of the process of extracting the metal; a very extensive and interesting series of views in and about the mines and works (including a few underground pictures which are especially valuable); a large number of publications relating to the history and development of the mines, and a very interesting and instructive glass model of the mine, showing the contour of the surface and the underground mine-workings.

Two important collections were turned over to the department by the United States Geological Survey. One of them, embracing 539 specimens, was collected by Mr. J. S. Curtis, in preparing his report on the silver-lead district of Eureka, Nevada;* the other was a very extensive collection, embracing 733 specimens, collected by Mr. S. F. Emmons, in preparing his report upon the geology of Leadville, Colorado.†

Of the specimens sent to this department in the ordinary way there is little that is of any value to the museum. They are, for the most part, sent for examination and report.

The total number of specimens in the department, October 1, 1889, was estimated at 51,000, of which 10,000 were on exhibition, 15,000 were duplicates and 12,000 were reserves, the remainder consisting of unclassified material which is considered as being in the reserve series.

Systematic Collections in Economic Geology and Metallurgy.

Of the exhibition series, 3691 specimens constitute the systematic collections in economic geology and metallurgy.

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* Monograph vii., U. S. Geol. Survey.

† Monograph xii., U. S. Geol. Survey.

These show in the case of each metal, first the series of minerals in which the metal forms an important constituent. The specimens are selected to show each mineral in its best perfection, in order that it may be seen just how the metals occur in the ore.

The next step is a series of ores selected to show the material actually mined, and the minerals associated with the ore. In the case of the base metals, there may not be much difference between the ore-specimens and the mineral specimens, except the greater purity and perfection of the latter. An ore, however, may contain several minerals of the same or of different metals, especially alteration-products of indefinite composition.

In the case of the precious metals, there may be a wide difference between the two kinds of specimens, according to the manner of occurrence of the valuable portion of the ore. One part of silver, for instance, distributed through a thousand parts of foreign material, may constitute a paying ore. If this one part be distributed more or less evenly throughout the whole mass, it will not be possible to detect any silver mineral in the ore. But if the same percentage of silver should be concentrated in separate portions of the mass, it might be possible to pick out distinctively silver minerals.

The same observations apply to gold, but in a much greater degree, since gold is so much more valuable, and, in some cases, can be extracted at much less expense than silver.

Frequently an ore contains more than one valuable metal. In many cases long and costly operations are required to separate the different metals, while in others their separation is quite easy. Sometimes a useful alloy can be smelted directly from ores containing more than one metal. These compound ores are particularly interesting; but they do not lend themselves to any system of classification.

After the ore is mined it is frequently subjected to some kind of concentration to prepare it for smelting. This varies from simple sorting by hand, whereby much worthless material is removed, to very elaborate treatment by machinery, whereby the valuable portion of the ore is concentrated to a smaller volume and weight, generally with a loss of a portion of the valuable constituents, and the waste material is rejected. In some cases, ores of different metals occurring together are separated from each other by simple mechanical treatment. Illustrations of these processes form the next step in the series, and comprise the ore as mined, and the products of each step in the dressing, including both the valuable and the waste products.

The final step of the series represents the processes of extracting the metals from their ores, and converting them into useful forms. These collections include the ores, fuels, fluxes and all other materials entering into the operation, and its useful products, waste-products, and by-products. To these, in many cases, are added various accessories, such as materials of construction and tools. Where there are different stages in the operation, each stage is fully represented.

The non-metallic ores are treated in the same general way as the metallic ores.

The collections described are but a beginning, and while generally complete in themselves, will require many additions before they can be considered as approximately covering the whole field of the economic geology and metallurgy of the United States.

Metallic Ores.

The mineralogy of gold is illustrated by twenty-nine specimens. Of these, twenty-eight represent free gold, and one sylvanite. Of the free golds, ten represent placer-gold. This series begins with a well-crystallized specimen that is only slightly water-worn, and continues through various degrees of rounding to a well-worn nugget, weighing an ounce. Fifteen specimens represent quartz-gold, beginning with crystals in a cavity in fresh quartz, and following through various steps of crystallization of the gold and decomposition of the matrix to a natural alloy of gold and silver in wholly decomposed material. Two specimens of crystallized auriferous pyrite represent sulphuret-gold, and a single specimen represents auriferous slate.

The gold-ores are divided into placer-, quartzose , sulphuret-, telluride- and fusion-ores. Besides the placer-golds in the mineral series, five specimens illustrate hydraulic gravels, varying from characteristic samples to very rich specimens, containing several flakes of visible gold. Nine specimens, in addition to the quartz-gold in the mineral collection, illustrate quartzose ores. These begin with very pure white quartz and follow through increasing amounts of other minerals, particularly sulphides, in the quartz, to a specimen containing large amounts of pyrite, chalcopyrite, galena and blende. The quartzose ores graduate insensibly into the sulphuret-ores, and the two really form one series. The ten sulphuret-ores begin with pyrite and quartz, and run through to pure sulphides; decomposed sulphides follow, and the series closes with two specimens of mispickel. Five specimens illustrate the telluride-ores from Boulder county, Col. No special collection of fusion-ores is shown, since

they are simply auriferous varieties of the ores of other metals with which they are shown. Good illustrations are found in the Argo and Newark collections.

The extraction of gold from its ores is illustrated by three collections. The operation of the hydraulic process is illustrated by 9 specimens from the North Bloomfield gravel-mine, California;* the stamping and amalgamating of sulphuret-ores by 7 specimens from the Bobtail mill, Black Hawk, Col.;† and the extraction of gold, silver and copper in a complex operation by 15 specimens from Balbach's Newark Smelting and Refining Works, New Jersey. This collection is especially interesting as showing the refining of copper and separation of the precious metals by the electric current.

The application of gold is illustrated by 11 specimens showing the manufacture of gold-leaf.

Seven specimens illustrate iridosmine. Of these, four show the original sand, and the gold dust, iridosmine and black sand separated by washing.

The mineralogy of silver is illustrated by 36 specimens, including all the ordinary silver minerals.

The silver-ores are divided into milling- and smelting-ores; these again are subdivided into free-milling and roasting-milling, and silver-smelting and silver-lead-smelting. A few, such as leaching-ores and argentiferous copper-ores, do not fall into this classification. Thirteen specimens illustrate the strictly free-milling ores, or such as readily yield to stamping and amalgamation. These carry mostly free silver or cerargyrite. Twelve specimens illustrate free-milling ores which require the use of chemicals in the pan; of these, 8 are characteristic ores from the Comstock lode. Ores requiring roasting before milling are illustrated by 5 specimens, and a complete series from the Ontario mine,‡ Park City, Utah. This latter collection contains 19 specimens selected by Captain Keith. Besides some general characteristic specimens, it includes a section along the 5th level and two sections across the vein on the 6th level. No good illustrations of silver-smelting ores were available, and the silver-lead-smelting ores are shown under lead.

The extraction of silver from roasting-milling ores is well illustrated by two collections made by the writer. The first, comprising 30 specimens from the Ontario mill,‡ illustrates every step in the

* See *Trans.*, vi., 31 *et seq.*

† See *Trans.*, ix., 97; xi., 40 *et seq.*

‡ See *Trans.*, viii., 551; xiii., 66 *et seq.*

process from the time the ore is dumped upon the grizzly until the bullion is shipped by express, and the tailings are turned to waste. The second collection represents in the same full way, with 17 specimens, the operation of the Moulton mill at Butte, Montana.

The treatment of argentiferous copper-ores is illustrated by a collection of 32 specimens from the Argo works, Colorado, selected by Mr. Pearce. This collection is particularly valuable on account of the assays and analyses accompanying the specimens.

The mineralogy of lead is illustrated with 14 specimens of sulphide, sulphate and carbonate.

Silver-lead ores are illustrated by 57 specimens. This series begins with roughly crystallized galena, showing the characteristic cleavage, and follows through several varieties of galena to the beginning of decomposition, and then through to complete decomposition, resulting in cerussite and anglesite. The association of galena with other sulphides is well illustrated, and a large collection of characteristic samples of cerussite closes the series.

The smelting and refining of base bullion is illustrated in three collections, from the Colorado Smelter, South Pueblo, Col.; the Cheltenham works, St. Louis, Mo.; and the Kansas City Smelting and Refining Works, Argentine, Kansas.

The collection from the Colorado Smelter contains 59 specimens selected by Mr. Eilers. It contains a full suite of the ores, fuels and fluxes used, the bullion and slag produced, and a variety of by-products, matte, speiss, flue-dust, accretions, etc. The value of the specimens in this case has been greatly enhanced by the large amount of information kindly furnished. The course of the operation has been fully described and platted. Nearly every specimen is accompanied by a careful and complete analysis. Numerous photographs show the disposition of the works. Taken as a whole, this collection is an excellent example of what such collections should be. The refining of base bullion is illustrated by a collection of 18 specimens from the Cheltenham works. The smelting of a charge containing a large proportion of oxidized ores, using iron to reduce the sulphide, and the subsequent refining of the base bullion, are illustrated by 13 specimens from the Kansas City works.

The southeastern Missouri lead-region is well represented by two collections, one containing 30 specimens from the St. Joe works, and the other 29 specimens from the Desloge works. These are particularly interesting as illustrating two distinct methods of separating the galena from the limestone, through which it occurs dis-

seminated. At the St. Joe works, the crushed material is delivered directly to the jigs without any sizing, while at the Desloge works, sizing is very carefully done. Mr. Kemp, who made these collections, has described the two mills in the *School of Mines Quarterly*.* A series of 28 specimens illustrates the smelting of galena with the recovery of the fume as a white paint at the Lone Elm works, Joplin, Missouri. The writer has fully described these works in a paper before the Institute.† The application of lead is illustrated by 11 specimens showing the Dutch process of making white lead.

The mineralogy of copper is illustrated by 22 specimens. All the ordinary copper minerals are shown.

The ores of copper are divided into metallic, oxidized and sulphuret-ores. The metallic copper-ores are very well represented by collections from three characteristic Lake Superior copper-mines,‡ "mass," "amygdaloid" and "conglomerate." The mass-mines, in which copper is found in large masses, are illustrated by a collection of 31 specimens from the Central mine. The amygdaloid-mines, in which the copper occurs disseminated through a soft amygdaloid rock, are illustrated by a collection of 61 specimens from the Osceola mine and 21 specimens from the Osceola mill. The conglomerate-mines, in which the copper occurs disseminated through a hard ferruginous conglomerate, are illustrated by 61 specimens from the Delaware mine and 17 specimens from the Delaware mill. These Lake Superior collections were made with great care for the Museum by special collectors. The location of nearly every specimen within the mine was carefully noted, and the collections include many complete sections. The smelting of the separated copper is illustrated by a collection of 17 specimens from the C. G. Hussey Works,§ Pittsburgh, Pa., and 11 specimens from the Lake Superior Native Copper Works. Taken altogether, these seven collections, with 219 specimens, form a complete illustration of the Lake Superior copper-industry.

Oxidized and sulphide-ores are shown in the collections from the various smelting-works.

The smelting of oxidized ores for pig-copper is illustrated by 14 specimens from the Copper Queen mine; and the refining of this

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* Vol. ix, pp. 74 and 212. See also Prof. Munroe's paper on the St. Joe Works, *Trans.*, xvii., 659.

† *Trans.*, xviii., 674.

‡ See *Trans.*, v., p. 584.

§ See *Trans.*, ix., p. 678.

pig to ingot is illustrated by 19 specimens from the Ansonia Brass and Copper Company, Ansonia, Conn. The smelting of sulphide-ores is illustrated by eight collections. The older form of the process is illustrated by 12 specimens from the Vermont Copper Company, Ely, Vt.; 23 specimens from the Schuylkill Copper Works, Phœnixville, Pa.; and 14 specimens from the Union Mining Company, Ducktown, Tenn. A newer form is illustrated by 8 specimens from the Orford Works, Bergenport, N. J., and a modification involving a long roasting of the matte in a reverberatory furnace, is illustrated by 34 specimens from the St. Genevieve works, Missouri. The smelting of sulphide-ores containing silver for the production of a rich matte, is illustrated by three collections from Butte, Montana. Reverberatory smelting is illustrated by 16 specimens from the Montana Smelter, and 10 specimens from the Parrott Smelter. Shaft-furnace smelting is illustrated by 11 specimens from the Bell Smelter. The treatment of copper-ores by the Hunt & Douglas* wet process, is illustrated by 14 specimens from the Schuylkill works, Phœnixville, Pa.

The application of copper is illustrated by 18 specimens, showing more particularly the rolling of copper.

Seventeen specimens illustrate the mineralogy of iron.

A slightly different plan was followed in the iron-ore series. Since we were fortunate enough to have such a complete illustration of the iron-ores of the country in the Tenth Census collection, we determined to utilize it in this connection. Therefore, the iron-ore series is arranged to show rather the geographical distribution of the metal than the scientific classification of the ores. The Tenth Census iron-ore collection was gathered with a great deal of care and trouble under the direction of Professor R. Pumpelly. Besides specimens of each characteristic variety of ore occurring at each mine, with the associate rocks, it includes an average sample for analysis; and most of the samples have been examined. These ores have been described in Vol. XV., Tenth Census. The whole collection numbers about 4000 specimens, of which 759 of the most characteristic have been placed on exhibition. In several cases, the associated rocks are shown. A few illustrations of manganese-ores are naturally included in the collection. The New England States are represented by 48 specimens, mostly limonite, with an occasional magnetite or siderite; New York by 141 specimens, mostly magnetite, but with

* See *Trans.*, x., 11.

some hematite and limonite and a very few of siderite; New Jersey by 95 specimens, of which 92 are magnetite and 3 limonite; Pennsylvania by 91 specimens, including magnetite, hematite and limonite; Delaware by 2 specimens of limonite; Maryland by 14 specimens, 13 being siderite and limonite and 1 magnetite; Virginia by 32 specimens, mostly limonite, with an occasional hematite or magnetite; North Carolina by 16 specimens, mostly magnetite, but with a few of limonite, hematite and siderite; Georgia by 32 specimens, mostly limonite, but with some hematite and manganese-ores; Alabama by 32 specimens, limonite and hematite, with a single siderite; Tennessee by 64 specimens, mostly limonite, but with a good representation of fossil-ores and an occasional manganese-ore; Kentucky by 16 specimens, mostly limonite, with a few of siderite; Missouri by 32 specimens, mostly hematite, with a few of limonite; Ohio by 48 specimens, mostly limonite and siderite, with a few of hematite; Michigan by 48 specimens, mostly hematite, but with a few of magnetite and an occasional limonite or manganese-ore; Wisconsin by 3 specimens of hematite; Minnesota by 13 specimens of hematite; Colorado by 12 specimens, magnetite, hematite and limonite; California by 4 specimens, magnetite, limonite and hematite; Utah by 10 specimens, hematite, limonite and manganese-ore; Oregon by 4 specimens, magnetite and limonite; and Washington by 2 specimens of limonite.

The direct extraction of iron from its ores is illustrated by 14 specimens from the Rogersfield mine and Belmont forge, New York. A large number of interesting photographs of the works accompany this collection. The manufacture of crucible steel from puddled bar is illustrated by a complete collection of 28 specimens from the Crescent Steel Works, Pittsburgh, Pa.

The manufacture of Bessemer steel is very fully illustrated by a collection of 74 specimens from the South Chicago works, taken by special collectors. This collection begins with the ores, fuels and fluxes used in the blast-furnace. Starting with a given charge, the metal was followed from the blast-furnace through each stage of manipulation until the rail was rolled, two hours and twenty minutes after the metal was tapped from the blast-furnace. Samples were taken wherever possible. To this systematic illustration of a blow are added many general specimens, particularly a full series of refractory materials.

It was designed to illustrate the smelting of pig-iron by collections from a considerable number of furnaces, selected to represent

the differences in the methods of running the furnaces, particularly such as are brought about by differences of fuel and ore used. This list is by no means complete. The smelting of magnetite with a mixture of anthracite coal and coke is illustrated by 19 specimens from the Crown Point, N. Y., furnace; the smelting of a mixture of several (6 to 8) different ores with anthracite and coke, by 16 specimens from the Warwick furnace, Pottstown, Pa.; the smelting of very sulphurous and cupreous magnetite from a single mine with a mixture of anthracite and coke, by 18 specimens from the North Cornwall furnace, Lebanon, Pa.; the smelting of limonite with coke, by 16 specimens from the Longdale, Va., furnace; the smelting of hematite-ores with a mixture of bituminous coal and coke, by 20 specimens from the Missouri furnace, Carondolet, Mo.; the smelting of hematite with coke for Bessemer pig, by 13 specimens from the St. Louis Ore and Steel Company, South St. Louis, Mo., and the smelting of fossil-ores with coke, by 17 specimens from the Roane furnace, Rockwood, Tenn.

To the iron collection is added a selection of 227 specimens of the results obtained by Kirkaldy, of London, in his classic investigation of the mechanical properties of Fagersta steel.

Manganese-ores are specially illustrated by 4 specimens.

The mineralogy of cobalt and nickel is illustrated by 4 specimens and the ores by 4 more. The application of the two metals is illustrated by 43 specimens.

The mineralogy of zinc is illustrated by 33 specimens. Among these are some very fine specimens of crystallized blende from Missouri. In addition to these, the ores are represented by 13 specimens.

Much blende associated with galena and a little pyrite occurs in a gangue of chert in southwestern Missouri and southeastern Kansas. The dressing of this material to separate the blende and galena is illustrated by 29 specimens from the South Side mill, Galena, Kansas.

The extraction of zinc is fully illustrated by five collections. The Glendale works at St. Louis, Mo., reducing a variety of ores, are illustrated by 22 specimens. The Joplin Zinc Works at West Joplin, Mo., smelting the separated blende from Galena, Kansas, are represented by 16 specimens. The Rich Hill, Mo., works, smelting the Joplin ore and using a Siemen's gas-furnace to heat the retorts, are illustrated by 17 specimens. The Passaic Zinc Works, Jersey City, N. J., are represented by 31 specimens. These works

treat the complex zinc-, iron- and manganese-ores from Franklin, N. J., and produce spelter, oxide of zinc and spiegeleisen.

The mineralogy of tin is represented by a single specimen; but the ore-collection includes specimens from nearly every locality in the United States, including the original tin-discovery at Jackson, New Hampshire, in 1840. It numbers 24 specimens; and to these are added 6 specimens of metal reduced from the ores, among them a small bar smelted from the Jackson ore in 1840. This is undoubtedly the "first tin ever smelted in America."

Antimony is represented by 7 specimens.

Chromium is represented by 7 specimens of chromite from all the principal locations in the United States.

Bismuth is represented by 2 specimens.

The mineralogy of mercury is illustrated by 5 specimens, and its ores by 28 specimens. To these are added 13 specimens illustrating the New Almaden, California, mine and works. This is an exceedingly interesting collection.

The mineralogy and ores of aluminum are illustrated by 7 specimens, exclusive of a large collection of clays. The extraction of aluminum is illustrated by 20 specimens, and its application by 22 specimens.

Under the head of the general use of metals are shown a large series of alloys. The interesting gold, silver and bismuth alloys described by Mr. Pearce* are illustrated by 12 specimens; the manufacture and utilization of brass, by 28 specimens; that of soft alloys, Babbitts, solders, etc., by 41 specimens; and that of type-metal, by 17 specimens. Welsh tin-plate is represented by 15 specimens.

Non-Metallic Ores.

These are illustrated as follows:

Sulphur by 7 specimens, and the manufacture of sulphuric acid by 5 specimens.

Grinding and polishing materials and their application by corundum in lumps and sizes, 15 specimens; emery, 28 specimens; quartz, 9 specimens; tripoli, 5 specimens; pumice-stone, 10 specimens, and the manufacture of sand-paper, 29 specimens.

The occurrence of asbestos is shown by a very large collection of 53 specimens from all over the world. The application of asbestos is shown by 30 specimens.

* *Trans.*, xiii., 738.

Twenty-four specimens show the occurrence of the various phosphates used in the manufacture of fertilizers.

Fictile materials are illustrated by 43 specimens, including all the usual varieties of clay.

The native compounds in which carbon forms the chief constituent, are shown in a series of 760 specimens, which begins with pure carbon in the form of diamond and graphite (25 specimens), and follows through coals (250 specimens); bitumen (10 specimens); petroleum (474 specimens); ending with natural gas (1 specimen). In this series the most important collections are that of anthracite coal from the Kohinoor colliery of the Philadelphia and Reading Coal and Iron Company, and the very extensive collection of petroleum, showing every variety from all portions of this country, with a complete illustration of its technology.

The systematic collections in economic geology and metallurgy have been fully described in a catalogue which will soon be issued by the museum. They are to be sharply distinguished from the geographical ore-series.

The following table summarizes the collections:

Cases.*		Minerals.	Ores.	Metallurgy.	Application.	Total.
1 and 2	Gold	29	29	31	11	100
2	Iridosmine	.....	.....	.....	.....	7
3 to 5	Silver	36	49	79	.....	164
5 to 7	Silver-Lead	.....	57	90	.....	147
5 & 8 & 9	Lead	14	.....	87	11	112
10 to 16	Copper	22	191	203	18	434
17 to 32	Iron	17	759	235	227	1238
26	Manganese	.....	.....	.....	.....	4
33	Nickel and cobalt	4	4	.....	43	51
34 to 36	Zinc	33	13	115	.....	161
37	Tin	1	24	6	.....	31
37	Antimony	.....	.....	.....	.....	7
37	Chromium	.....	.....	.....	.....	7
37	Bismuth	.....	2	.....	.....	2
38	Mercury	5	28	13	.....	46
28	Aluminium	7	.....	20	22	49
39 and 40	Alloys	.....	.....	.....	.....	113
41	Sulphur	1	6	.....	5	12
42	Grind'g and Polish'g Mat'r'l	.....	.....	.....	.....	96
43	Asbestos	.....	.....	.....	.....	83
44	Phosphate	1	23	.....	.....	24
45	Clay	.....	.....	.....	.....	43
46	Pure carbon	.....	.....	.....	.....	25
47 to 52	Coal	.....	.....	.....	.....	250
53	Bitumen	.....	.....	.....	.....	10
53 to 58	Petroleum	.....	.....	.....	.....	474
58	Natural Gas	.....	.....	.....	.....	1
						3691

* See Plan of Hall, page 26.

The Geographical Ore-series.

This series shows the mineral resources of the country, arranged upon a purely geographical basis, together with some important and characteristic collections from foreign countries. In its arrangement the geographical idea has been followed as closely as possible, but the necessities of the case-room have caused some departures, especially in the arrangement of counties within the States, and of some of the foreign countries. Different regions are represented as follows:

The New England States, by 102 specimens, of which 59 are iron-ores, associates and products; 11 silver-lead-; 3 copper-; 1 zinc- and 4 emery-ores, and 11 refractory material; 4 coal and 9 paints.

New York, by 114 specimens, of which 88 are iron-ores, associates and products; 6 limestone; 2 talc; 1 silver-lead-ore, and 17 refractory materials, mainly products.

New Jersey, by 74 specimens, of which 27 are iron-ores, associates and products; 31 represent the Franklin ores and their treatment, and 16 are clays.

Pennsylvania, by 135 specimens, of which 103 are iron-ores, associates and products (including a very complete illustration of the operations of the Powellton Estate, embracing 53 specimens); 7 zinc-; 13 copper-; 5 nickel-, and 1 chrome-ore, and 6 coal and coke.

Maryland, by 65 specimens, of which 57 are iron-ores, associates and products (including a complete illustration of the operations of the Muirkirk furnace, embracing 51 specimens); 2 gold- and 2 copper-ores; 3 refractory materials and 1 mica.

Virginia, by 92 specimens, of which 35 are iron-ores, associates and products; 27 gold- and silver-ores and associates; 8 zinc-; 3 lead-; 2 lead-zinc-; 7 tin- and 1 copper-ore; 6 pyrite and associates; 2 salt and 1 asbestos.

West Virginia, by 10 specimens of coal.

North Carolina, by 98 specimens, of which 8 are iron-ores; 50 gold- and silver-ores and associates; 11 copper-; 9 nickel-; 1 each of tin-, manganese- and chrome-ores; 2 each of barite, talc and garnet; 3 phosphates; 5 clays, and 1 each of corundum, asbestos and graphite.

South Carolina, by 14 specimens, of which 8 are gold- and silver-ores; 5 phosphates and 1 manganese-ore.

Georgia, by 63 specimens, of which 54 are gold- and silver-ores and associates; 3 iron- and 3 manganese-ores, 1 each of clay, corundum and asbestos.

Alabama, by 25 specimens of iron-ores, associates and products.

Tennessee, by 18 specimens of iron-ores.

Kentucky, by 29 specimens, of which 20 are iron-ores, associates and products; 7 coal and 2 lead-ores.

Arkansas, by 13 specimens, of which 4 are iron; 2 manganese; 3 silver-lead-zinc, and 1 antimony-ore, and 1 each of novaculite, soapstone and bauxite.

Louisiana, by 6 iron-ores and 1 salt.

Texas, by 9 specimens, of which 6 are iron- and 2 copper-ores and 1 refractory material.

Wisconsin, by 3 specimens of iron-ore.

Ohio, by 9 specimens, of which 4 are iron-ores; 2 coals and 3 clays.

Illinois, by 15 coals.

Indiana, by 5 iron-ores; 5 coals and 3 clays.

Michigan, by 125 specimens, of which 54 are from the numerous iron-mines of the State; 68 are from the copper-mines, and 3 are gypsum.

Missouri, by 228 specimens, of which 50 are iron-ores and associates, and 4 are coal; the remaining 174 specimens representing the complex zinc-lead-ores which occur in the State in great abundance and variety. With these are also included the copper and cobalt and nickel varieties.

Kansas, by 30 specimens of lead-zinc-ores, resembling the Missouri ores.

Oregon, by 27 specimens, of which 17 are gold- and 4 copper-ores; 4 iridosmine and 2 coal.

Dakota, by 3 tin-ores and 2 coals.

Montana, by 246 specimens, nearly all of which carry the precious metals, although base metals are frequently present in sufficient amount to be extracted, and in a few cases, as at the Butte copper-mines, form the chief value. From Deer Lodge county there are 30 specimens of gold, milling- and smelting-silver- and rich copper-ores; from Lewis and Clarke county, 78 specimens, of which 39 are silver-lead-ores from the Ten Mile district, and the remainder comprises gold-, milling- and smelting-silver- and a few copper-ores; from Silver Bow county, 60 specimens of milling-silver- and copper-ores, a large proportion of these specimens coming from the mines in and around Butte; from Meagher county, 4 specimens of silver-, copper- and lead-ores; from Jefferson county, 3 specimens of gold-, silver-lead-copper- and tin-ores; from Madison county, 29 specimens of gold- and free-milling silver-ores, in which chloride of

silver appears abundantly, and of tripoli; from Beaver Head county, 42 specimens of gold- and milling- and smelting-silver-ores.

Idaho, by 147 specimens, most of which are precious metal-ores, only a few of the smelting-ores being included. From Owyhee county there are 87 specimens, including placer-gold, and milling and smelting silver-ores; from Alturas county, 32 specimens of gold-, silver-lead- and mercury-ores; from Custer county, 14 specimens of silver-, lead- and copper-ores; from Boise county, 7 specimens of gold-ores; and from Lemhi county, 7 specimens of gold- and silver-lead-ores.

Colorado, by 234 specimens, nearly all of which carry the precious metals, while many also carry base metals, particularly lead, as in the famous Leadville mines. There are from Gilpin county 34 specimens, mostly auriferous pyrite and associates, but with a few silver-lead-ores; from Boulder county, 80 specimens, including many telluride-ores of gold and silver, silver-lead-ores, and auriferous pyrite, the latter two frequently carrying copper; from Clear Creek county, 19 specimens, mostly of complex silver-ores carrying also lead, zinc, and occasionally copper; from Park county, 12 silver-lead-ores; from Chaffee county, 13 silver-lead-ores; from Ouray county, 1 complex ore; from Fremont county, 7 silver-lead- and copper-ores; from Summit county, 3 complex silver-lead-ores; from Lake county, 35 specimens, mostly silver lead-ores, both carbonate and sulphides of the Leadville region; from Jefferson county, 11 specimens of auriferous pyrite, copper- and silver-lead ores, and clay and products; from Gunnison county, 7 silver-lead-ores; from Pueblo county, 1 zinc-ore; from Custer county, 8 complex silver-lead- and iron-ores; from Las Animas county, 2 iron-ores and coke; and from Arapahoe county, 1 coke.

Utah, by 402 specimens, which present the greatest variety of appearance and form the most attractive collection of its kind in the department. Many of the specimens are colored by decomposition products, giving every hue of the rainbow. Nearly all the ores carry the precious metals, and while many are valuable only for these, yet many more carry the base metals also in valuable quantities. Besides the precious metal-ores the collection shows iron-, antimony- and bismuth-ores, and coal.

Tooele county is represented by 113 silver-lead- and silver-lead-copper-ores. There are from Salt Lake county 163 silver-lead- and silver-lead-copper-ores; from Beaver county, 46 silver-lead- and bismuth-ores; from Summit county, 12 milling-silver and silver-

lead-ores; from Washington county, 7 free milling-ores; from San Pete county, 2 specimens of alabaster; from Millard county, 2 specimens of sulphur and silver-copper-ore; from Fremont county, 1 coal; from Iron county, 7 iron-ores; from Morgan county, 2 iron-ores; from Piute county, 3 silver-lead-copper-ores. The American antimony-mine is illustrated by 4 specimens; from Juab county, 3 silver-lead-ores; from Weber county, 1 iron-ore; from Utah county, 2 iron-ores; from Box Elder county, 34 silver-lead- and silver-lead-copper-ores.

New Mexico is illustrated by 43 specimens, about equally divided between purely precious metals and precious and base metals mixed.

Arizona, by 104 specimens, a few being precious metal-ores, but most being copper-ores without the precious metals.

Nevada, by 772 specimens, a very large proportion of which are ores of the precious metals solely, but a few smelting-ores, particularly those of the Eureka district, being also shown. There are from Lander county 74 specimens, of which 67 illustrate the beautiful and rich ruby-silver-ores of the Reese river district; from Washoe county, 125 specimens, of which 113 are from the various mines on the Comstock lode, and present, in marked contrast to the very beautiful and attractive Reese river ores, a dull and uninteresting general appearance. The difference between very rich and poor Comstock ore is too slight to be detected by the ordinary observer. Nevertheless the extraordinary production of bullion by these mines gives the collection special interest and importance. The remaining specimens from Washoe county show silver-, copper-, mercury- and sulphur-ores. There are from Esmeralda county 113 specimens, mostly milling-silver-ores, but including a few gold- and silver-lead-ores; from Lyon county, 5 copper-ores; from Nye county, 73 specimens, mostly milling-silver, with a few silver-lead-ores; from Humboldt county, 50 specimens, mostly milling-silver, with a few very good samples of sulphur; from Eureka county, 44 silver-lead-ores; from White Pine county, 162 specimens, mainly milling-silver, with a very few silver-lead-ores; from Churchill county, 7 specimens of borax, and silver- and lead-ores; from Elko county, 35 specimens, mostly silver-lead-, with a few milling-silver-ores; from Lincoln county, 74 specimens, mostly silver-lead-, with a very few milling-silver-ores; from Douglass county, 3 iron-ores; and from Ormsby county, 7 silver-lead-copper-ores and iron-ores.

California, by 203 specimens, mainly of the various gold-ores, in-

cluding quartz, pyritiferous quartz and gravel, and the mercury-ores of the State. There are from Butte county 18 gold-ores from both veins and gravel; from Mariposa county, 10 quartz-gold-ores; from Inyo county, 13 quartz-gold-ores containing considerable copper; from Calaveras county, 18 quartz-gold-ores and the Spenceville copper-ore; from Eldorado county, 7 quartz- and gravel-gold-ores; from Nevada county, 20 quartz- and gravel-gold-ores; from Mono county, 3 quartz-gold- and antimony-ores; from Tehama county, 1 sulphur specimen; from Santa Barbara county, 2 specimens, asphalt and gypsum; from Placer county, 21 quartz- and gravel-gold-ores; from Lake county, 7 mercury- and sulphur-ores; from San Luis Obispo county, 5 mercury-ores; from Del Norte county, 2 copper-ores; from Sonoma county, 14 mercury-ores; from Tuolumne county, 11 placer- and quartz-gold-ores; from San Bernardino county, 6 silver-lead-copper-ores and tin-ores; from Monterey county, 1 mercury-ore; from Kern county, 6 silver-ores and antimony-ores; from Sierra county, 5 quartz- and gravel-gold-ores; from Plumas county, 7 quartz- and gravel-gold-ores; from Yuba county, 5 gravel- and quartz-gold-ores; from San Diego county, 4 gold-ores and copper-ores; from Napa county, 4 mercury-ores; from Tulare county, 4 specimens of gold-, silver- and copper-ores; from Santa Clara county, 3 mercury-ores; from Shasta county, 3 specimens of gold-, silver- and lead-ores; and from Fresno county, 3 mercury-ores.

Alaska, by 16 specimens of gold-ores, many of which contain mispickel, and 5 coals.

Mexico, by 133 specimens, mostly of silver-ores, although other metals are also represented. Of the beautiful and rich silver-ores of Zacatecas, there are 17 specimens; of copper-ores (mainly utilized in the preparation of sulphate of copper for extracting silver), from the State of Aquas Calientes, 3 specimens; from the State of Sonora, 50 specimens, including some very rich gold-quartz, silver-, silver-lead- and copper-ores and coal; from the State of Pachuca, 5 silver-ores (in the reserve series there is also a large systematic collection illustrating the mines of the Real del Monte Company); from the State of Mexico there are 12 specimens of silver-, sulphur- and iron-ores and clay; from the State of Chihuahua, 14 silver- and copper-ores; from the State of Durango, 9 silver-, tin- and iron-ores; from the State of Oaxaca, 13 silver-, silver-lead-, copper-, tin- and iron-ores, and coal and gypsum; from the State of Puebla, 2 manganese- and mercury-ores; from the State of San Luis Potosi, 4 tin-

ore, sulphur and gypsum; from the State of Vera Cruz, 1 specimen of pumice; from the State of Jalisco, 3 specimens, iron-ore, zinc-copper-ore and barite.

San Salvador, by 51 silver-, silver-lead- and iron-ores, and lignite.

Costa Rica, by 16 free-milling silver-ores.

British America, by 92 specimens, including iron, manganese-, chromium-, copper-, silver- and silver-lead-ores, and coal, graphite, phosphates and gypsum.

Japan, by 95 gold-, silver-, silver-lead-, copper-, iron-, mercury-, tin-, antimony- and chromium-ores, and graphite, coal and petroleum. This collection is especially interesting on account of the illustrations of the native processes of extracting copper, silver-lead and iron.

Portugal, by 145 specimens, particularly rich in antimony- and tin-ores. Lead-ores, some of them argentiferous and zinciferous, and processes of dressing, are also well illustrated. Besides these, the collection includes copper-, iron- and manganese-ores, and coals.

Spain, by 206 specimens, principally from the collection shown by the Royal Engineers at the Centennial. It includes lead- and silver-lead-ores and processes of dressing, copper-, iron-, manganese-, zinc-, tin- and sulphur-ores, and coal, asphalt and clays.

Russia, by 185 specimens, including a complete illustration of the mineral resources prepared by the Mining School of St. Petersburg, and several suites from prominent mines and works. The ores include those of gold, silver, platinum, copper, lead, bismuth, molybdenum, iron, manganese, chromium and zinc, and coal and corundum. The iron- and copper-works are well illustrated.

Sweden, by 170 specimens, mostly iron-ores, associates and products, a few copper-ores being also shown.

Norway, by 12 specimens of nickel-ore, coal, feldspar and phosphate.

Great Britain, by 19 specimens of tin-ore, clay, coal and asphalt.

South America is imperfectly represented by 61 specimens; Brazil having 36 gold- and iron-ores, and coal; Peru, 5 silver-lead ores; Venezuela, 6 copper-ores, coal and ochre; Chili, 8 copper-ores; and Bolivia, 6 copper-ores.

New Zealand, by 65 specimens, including quartz- and placer-gold, ores of silver, lead, copper, iron, zinc and antimony, and sulphur and coal.

Australia, by 290 specimens, especially rich in gold- and tin-ores.

There are from Victoria 138 specimens, of which 30 are models of famous nuggets, most of the rest being gold- and tin-ores, with a few specimens of antimony-, copper-, iron-, manganese- and zinc-ores and coal; New South Wales, 152 specimens, of which 46 are tin-ores and associates, the remainder being gold-, antimony-, copper-, iron- and lead ores; and from Tasmania, 12 tin- and iron-ores.

The celebrated works of Krupp, at Essen, Germany, are represented by 185 specimens, and show a complete series of the iron- and manganese-ores used, fuels, fluxes and a few crude products.

The interesting region about Carinthia in Austria is represented by 164 specimens, showing the various silver-, lead-, copper-, zinc-, iron- and manganese-ores, the processes used to separate the silver, lead, copper and zinc, and the products of the iron- and manganese-ores.

The following table summarizes the geographical ore-series:

Cases.*		Cases.	
101 and 102	New England States....102	136 to 138	Idaho....147
103 to 105	New York....114	139 to 144	Colorado....234
106 to 107	New Jersey....74	145 to 151	Utah....402
108 to 111	Pennsylvania....155		152 New Mexico....43
112	Maryland....65	153 and 154	Arizona....104
113 and 114	Virginia....92	156 to 167	Nevada....772
114	West Virginia....10	168 to 171	California....203
115 and 116	North Carolina....98	155	Alaska....21
116	South Carolina....14	172 to 174	Mexico....133
117	Georgia....63	175	San Salvador....51
118	Alabama....25	175	Costa Rica....16
118	Tennessee....18	176 and 177	British America....92
118	Kentucky....29	178 and 179	Japan....95
118	Arkansas....13	180 to 183	Portugal....145
119	Louisiana....7	184 to 187	Spain....206
119	Texas....9	188 to 190	Sweden....170
119	Wisconsin....3	191	Norway....12
119	Ohio....9	192 to 195	Russia....185
120	Illinois....15	196	Great Britain....19
120	Indiana....13	197	South America....61
121 to 123	Michigan....125	197	New Zealand....65
124 to 127	Missouri....228	198	Tasmania....12
128	Kansas....30	199 to 203	Australia....200
129	Oregon....27	204 and 205	Austria....164
129	Dakota....5	206 and 207	Germany....185
130 to 135	Montana....246		

Besides these two systematic collections, a considerable amount of miscellaneous material of a more or less decorative nature, particularly large and showy specimens of ores, is displayed about the hall.

A large safe, especially designed for the purpose, holds 107 very rich specimens, most of which are placer-gold, representing every prominent region of this country and a few foreign countries, particularly New Zealand, from which 45 specimens are shown.

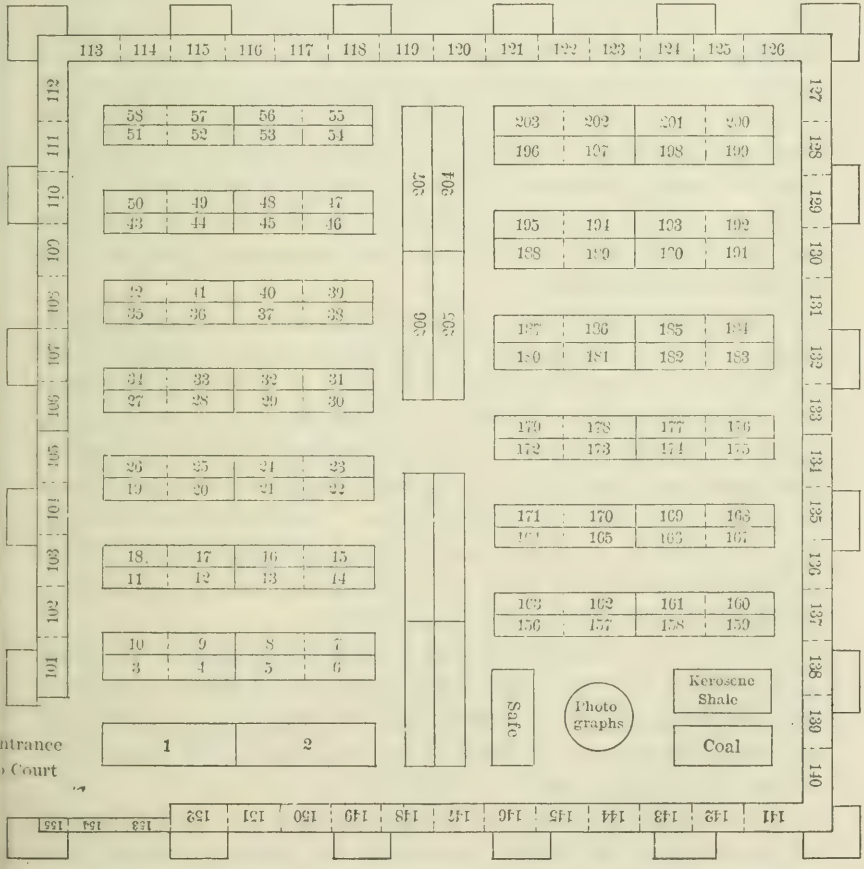
Upon the tops of the cases are shown samples of finished pro-

* See Plan of Hall, page 257.

ducts, highly-wrought articles and tested specimens, tools and models, and upon the walls are many photographs, charts and drawings, illustrating the subject generally. Many photographs are shown upon a swing-pillar screen.

On the west front of the museum building are arranged many ore-specimens which are too large to be exhibited inside. Here are also several group-collections illustrating large areas, the individual members of which are not accurately located in the respective regions, especially a large collection of Swedish magnetic iron-ores and German siderite iron-ores.

The following plan shows the general arrangement of the hall:



Department of Metallurgy and Economic Geology in the U. S. National Museum. Ground-Plan.

ELECTRIC POWER-TRANSMISSION IN MINING OPERATIONS.

BY H. C. SPAULDING, BOSTON, MASS.

(New York Meeting, September, 1890.)

THE rapid increase, during the last few years, in the number and magnitude of applications of electric power-transmission to commercial uses in this country, has been due principally to three causes: first, the ability and enterprise of those who have been identified with the progress of electrical industries, since electric illumination became an economic fact instead of a laboratory-experiment: secondly, the readiness of the American people to adopt new and advanced methods as soon as their superiority has been fairly demonstrated; and thirdly, the economy and flexibility of the apparatus employed, when properly designed and constructed, and installed under suitable conditions.

Basing our judgment on the world's advance in scientific developments during the past century, we may reasonably conclude that in no other country would the pioneers of electrical invention, whose names have now become household words, have received the prompt and plentiful financial support which has enabled them to inaugurate undertakings of exceptional magnitude, even in this era of tremendous monetary organizations. And, on the other hand, we have seen, in the numerous financial and engineering failures inseparable from such rapid development, the natural results of a too implicit faith in electrical omnipotence.

It is not to be wondered at that conservative engineers have been slow to adopt this new and mighty agent in the solution of extensive problems, and under circumstances where a single failure means not only a greater or less financial sacrifice, but the loss, in a greater or less degree, of professional reputation.

The mechanical engineer, while recognizing the fact that thousands of horse-power are to-day transmitted by electrical means, from running the lathe in a dentist's office to the operation of a fifty-ton travelling crane; from passing a cooling draught of air through a

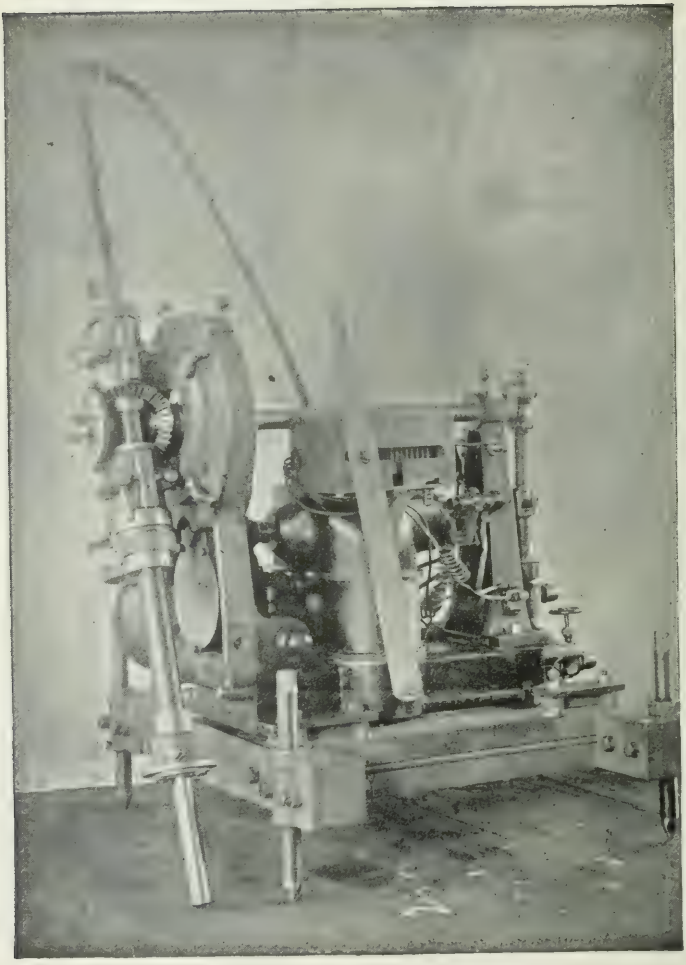


Fig. 1.—ELECTRIC DIAMOND DRILL

sick chamber to printing the plethoric sheets of our daily papers, hesitates, nevertheless, before placing reliance upon a system which has undeniably had its failures.

The mining engineer looks with interest upon the statement that 258* electric roads in the United States alone, with 1927.26 miles of track, and 3024 cars are in daily service or under construction, and then compares the demands of street-service with those found in mining practice, looking meanwhile for practical suggestions from those who have "been there" and can hold up a warning or beckoning finger to the traveller in this unfrequented path. It is the object of this paper to present briefly some of the work already done towards the application of electrical apparatus to mining processes, as well as to embody some practical suggestions and statements from those who have had personal experience in the operation of such apparatus.

Following the natural order of operations, let us consider drilling and cutting machinery in the first place. Fig. 1 shows the drill manufactured by the Diamond Prospecting Company of Chicago. This is designated by them as a Type "R" machine, nominal capacity 300 feet, and equipped with a motor of 3 horse-power, rated capacity. The total weight of the machine, set up and running, is 1000 pounds, and the heaviest piece, when taken apart for shipment, weighs about 170 pounds. The machine is mounted on trucks fitting the gauge of the mine-track for easy handling, and can be taken apart in fifteen minutes, and put up in a half-hour without difficulty.

The drill swivels so that holes can be put in at any angle, and can be operated in a space giving 5 feet in the line of the drill-rods.

The general arrangement of electrical and mechanical parts is sufficiently evident from the illustration, though the pump is hidden by the pedestal on the right. This is operated by the horizontal shaft driven by the bevel-gearing shown, and supplies a constant stream of water to the diamonds through hose connected with the top of the drill-tube. In a recent test on a granite boulder, hole $1\frac{1}{2}$ inches in diameter, with slow speed, this drill cut 22 inches in 40 minutes. When set up in the mine and working on hard compact limestone, it cut the rock at the rate of 1 inch per minute, not including stoppages for changing the rods.

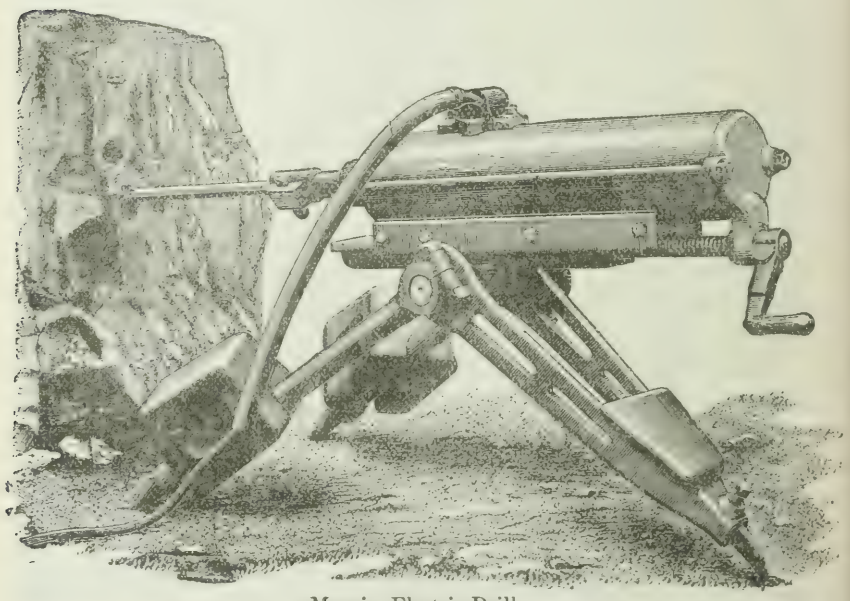
For general prospecting-purposes this drill seems to satisfactorily meet the requirements of its special line of work.

* August 1, 1890.

The general principle of construction of most electro-dynamic machinery provides us with a rotary motion, which it is necessary to transform to a reciprocating in order to obtain drills of the Rand or Ingersoll type. Any such transformation, however, entails a considerable loss of power, and we are happily relieved of this necessity by recent inventions which are based on the general characteristics of the solenoid.

Fig. 2 shows such a drill, constructed under the patents of H. N. Marvin, of Syracuse. Mr. C. J. Van Depoele, of Lynn, Mass., was one of the first to appreciate the demand for this class of apparatus, as well as the possibilities of the principle involved; and a large factory is under course of construction by the Thomson-Houston Company, which will be especially devoted to drilling and pumping machinery of the reciprocating type.

Fig. 2.



Marvin Electric Drill.

A description of the general features of these drills will be found in the communication of Mr. Van Depoele (page 273).

Many machines have been placed upon the market, in recent years, for making the under-cut in soft-coal mining. Several of them have met with a certain degree of success in clean and easily-worked veins, steam or compressed air (generally the latter) being

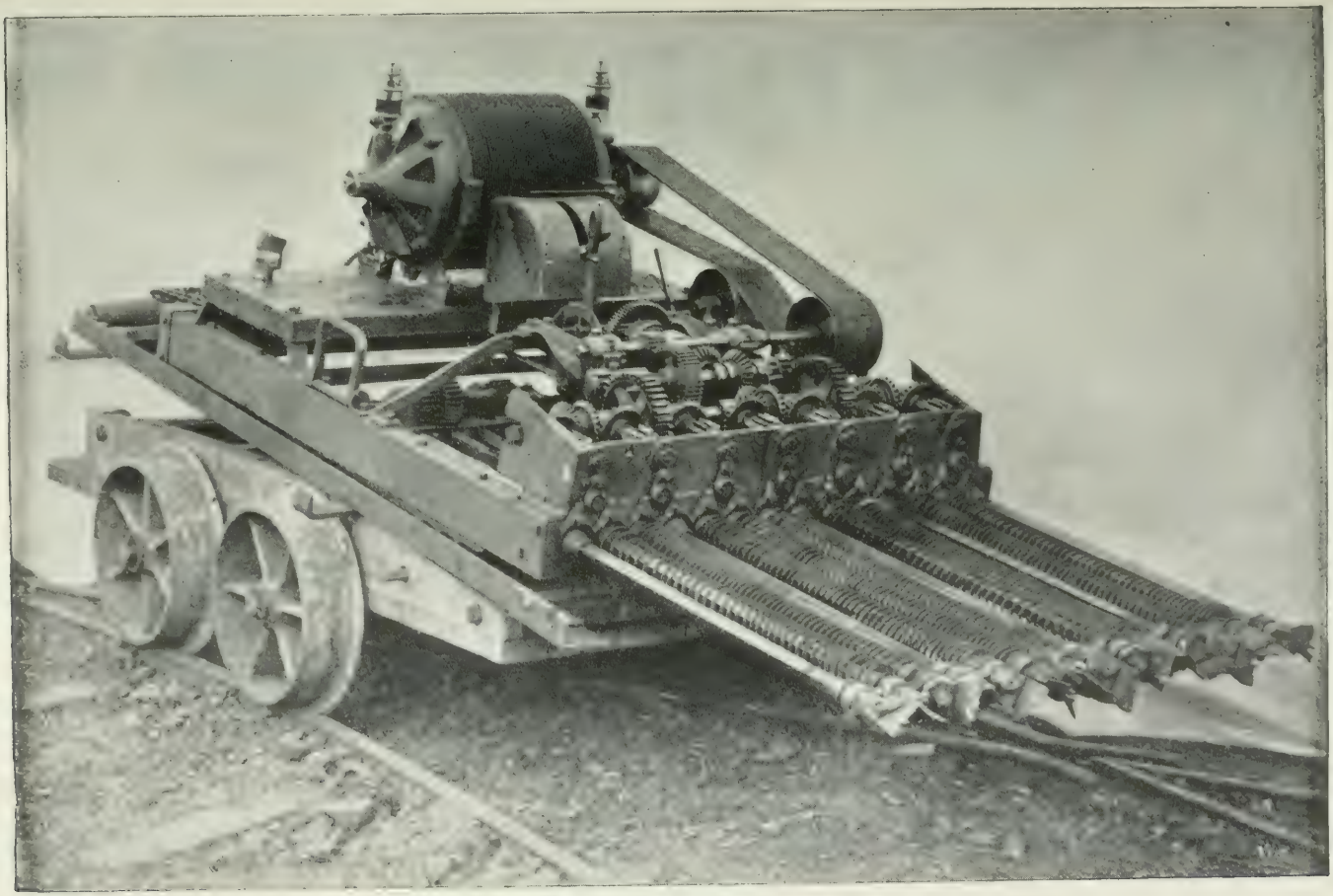


FIG. 3.—ELECTRIC COAL-CUTTER OF THE HERCULES MINING MACHINE COMPANY

the operating force. Perhaps the most successful of these machines has been that manufactured by the Jeffreys Mining Machine Company, of Columbus, Ohio; and a communication from Mr. Doe, their engineer, will be found appended (page 276).

Appreciating the advantages of electric power for this class of work, Mr. Jeffreys was not slow in adopting a motor, and has used a number of those manufactured by Foree Bain, the coal-cutter being modified somewhat to utilize this power to the best advantage.

In the opinion of many practical coal-miners, the principle of the rotating drill has many points of superiority in under-cutting machines, and Fig. 3 shows the result of considerable experimenting and outlay in this direction by the Hercules Mining Machine Company, of Pittsburgh. A series of drills is operated by a Tesla alternating motor, the power being transmitted by a belt as shown, and the current being supplied to the motor by three armored cables. When in operation, the cutter is clamped upon rails parallel with the face of the coal, being shifted along this track after each cut, ready for another.

Compressible springs are wound upon each drill-rod, and serve as conveyers for the coal-dust cut out by the drills.

A machine also embodying the boring principle, though entirely different in its mechanical and electrical design, is shown in Fig. 4.

A Thomson-Houston motor of a special type is used in this machine, current being supplied from the main-entry wires (which may also be used for lighting, haulage, and pumping) by flexible wire-covered cables.

The series of cutters (nine in number, each 4 inches wide) is so arranged as to cut close to the wall beside which the machine is placed, and within $\frac{1}{8}$ inch of the level of the floor.

Although the weight of the apparatus complete is less than 1400 pounds, no clamping is needed, as the drills will "pull themselves into the coal," with only the friction of the machine behind them as it rests on the floor. The device for clearing the drills of coal-dust is as simple as it is effective. It consists of a series of hinged scrapers hung from the under side of reciprocating bars, one of which is placed between every pair of drills, and which also operate an ingenious device for cutting out the triangular space left by the drills at top and bottom of the cut. The present type of machine makes a cut 3 feet wide, 5 feet deep, and 4 inches thick, in two minutes and a half, including withdrawal of the drills. With these figures in mind, the claim of 180 tons capacity per day for this machine does not

seem excessive. As the extreme height of the apparatus is only $23\frac{1}{2}$ inches, it can be used in a vein of any thickness, and two small drums are so placed in the rear of each machine as to enable it to

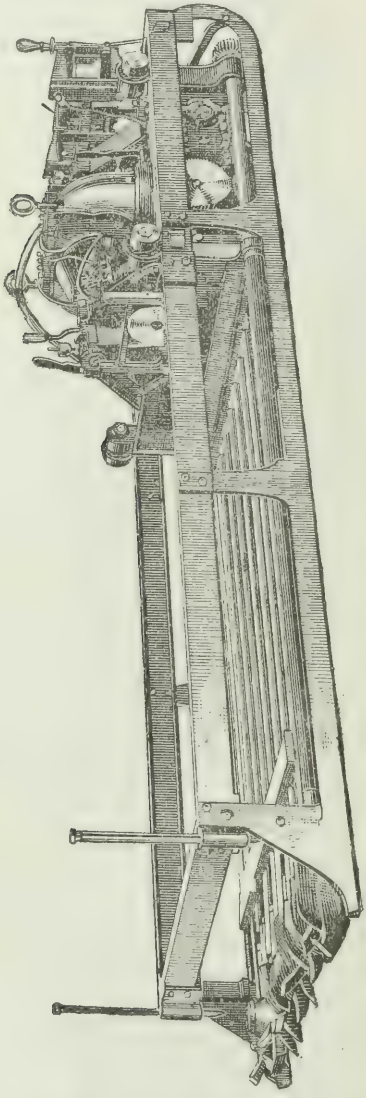


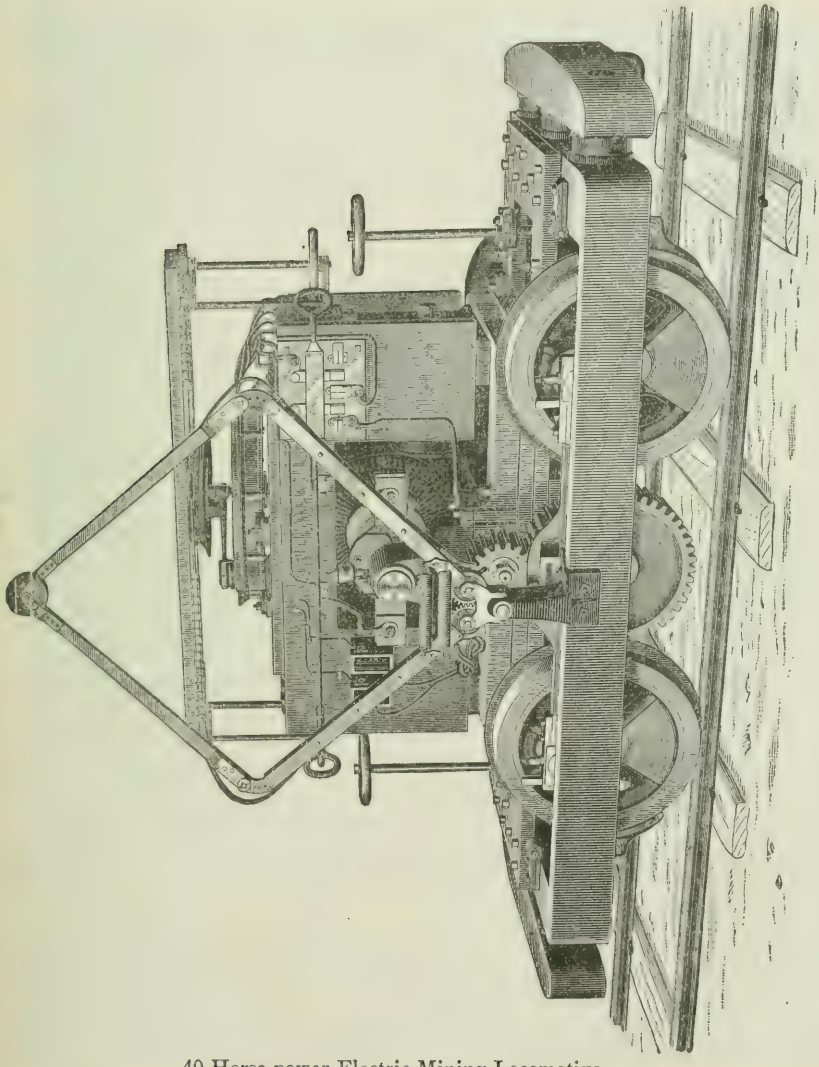
FIG. 4. Electric Coal Cutter.

draw itself on to a truck which accompanies each machine, ready for moving into another chamber.

A simple but ingenious rotary drill is being constructed for opera-

tion in connection with this apparatus, deriving its power from it by a flexible shaft, and drilling the blast-holes at the same time that the vein is being undercut.

FIG. 5.



40 Horse-power Electric Mining Locomotive.

Having thus briefly examined the present electrical apparatus for boring into ore or coal, so that it may be blasted out, let us see what means are at disposal for hauling it out of the mine, or from one point to another under the surface. Of course, in the majority of

cases, the blast will be fired by the electric current, but a discussion of this application hardly comes within the scope of this paper.

To Mr. W. M. Schlesinger, of what was, at the period referred to, the Union Electric Company, is due the honor of constructing the first electric locomotive for strictly mining uses in the United States. This was of 35 horse-power rated capacity, and was put in by the Lykens Valley Coal Company, Lykens Valley, Pa.

A series of iron rails were joined together to form a conductor for the current, which, after passing through the motor, completed its circuit to the generator by the track-rails, which were connected also by copper wires.

A locomotive of 40 horse-power capacity has been in operation at the Hillside Coal Company's Erie Colliery, near Scranton, Pa., and the installation is fully described here as showing very fairly the conditions to be fulfilled by this class of apparatus in the anthracite coal-regions of Eastern Pennsylvania.

The power-plant consists of a standard Armington & Sims engine capable of developing 60 horse-power, and a 50 horse-power Thomson-Houston generator wound for a current of 220 volts potential, and the necessary appliances for its operation. The engine- and dynamo-room at the top of the shaft are in charge of the engineer and assistant, who operate the other mining machinery.

From the dynamo to the foot of the shaft the current is conducted by No. 0 Clark wires, enclosed in gas-pipes to protect them from damage. From the bottom of the shaft the wires are carried overhead, about 12 inches outside of the low rail of each track, and are suspended from an insulator specially designed for this class of work, the construction of which can be readily seen from the accompanying illustration (Fig. 7).

Wherever turnouts occur, frogs (shown in the cut) are used, the conductors being soldered to them in the same manner as when used for street-railway work. Connections from the mains to the overhead conductor are made at suitable intervals, and a portion of the current is utilized for lighting purposes, two 110-volt lamps being placed in series. There are fifty of these lamps—eight at the foot of the shaft, two in the pumping-room, four in the blacksmith-shop, and two in the slope-room; the remainder being distributed along the gangway, as shown by the black dots in Fig. 8, Plate I.

The rails are used as conductors for the return-current, copper end-connections effecting a complete metallic circuit. In adapting the tracks to the electric system it was found necessary to make a few changes to accommodate the increased output. The shaft-sidings, as shown in the plan, will accommodate seventy loaded cars and fifty empties, whereas, before, they had a capacity for but fifteen on each side. The profiles and plan of the tracks will be seen from the diagram (Fig. 8). The track from X to R, or foot of the rock-plane, is 36-inch gauge. The other tracks are double-gauge, or three-rail tracks, the gauges being 36 and 28 inches. They are made in this manner to accommodate the slope-wagons, which are all of the narrower gauge.

The locomotive shown in the illustrations (Figs. 5 and 6) embodies many new fea-

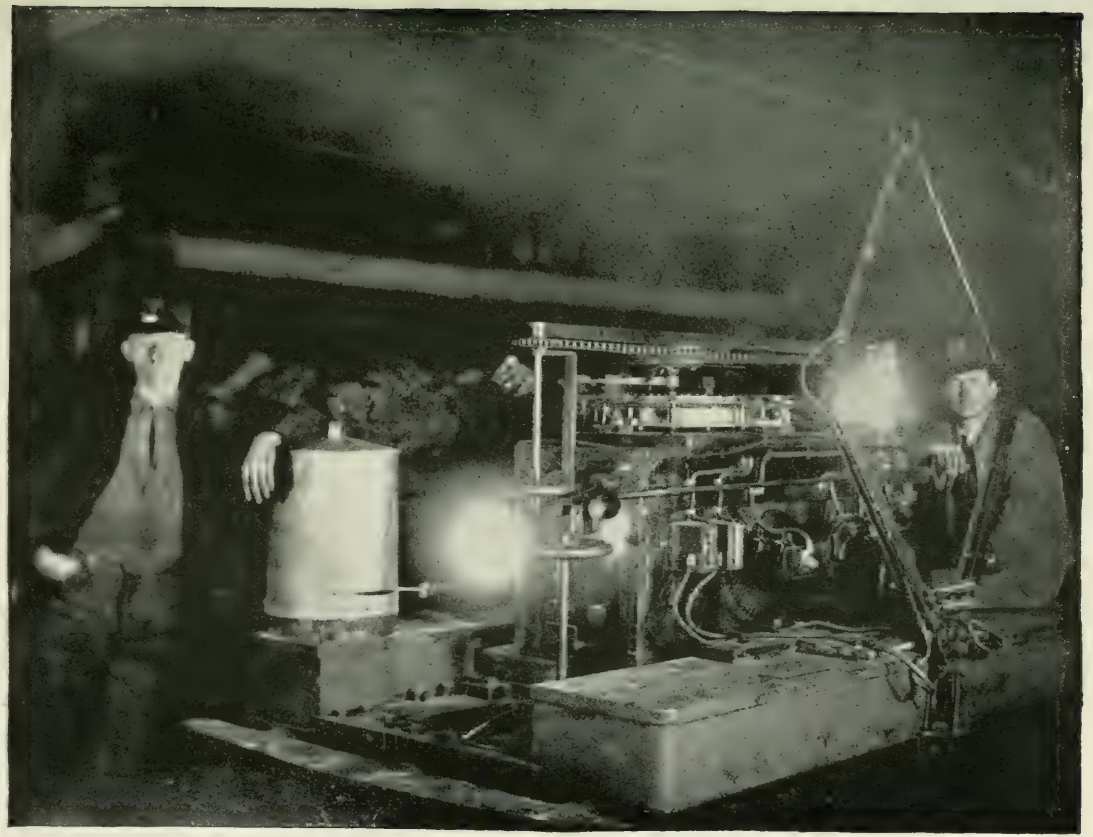
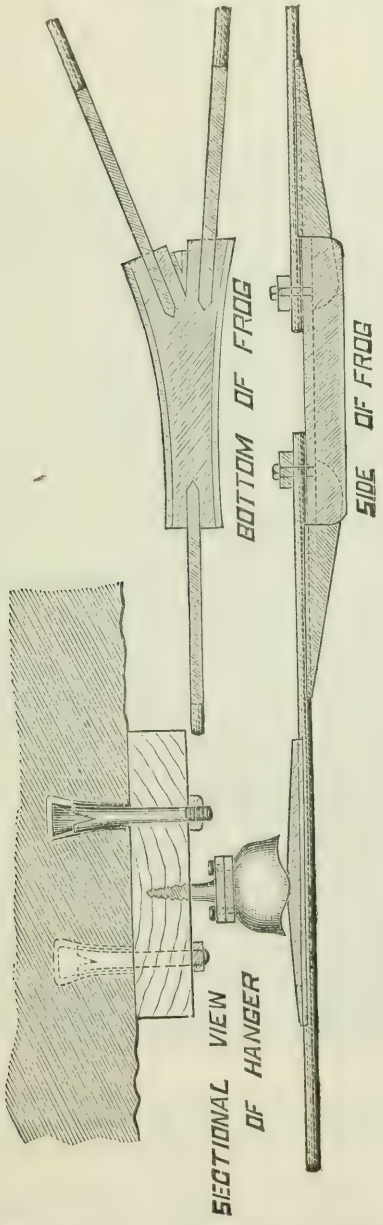


Fig 6.—ELECTRIC MINE-LOCOMOTIVE

tures in motor-construction and general design, and, under practical test, has shown that it is particularly adapted to the work required of it. It is built for a 3-feet

FIG. 7.



gauge, and is of the following dimensions: Length over all, 9 feet 7 inches; width, 5 feet 3 inches; and height, 5 feet 6 inches. This last dimension can be consider-

ably reduced by placing the rheostat at one end instead of on the top, as has been done in the present instance. The weight of the locomotive is 10,500 pounds, to which 1800 pounds has been added to increase traction. The motor employed is of the type "G" railway motor of 40 horse-power.

A novel trolley-arm is used, which requires no attention when the motor is reversed. Its construction is such that a wide variation in the position of the conductor is permissible, a range of 3 feet 6 inches being easily covered, while the meeting of an obstruction simply causes the trolley-arm to fall by the side of the car without resulting in any damage. From the trolley-wheel the current passes along the arm to the fuse-boxes, then through the rheostat and motor to the rail. Pinions on the armature-shaft mesh with intermediate gears, connection between these and slotted connecting-rods being made through the ordinary crank-pin and box. This arrangement allows for variation in position between the wheels and body of the locomotive which carries the motor; and, as the crank-pins on opposite sides are placed at an angle of 90 degrees, there are no dead points. The brake mechanism, rheostat and reversing-switch may be operated from either end by the hand-wheels shown in the cut. The operator has everything under complete control, and can start or stop the car and reverse its direction, without moving from one position.

The locomotive is run by one man, who is assisted by a boy in making up the trains and turning the switches. It displaces seven mules and three drivers. During a period of 11½ days the average number of cars delivered at the shaft-bottom by the locomotive was 559.5, against 526.95 per day delivered by mule haulage, much time being consumed by waiting at the bottom of the shaft for empty cars. Thus far, the locomotive has shown that it will increase the daily output to 700 cars per day. The operations are as follows:

East or Slope Side.

Distance run per trip, including making up, etc.,	2884 feet.
Time of trip,	10½ min.
Cars per trip,	15
Trips per day,	16
Miles run per day,	8.73
Total time,	2 h. 40 min.

Locomotive reversed 128 times per day.

West or Plane Side.

Distance run per trip, including making up, etc.,	2546 feet.
Time of trip,	6½ min.
Cars per trip,	20
Trips per day,	25
Miles run per day,	12.55
Total time,	2 h. 50 min.

Locomotive reversed 104 times per day.

To deliver 700 cars per day of 10 hours, the time of running the locomotive is 5 hours and 30 minutes, leaving 4 hours and 30 minutes for contingencies. The total distance run is 21.28 miles, and the locomotive is reversed 232 times.

The Thomson-Houston Company has now under construction for another Pennsylvania mine, a locomotive of 60-horse-power capacity,

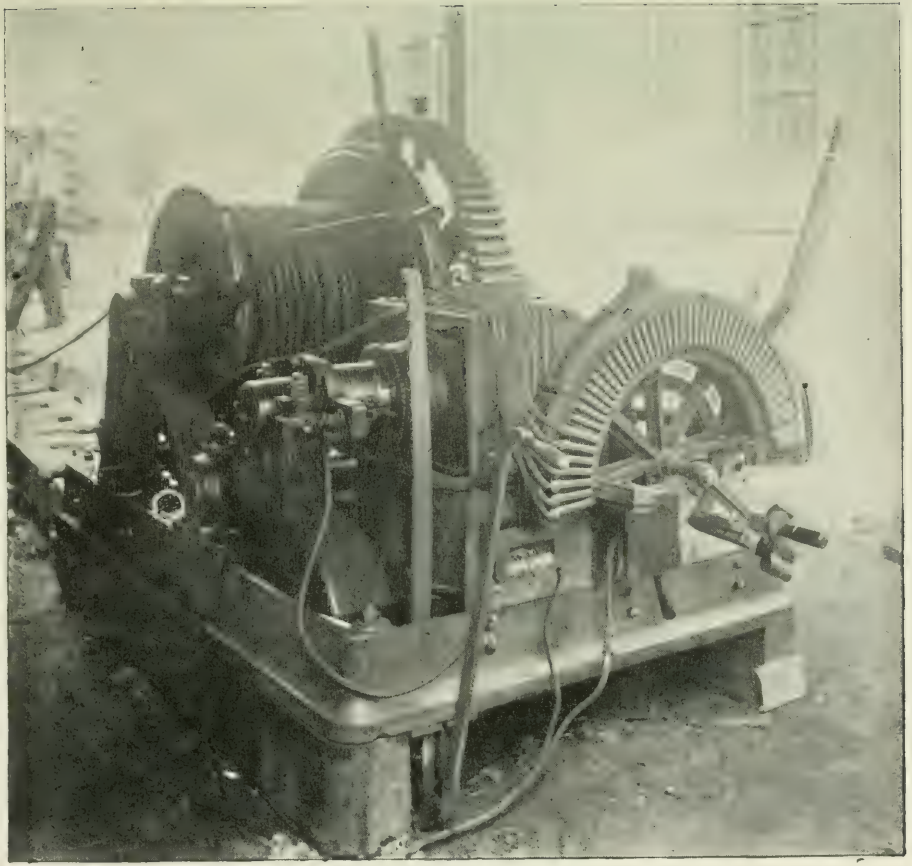


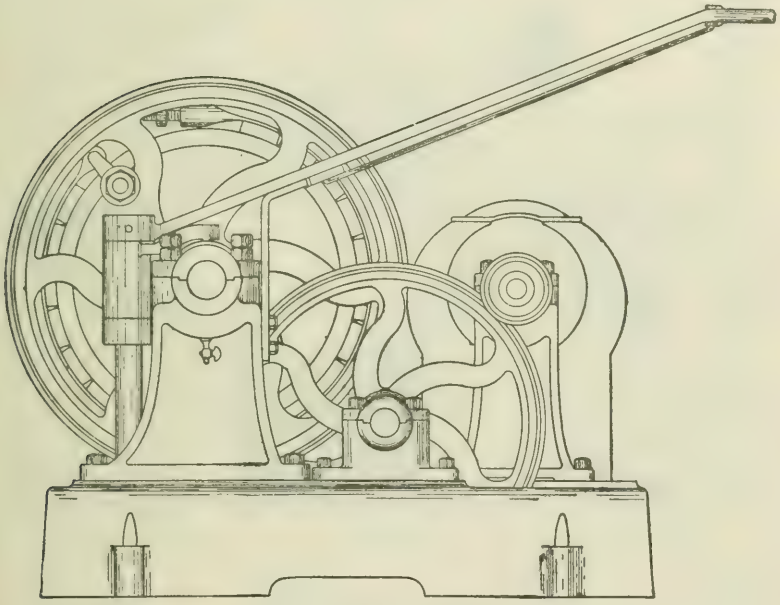
Fig. 9.—ELECTRIC HOIST OF WEBSTER, CAMP AND LANE

which (although of the same general type) will be so modified in design as to stand only 4 feet in the clear, above rails, the gauge being 3 feet.

The same general plan is applicable to mine-tramways, of even 12 or 18 inches gauge, suitable for narrow entries, such as are found in some of the western gold- and silver-mines.

For hoisting-purposes, we may have a variety of forms and sizes, from a 3-horse whip, for hoisting small buckets, to a machine capable of operating a full-sized cage under its maximum load.

FIG. 10.



Electric Hoist, Side View.

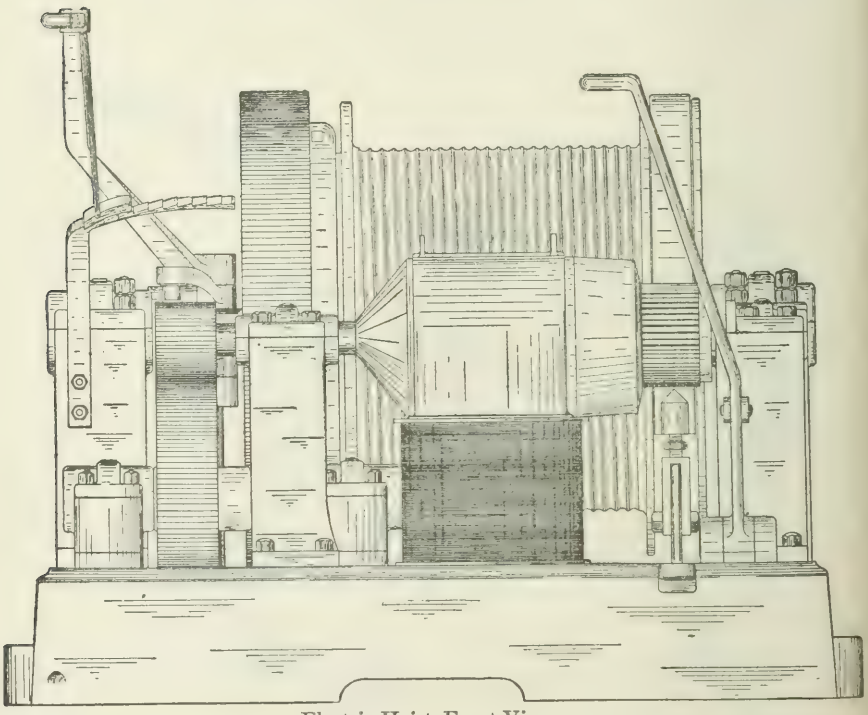
Fig. 9 shows a hoist of the Webster, Camp & Lane make, equipped with a motor, for use in the Ashland iron-mine, at Ironwood, Michigan, while Figs. 10 and 11 show different views of a line of electric hoists recently put upon the market for miscellaneous uses and ranging from 10- to 80-horse-power.

In the Castle-Gate mine, Utah, a 35-horse-power Lidgerwood Thomson-Houston hoist is located about 1000 feet inside the main entry, for handling the trains at that point.

In pumping-machinery, a peculiar apathy seems to exist among

prominent manufacturers as to the demands of this kind of work. There are, of course, innumerable cases where centrifugal and plunger-pumps have been belted or geared to motors to accomplish certain objects; but the pump shown in Fig. 12, is the first of considerable size which, to the writer's knowledge, has been designed, with special reference to operation by electric power, and which is, in itself, a mechanical unit, so to speak.

FIG. 11.



Electric Hoist, Front View.

This pump is manufactured by the Gould Manufacturing Company, of Seneca Falls, N. Y., and the table of tests given below, shows its efficiency when operated by a 15-horse-power motor of special winding. These tests were made, with a view to determining the general fitness of the type for deep-mining pressures, and the results have been considered sufficiently satisfactory to warrant the designing of a standard line of pumps of from 50 to 500 gallons per minute capacity.

Tests on Gould Pump, July 31, 1890.

Pressure, lbs. Per sq. inch.	Gallons per min.....	100	125	150	175	200	225	250
0	Electrical H. P.....	1.2	1.5	1.8	2.2	2.7	3.3	3.9
	Mechanical H. P...	.12	.16	.18	.21	.23	.26	.26
	Efficiency.....	.10	.10	.10	.10	.071	.071	.072
20	Electrical H. P.....	2.91	3.67	4.16	5.06	5.87	6.56	.....
	Mechanical H. P...	1.38	1.74	2.09	2.42	2.83	3.10	.....
	Efficiency.....	.48	.47	.50	.48	.48	.47	.....
50	Electrical H. P.....	5.16	6.52	7.75	8.98	.....	.....	.....
	Mechanical H. P...	3.11	3.95	4.75	5.52	.....	.....	.....
	Efficiency.....	.602	.606	.612	.615	.....	.....	.....
75	Electrical H. P.....	7.30	8.64	10.48	.....	.....	.....	.....
	Mechanical H. P...	4.69	5.79	7.06	.....	.....	.....	.....
	Efficiency.....	.643	.670	.674	.....	.....	.....	.....
100	Electrical H. P.....	9.21	11.54	12.25	Speed of Pump, 27 revolutions.			
	Mechanical H. P...	6.16	7.77	8.36	.....			
	Efficiency.....	.670	.673	.682	.....			
125	Electrical H. P.....	11.29	13.88	.....				
	Mechanical H. P...	7.74	9.37	.....				
	Efficiency.....	.686	.675	.....				
150	Electrical H. P.....	14.02	15.16	Speed of Pump, 22 revolutions.				
	Mechanical H. P...	9.17	10.19	.....				
	Efficiency.....	.654	.672	.....				

The Efficiency, in this Table, is the Mechanical, divided by the Electrical, Horse-Power.

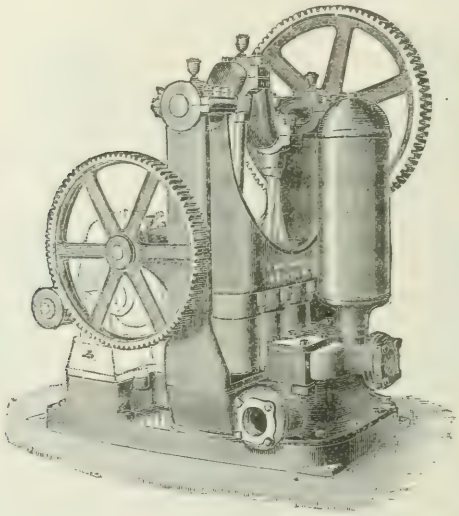
The pump as shown, consists of three vertical cylinders, within which are three single-acting plungers, their cranks being hung from the main shaft at 120° angles, in order to produce the most even application of power. In addition to this class of pumps for general hydraulic work, the Van Depoele type of reciprocating-engine is being adapted to a sinking-pump which has, as yet, not been sufficiently tested to warrant further mention here.

I need hardly mention the subject of ventilating-apparatus, as the application of motors to revolving-fans of any kind is a mere question of belts or gears, though the fact that the fan, with its motor, may be located at any desired point within the mine, with an expenditure of power hardly greater than would be necessary

outside, has a marked bearing upon the general arrangement and efficiency of ventilating-systems.

The question of lighting, also, though a most interesting one to the engineer and operator, hardly falls within the scope of this paper. It is sufficient to call attention to the fact that the same wires which furnish current for a part or all of the apparatus enumerated above, will make the interior of the mines as light as day, and give a beneficial result, not only directly in the amount of product, but indirectly in the satisfaction and comfort of the men. This

FIG. 12.



Gould Electric Pump.

is not theory, but has been demonstrated to the satisfaction of all concerned, and in cases where there was, at the start, among the miners themselves, a hearty and bitter opposition to the new system.

Before considering the question of surface-arrangements, let us pause a moment, to answer the oft-repeated question as to danger.

The danger of accident from any class of electric machinery in mining-operations, is of three kinds: first, physical injury from contact with conductors or apparatus, through which a current is passing; secondly, danger of fire or explosion from a spark in some part of the electrical apparatus; and, thirdly, failure of the apparatus to work properly at critical moments. Regarding the first point, it may be said that a limit of 250 volts has been fixed by many experts in mining and electrical engineering for all apparatus to be

used below the surface ; and that *permanent injury* from a current of this tension, *regardless of its amount*, is *absolutely impossible*, the only liability to danger lying in secondary and indirect accidents, owing to the surprise and momentary discomfort due to an accidental shock. A hot steam-pipe carries more danger for the man who unexpectedly touches it than a bare copper wire carrying enough current to cut and haul two thousand tons of coal a day, at the voltage named above. As to the second source of danger, no electrical apparatus, if properly installed, should ever spark, at known and properly-guarded points; the amount of protection at these points being entirely dependent upon the knowledge of the designer, faithfulness of the operator and common sense of the purchaser, in spending sufficient money on safeguards. As to the third point, no class of apparatus ever designed is free from the possibility of accidental derangement, and at critical moments. But of electrical apparatus, it may be said that no type of machinery will answer more quickly to sudden demands upon its capacity, or give a plainer indication of continued abuse ; while, on the other hand, no system of power-transmission is capable of more rapid repair in case of temporary derangement. We have but to compare the work of mending a broken air- or steam-pipe, or splicing a parted cable, with that of making a simple twist in a broken wire, and we need not wonder at the recent prophecy of a prominent naval officer, that coming war-ships would carry no steam outside their boiler- and engine room.*

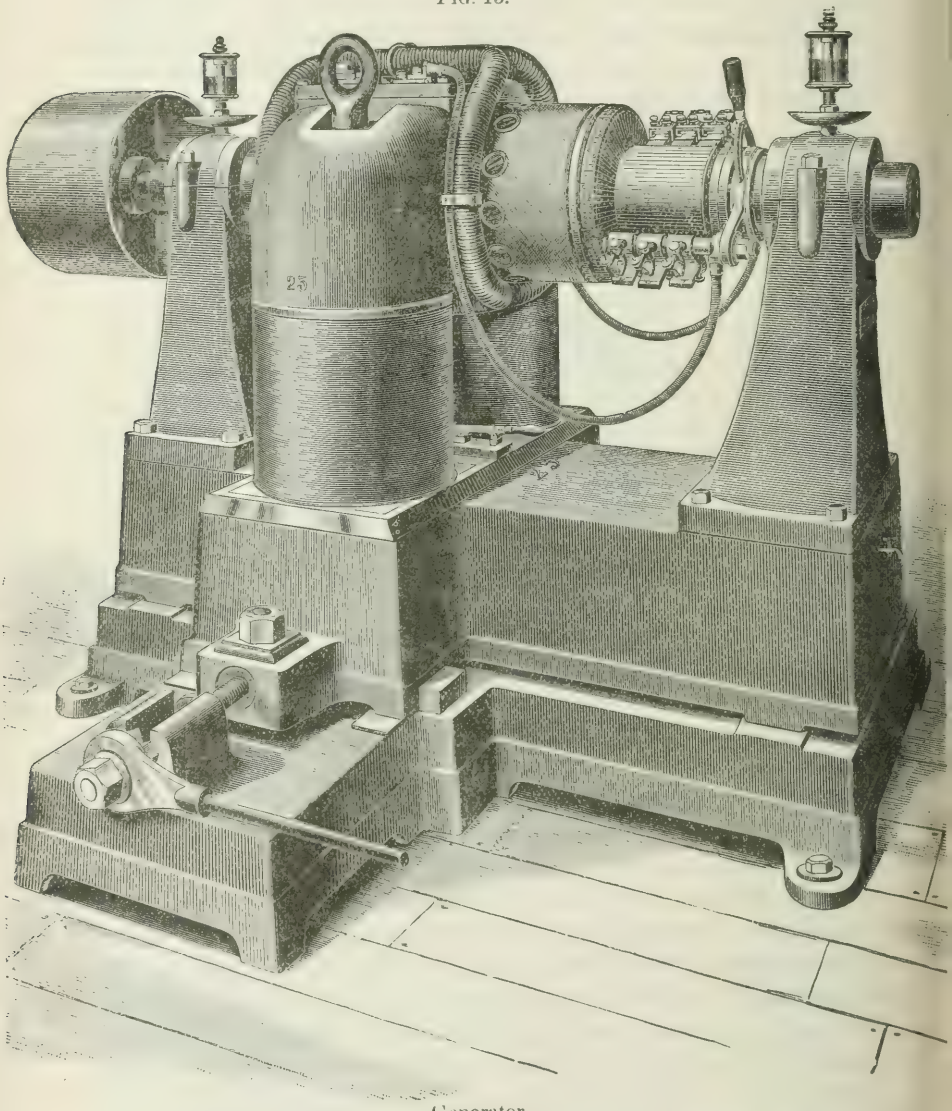
Let us consider now the demands of the generating-system which will enable us to furnish the proper current to our apparatus to the greatest advantage.

If we have an abundant and steady water-power at hand, so much the better, as the expense of maintenance then becomes practically a matter of interest on investment. If, on the other hand, we must rely on steam, let us be sure and have an ample boiler-capacity to begin with. Many an electric installation has been condemned by casual observers when the fault lay between the coal-pile and the dynamo-belt. It is, however, the province of this paper to consider, not the generation of mechanical energy, but its transmission ; and, accordingly, attention will be given first to the current-generators.

* Some of our latest additions to the navy have more than twenty steam-engines for hoists, pumps, etc., in different locations, any one of which might be rendered useless by a stray shot, not to mention the demoralizing effect of escaping steam, while the wires of electric apparatus for the same purposes may be almost instantly repaired.

Fig. 13 shows a generator of 85 horse-power capacity, suited for mining work, manufactured by the Thomson-Houston Electric Com-

FIG. 13.



Generator.

pany. It is equipped with sliding base, and self-oiling bearings, and weighs 10,000 pounds.

Mounted on brick or stone foundations, in a dry, cool room, free

from flying dust of any kind, this, with A 1 oil, and *cleanliness*, should need practically no attention except at starting and stopping.

Cheap machines will not answer for mining-work. One cannot shut down for a few hours whenever it is desirable to have a new commutator put on, or a little shellac and braid here and there; and a thousand dollars more on the original investment is a good deal better than six or seven hundred a year for repairs.

Plates II., III. and IV. show, respectively, the front and end elevations and floor-plan of a power-plant just erected by the Pleasant Valley Coal Company, at Castle Gate, Utah, which is, in many respects a model of its kind.

It has a capacity of about 750 horse-power of generating machinery, which will be used for furnishing current to electric coal-cutters, drills, pumps, haulage-engines, ventilating-fans, etc., the entire power needed for the various mining operations being transmitted electrically. As this is the first plant of such magnitude employing electric apparatus to the exclusion of other systems, the result will be watched with interest.

I append some communications kindly furnished by gentlemen who have given special attention to various branches of the subject of this paper. To these, as well as to others who have furnished various data of interest, I wish to express my thanks.

The Electric-Magnetic Reciprocating Engine.

By Charles J. Van Depoele.

One of the recent developments in the electric line is the perfection of a new electric-magnetic reciprocating engine.

For years past, and, indeed, ever since electricity was first used to magnetize an iron bar by means of a coil of wire, the idea of producing a reciprocating motion by the electric current has been entertained, and many unsuccessful attempts have been made to construct and operate such machines.

It is a well-known fact that when a current is made to flow through a coil of copper wire, or what may be termed a solenoid, a bar of iron placed near the ends of such a coil will be sucked into the solenoid, and the moment the current is broken the plunger or iron bar will be dropped by its own weight, or may be expelled by means of springs if the machine is in any other position than vertical. This principle, however beautiful, could never be successfully applied in the construction of heavy machinery on account of the make and

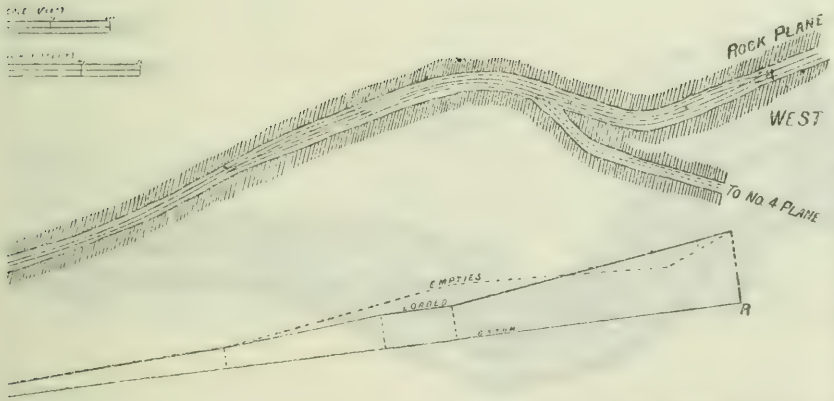
break of the circuit, which has to take place in order to produce the pulsations of the current which causes the pulling and letting go of the plunger.

Not until the invention and construction by the writer of an electric generator which would give currents rising and falling at a definite speed, could the current be sent to the coils of a reciprocating engine, there alternately attracting and repelling its plunger. In this apparatus, however, as will be seen later on, the rising and falling of the current is produced in such a way that it is absolutely certain that no spark is produced either in the machine or in the generator producing the current. The current is caused to rise and fall in closed circuits, and actuates the plunger of the reciprocating-engine with a speed corresponding absolutely to the speed of the defined rise and fall of currents produced by the generator. Thus, the generator can be caused to produce, say 400 pulsations of current a minute, or it can be made to produce either a higher or lower number than this, so that the engine's speed can be regulated according to the size of its piston and the work it has to do.

The engine itself is a simple piece of machinery, consisting of two or more coils of copper wire, or solenoids, incased in an iron envelope protecting them from outside injury. Within these coils is placed a brass tube, and within this an iron plunger, capable of moving to and fro under the action of the currents in the coils. To the end of this iron plunger is attached a piston-rod, similar to that of an ordinary engine, and to this is attached the hammer, drill or whatever tool is to be operated by the engine.

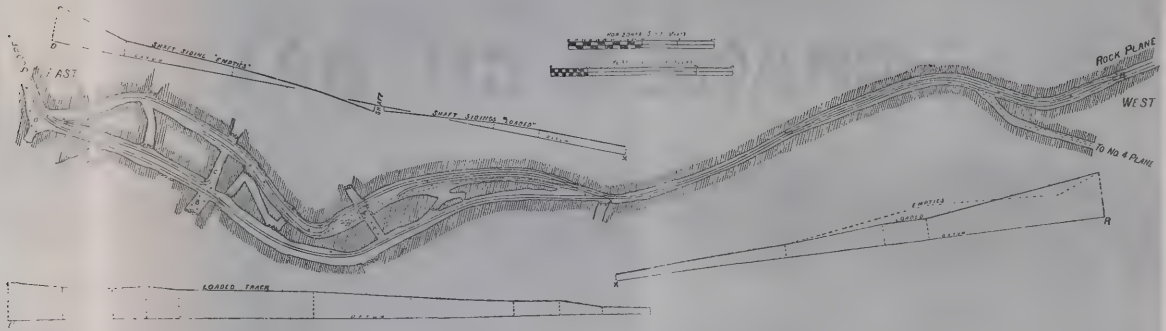
A very simple means of connecting the reciprocating-engine to the current, is applied to the machine, so that it can be stopped and started instantly. The cables leading from the generating-station to where the current is to be used, are perfectly insulated in the usual manner, and switch-boxes are disposed along the main cables. From these, flexible cables are led to the reciprocating-machine, so that it can be moved, if necessary, from one place to another, without any trouble whatever. The cable is incased in a rubber tube to protect it from injury and to prevent loss of current by contact with the ground.

It will readily be seen that one of the first applications of these machines will be to rock-drills, such as are now in use in nearly all mining and rock-work, for quarrying, prospecting, etc. The simplicity of the machine lends itself most readily to this application, as it will be seen that there are no movable parts on the whole

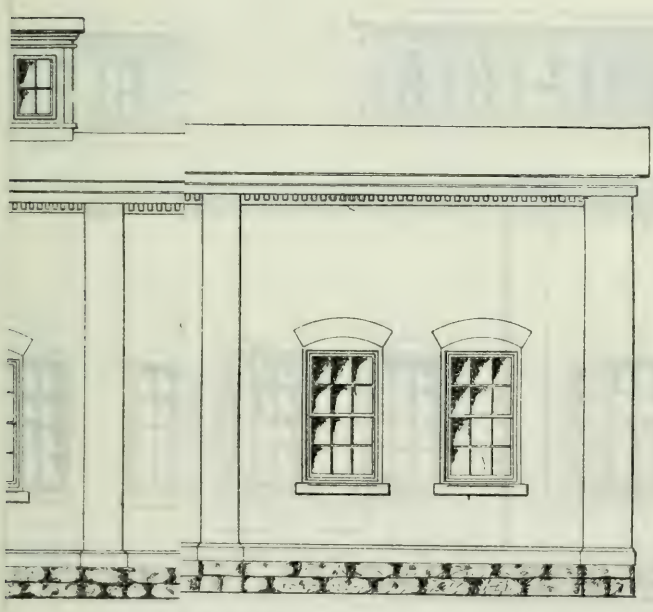


ıide Colliery, Pa.

Fig. 8.



Plan and Profiles of Haulage at Hillside Colliery, Pa.



LEY COAL CO. —
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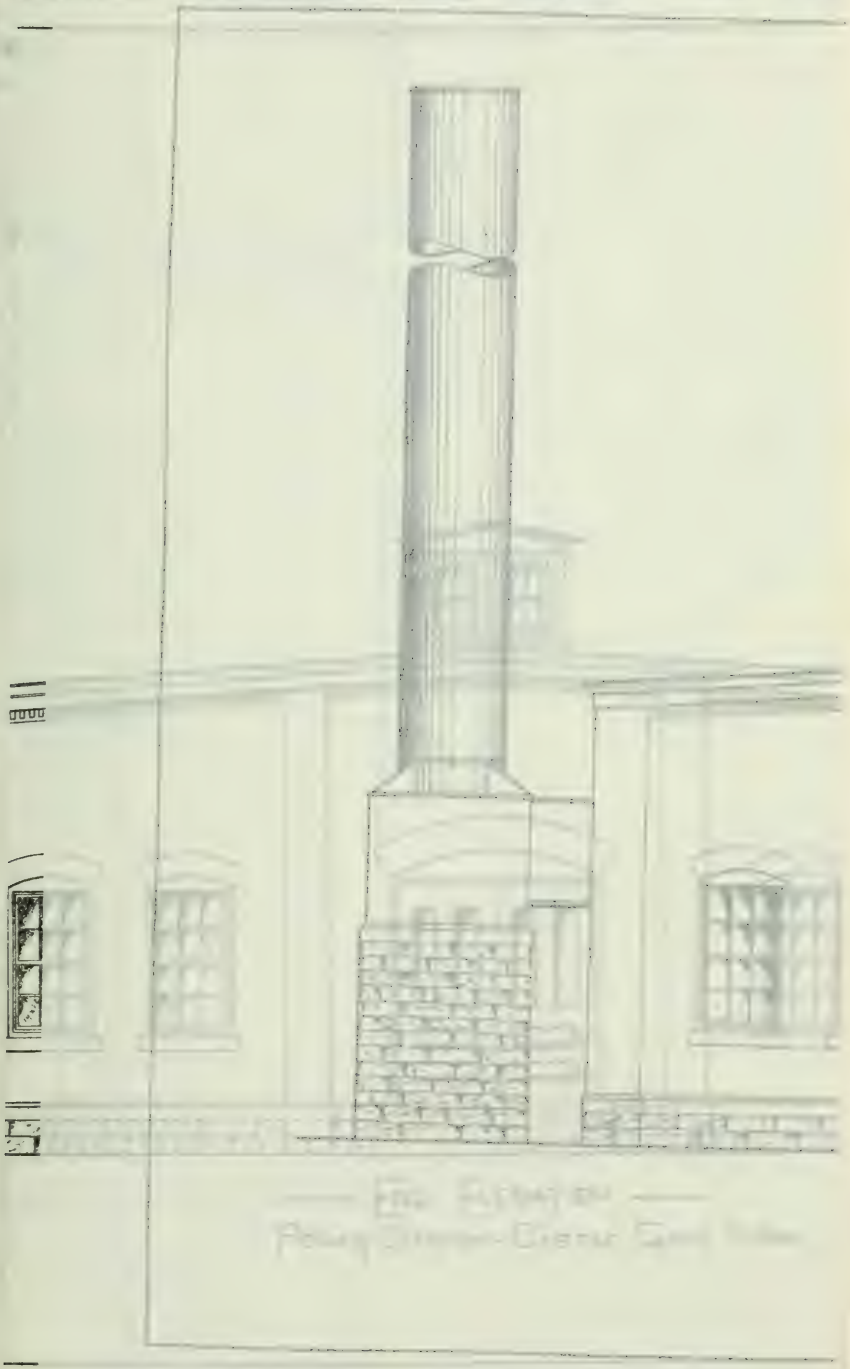
— FRONT ELEVATION —
POWER STATION — CASTLE GATE MINE.

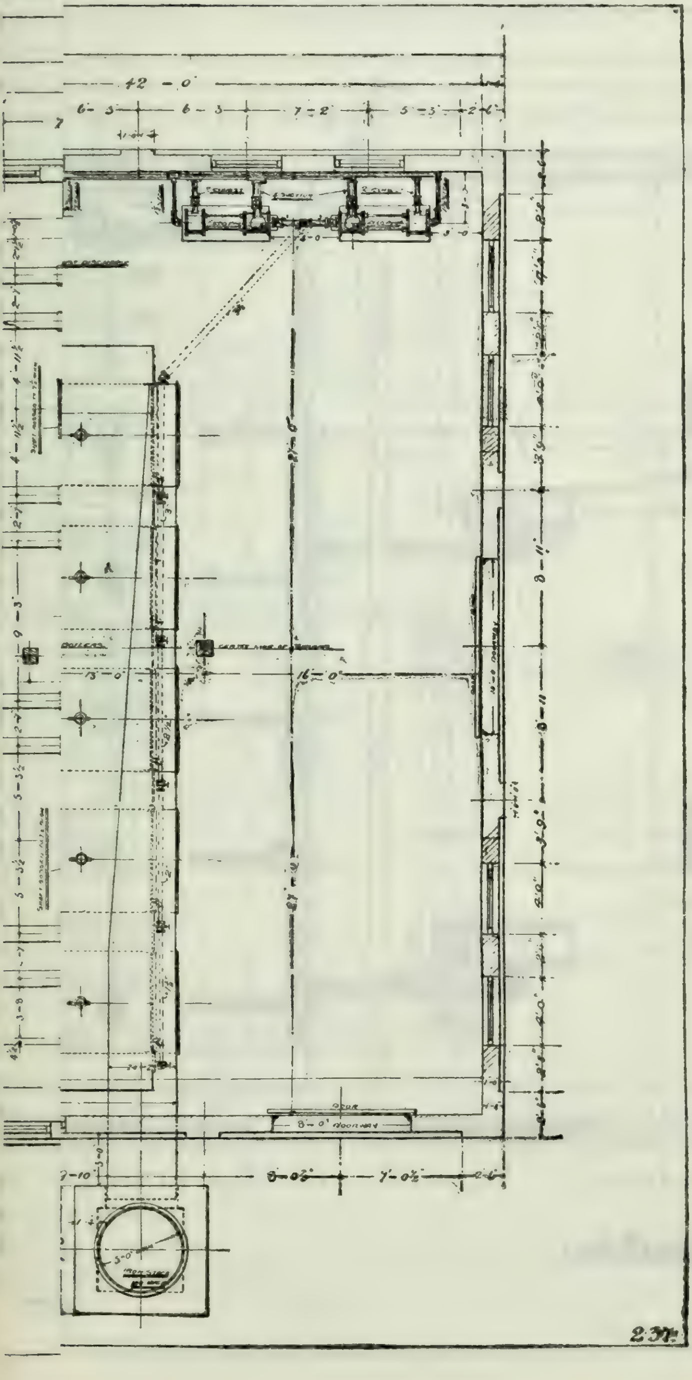
— PLEASANT VALLEY COAL CO —
SALT LAKE CITY, UTAH.

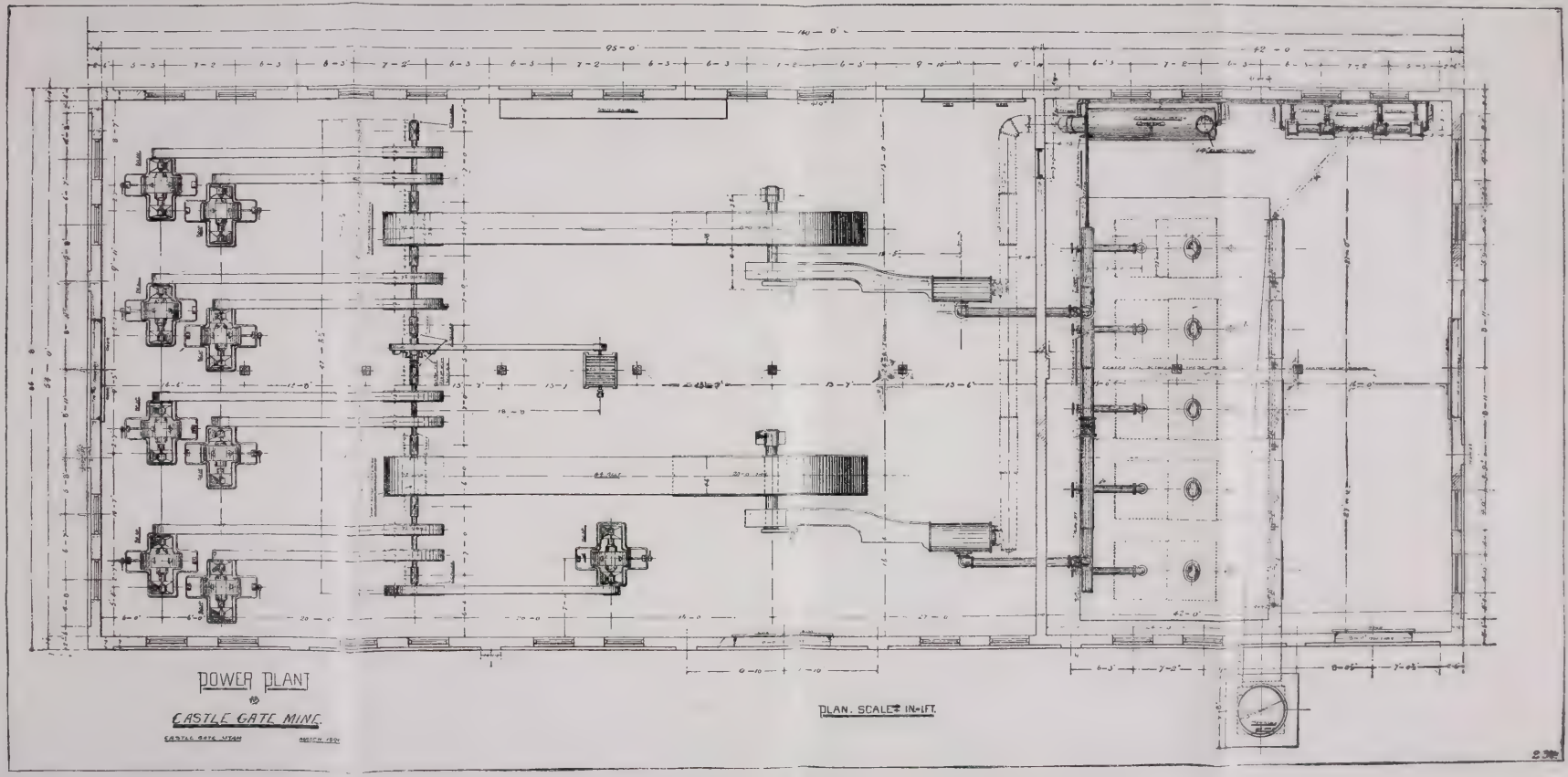


END ELEVATION
POWER STATION - CASTLE GATE MINE

ate

SPAULDING.—Plate IV.




POWER PLANT
OF
CASTLE GATE MINE.
CASTLE GATE, UTAH
PLAN. SCALE 1" = 1 FT.

machine, except the plunger and piston-rod carrying the tool. There are no valves, as in steam-engines, no switches, no make and break of the circuit, and no exposed current-carrying parts, so that the whole can be handled with safety and without any skill above that of common laborers. By turning on the switch the machine is started; by turning off the current, it is stopped. There is absolutely nothing to be done to the machine, except occasionally to pour in a few drops of oil, to lubricate the piston and its rod.

For ordinary mining-work the drill is mounted on a tripod similar to that now in use with the steam- and air-drills which are well known to-day, or it can be attached to horizontal bars or to vertical columns; in fact, it lends itself to absolutely the same work as has been done heretofore by the air- and steam-drills. It will work in any position, from horizontal to vertical. The weight of these drills will be approximately the same as that of the steam- or air-drills of the same capacity, and everything is so arranged that the men accustomed to work the latter machines will find no difficulty in operating the new electric drills.

The apparatus for generating currents to operate these drills or electric-magnetic engines, can be placed at any distance from where the drills are at work, and any number of drills can be worked from the same source, each drill working independently of the other, and whether one or more are in operation, the generator will regulate and furnish a current exactly proportional to the demand.

Where a long distance exists between the generator and the place where the current is to be distributed, a system of conversion is used; that is, the current is changed from a high to a low potential, so that the primary current, transmitting the power from the prime motor, can be of a high tension. Since this current could be guarded against possible contact, there would be no danger in using high voltage. At the point of application, the high potential is converted to a low potential of such pressure only as may be found desirable and practicable. Where the distance is, say, only 1 or 2 miles, it will not be necessary to use any system of conversion whatever, as a current of suitable potential can be run directly from the generator to the mines.

The main advantage, however, is the superior economy over the motors now in use where steam or air is to be conveyed to a distance of only a few hundred feet. There is an enormous loss in this transmission, and methods of steam- or air-transmission are certainly very limited in scope. There is, also, the trouble of leakage, making

a system of piping much more costly in maintenance; but this is done away with when electricity is employed. All parts are so simple, that they will require little or no repair for very long terms; and such repairs as will have to be made, will always be much less expensive than in the case of steam- or air-driven machines.

The Practical Application of Electricity to Coal-mines.

By J. S. Doe.

It is now only about fourteen months since the Jeffrey electrical coal-mining machine made the first really practical test of electricity in the mining of coal on record in this country. This was carried out in April, 1889, in the mines of the Shawnee and Iron Point Coal and Iron Company, at Shawnee, Perry County, Ohio, in what is known as the Big Vein coal-field of Hocking Valley. This company had previously been experimenting upon electrical haulage, and, to this end, had put in a power-plant consisting of a 40 horse-power generator, built by Foree Bain, of Chicago, Ill., with an exciter, both being belted direct to a Beck high-speed engine of 60 horse-power. The terminal potential of the generator is 260 volts, and that required by the mining machine motors is 220 volts; therefore, there is a loss of 40 volts in the line. In the course of the experiments at the Shawnee mines, the Jeffrey Manufacturing Company, which had been manufacturing for the past ten years a coal-mining machine run by compressed air, was called upon to assist the coal company in the installation of its plant, and, in connection with that work, made and put in for trial the first electrical mining-machine. The same machine, with only a few minor changes, has been in practical daily operation in the mines ever since; and during the same period the Jeffrey Manufacturing Company has installed or contracted for 23 electrical mining-machines, as follows:

The Thurmond Coal Company, Thurmond, W. Va., four machines, with two 80 horse-power generators, and two 85 horse-power high-speed engines.

The Sterling Mining Company, Cannelton, Pa., two machines, with one 80 horse-power generator, and one 85 horse-power engine.

The Monongah Coal and Coke Company, Camdenburg, W. Va., five machines with three generators and three engines. This company will add more machines as fast as territory is opened to permit their use.

The Consumers' Coal Mining Company, Spilman, W. Va., two machines, one 60 horse-power generator, and one engine.

The Ellsworth and Morris Coal Company, Jobs, Ohio, four machines, with two generators, and one 150 horse-power engine. This company has also given an order for another plant of four machines.

The Upson Coal Company, Shawnee, Ohio, is now putting in a plant to consist of two or more machines.

All the foregoing plants have sufficient additional power to run from one to four more machines when required. A number of other orders, calling for many machines, are under negotiation at this time, and several of the above companies have contracted for additional machines, after a practical trial of their first ones, thereby testifying to their satisfaction with the use of electric ty in coal-mining. The distance from the power-house to the working-places in the mines, varies in the above instances from 1000 to upwards of 5000 feet.

As already observed, the motors on the mining-machines are wound for 220 volts, with a capacity of 15 horse-power. This voltage has been found to be perfectly safe, there being no danger from any accidental contact with the wire. The amount of work done by each machine averages from 600 to 900 square feet of surface under-cut in 10 hours. It requires two men to handle the machine. The machines cut into the coal or fire-clay bottom, as may be desired, at the rate of about 1 to 1½ feet per minute; the standard machine under-cutting 5 or 6 feet deep, 39 or 42 inches wide and 4 inches high. After the cut is made, the cutter-bar is withdrawn by throwing a lever; the machine is then moved over the length of the cutter-bar for another cut, and proceeds as before. Upon completion of the room, the machine is loaded upon a truck provided for the purpose, and moved into another room for more cutting; the first room being then drilled and the coal shot down for loading.

The same power, electricity, has been applied to the running of electrical drills, manufactured by the Jeffrey Company. These drills penetrate the coal at the rate of two or more feet per minute. The mines are also, in some instances, lighted by incandescent electric lamps. Electricity is also applied to the running of pumps and fans in some of the above-named coal-mines, with perfectly satisfactory results. There can be no doubt that electricity in coal-mines has become a permanent, practical success. We are, however, only in the infancy of its uses. There will be many and valuable improvements made as time progresses; but, with the light we have, it is evident that considerable advantages will accrue to those who first put in electrical coal-mining machinery. The saving in the

cost of mining coal by machinery, as compared with hand-work, varies considerably in different localities and in different sized veins; but it may be roughly estimated to average about 20 to 25 per cent., with an additional saving of merchantable product, since machine-mined averages coarser than hand-mined coal.

A few words may be added on the use of electricity in the hauling of coal. It is unnecessary, at this late day, to call attention to electricity as a motive power in the transportation of heavy and quick-moving loads. Our many street-railway lines give us daily and hourly demonstration of that. The Jeffrey Manufacturing Company is building and has in practical operation electric mine motor-cars of 20 horse-power capacity, each weighing 5 tons; the same type of motor, with the same voltage, being used as on the mining-machines. The frame of the motor-car is made of cast-iron, with heavy cast-iron draw-bars, and pilots to clear the track of any obstruction, such as coal, etc. The wheels, 20 inches in diameter have steel tires with steel axles, power being transmitted from the motor to the axles through straight pinions and gears. The speed of the motor-car is 8 miles per hour. The electrical power is transmitted to the motor by means of a 4-wheeled trolley running on a double all-metal line, placed along the side of the entry, there being no rail- or ground-return. Sand-boxes and electric head-lights are placed upon the motor. The "motorneer" sits at one end, and handles and guides his iron horse, fed from nature's storehouse, with far more comfort and ease than can be had with the stubborn mule.

The Jeffrey motor-car has hardly as yet demonstrated its maximum power. We cannot say how much coal it would haul on a level. It has hauled as high as 50 tons in one train, up and over a 4½ per cent. grade. The Shawnee and Iron Point Coal and Iron Company have one of these cars, and the Thurmond Coal Company another, while the Upson Coal Company is putting in the plant for one.

Accumulators and Mining.

By Francis A. Pocock.

At the Washington meeting, in February, 1890, Mr. Pedro G. Salom read a paper on "Electrical Accumulators, or Storage Batteries."

In the following paper, it will be my object to investigate the question, as far as possible, from a purely mining point of view. That the use of accumulators in mines is not far off, is almost certain; and it will be of interest to many of the members to see how far this reservoir of power will bear filling and drawing upon at the

present time, and what the relative cost of the two electrical systems may be expected to be, not so much in first outlay as in the running expenses of the plant.

The first thing to be decided is the weight of these accumulators, and the easiest way to define this, is the weight per horse-power. Mr. Salom says, it takes 25 pounds of battery to give 1 horse-power-hour, and that to give 100 horse-power-hours, or 10 horse-power for 10 hours, requires 5500 pounds of battery, or 220 elements; but 25 pounds is the net weight and 32 pounds the real total upon which we must base our calculations, so that we have a total of 7040 pounds as the weight of this battery. The next question is, How much room will this take upon a mine-locomotive? A mine-locomotive of 10 horse-power should not be more than 9 feet long, and 2 feet of this will be taken up by the bumpers, leaving 7 feet in length for the battery. Then for a 3-foot track it should not be more than 5 feet wide. Allowing 1 inch all round a cell, it will be possible to set ninety-six of these on the floor-space of the locomotive; but we must have fourteen more than this, which will make the width 68 inches. The height of this cell is 8 inches; and, allowing 2 inches space above it, and $1\frac{1}{2}$ inches of plank, the top of the second tier will be $19\frac{1}{2}$ inches high above the floor.

If the motor is to be placed below this floor (and there is no other place for it), then the bottom of the floor will be 2.6 inches from the track; and, allowing the floor to be 3 inches thick, to stand the weight, the top of this car will be 4 feet $4\frac{1}{2}$ inches above the track. Remembering that the height of the 40 horse-power motor at Lykens Valley is only 4 feet high, and the one at Erie Colliery, also 40 horse-power, is 4 feet 4 inches high, and that they are both narrower and of the same length as the proposed storage-motor, and of four times the power, there is certainly one point established against the present use of accumulators.

Table of Weight, etc., of Electric Mine-Locomotives.

Location.	Horse-Power of Motor.	Weight of Locomotive. Pounds.	Largest Load. Tons.	Speed. Miles Per Hour.
Zankerode,	4.5	3000	$1^{3}\frac{1}{2}$	6
Paulus,	5 to 6	4200		6
Lykens,	40	12,000	165	6.8
"	40	12,000	150	6.8
Shawnee,		4500	21	5
Buckingham,		7000	60	8
"		4000	30	8
Bear Run,	60	18,000	150	6
Erie,	40	13,500	107	6

Returning to the subject of weight, we calculate that, to that of the accumulators, 7040 pounds, must be added that of the motor, say 1300 pounds, the wheels and axles, say 1100 pounds, and the frame of the machine, say, for strength alone, 1400 pounds, giving the total weight of motor, etc., as 10,840 pounds.

From the above, it appears that the small German motors weigh about 700 pounds per horse-power, and that the large American motors only weigh 300 pounds per horse-power, whereas the accumulator-motor would weigh at least 1000 pounds per horse-power. It is true that weight is necessary to traction, but it is also true that unnecessary weight will entail loss of power, and from the very limited information at hand it appears that 700 pounds per horse-power is found to work satisfactorily with small motors, and that the weight per horse-power decreases as the horse-power increases. Consequently, for a 10 horse-power motor, 1000 pounds per horse-power appears to be too high. There will be a waste of power in moving this weight and, if we wished to operate a 40 horse-power locomotive with batteries, the number of cells would be 880, and their weight 28,160 pounds. They might be divided into two batteries and towed in a tender, but even then each would weigh 14,080 pounds and this would absorb at least 500 pounds of the total pull of the motor, when running on the level, and considerably more where grades are to be overcome.

Dr. Lewis Bell has pointed out that, whereas a good roadbed is necessary for electrical traction by overhead wire, it is even more imperatively so when storage-batteries are used. We have found in practice that a 25 pound-per-yard steel rail is too light for a locomotive of 13,000 pounds weight to be run upon; therefore, there is no saving to be looked for in this direction by the use of accumulators. On the contrary, when we begin to put in a roadbed heavy enough to stand the weight of a locomotive weighing, say, 23,000 pounds, there are many disadvantages that the colliery-manager will be the first to see. Besides the first cost of the track, the keeping of this weight of track in good repair, in a mine where the floor is forever moving (as it is in many of our mines), would be a work of no slight expense in itself. It appears, therefore, that not much is to be expected from accumulators as a means of haulage.

However, this is one side of the question only. There is another side which should be considered, and that is the use of the accumulator-motor in collecting the cars to a point where the heavy haulage-motor can reach them. The prospect, as viewed from this point, is

more encouraging. From what I have seen of the work, a motor, of about 5 horse-power would do the work of three or four mules, and the weight would be about 5000 pounds, as follows:

	Pounds.
Accumulators (110 cells),	3500
Motor,	600
Wheels and frame,	1000
Total weight,	5100

This machine could be built low, so as to take up little height. It is only possible to make this form a commercial success, in my opinion, when the gangways are low and the roof would have to be cut to gain height enough for mules to work. Then, the cost of the cutting saved by the use of the motor would counterbalance the repairs on the battery and the extra care and expense in laying and keeping the track in order. The cost of the system may be estimated as follows—assuming the first cost of the locomotive complete to be $2300, and allowing that the mine can afford to charge these cells for $40 per horse-power per year, the generator and engine being already installed and doing work during the day-time, and this sum representing fuel and interest on machinery. The attendance should not be more than $300 per year, as the pump-man and night engineer can do the work:

Estimate of Expense of 5 Horse-power Accumulator-Motor.

Interest, at 6 per cent., on $2300.00,	$138.00
Repairs to battery,	500.00
Repairs to motor, etc.,	150.00
Cost of power, at $40.00 per horse-power,	200.00
Attendance in charging at night,	300.00
Engineer, at $2.00 per day, for 260 days,	520.00
	$1808.00

The cost of running three mules and drivers would be about as follows:

Interest and depreciation (26 per cent.), on $450.00,	$117.00
Feed, shoeing, harness and attention, at $0.33 per day,	361.00
Three drivers, for 260 days, at $2.00 per day,	1560.00
Total,	$2038.00
Less,	1808.00
Annual saving on 3 mules,	$230.00
Or, for 4 mules: total expense,	$2717.00
Less,	1808.00
Annual saving on 4 mules,	$909.00

Or, about 44 per cent. on the investment, under the circumstances assumed.

The rail and track being an important item in the economy of this method, I think that perhaps the cheapest, and at the same time the best method would be to use from 20- to 25-pound steel rail, and, in laying the track, to place the ties first about 3 feet apart from center to center, and on these, and under each rail, to place a string-piece of wood 1½ by 3 inches, nailed to the ties, and spike the rails on the top, keeping the stringer-joints and the rail-joints from coinciding. The combination makes a solid track, and a very smooth-running one, and it has the advantage of not lifting easily into very uneven points; the ends of the rails do not jump as when only laid on the ties, and the track has not the spring to it which is so injurious.

The true place for accumulators at present is in lighting, and it looks as if they could be used to advantage in this connection. They are very heavy, it is true; but the lighting-arrangement for eight hours' work would not be a large or very heavy affair, and could be taken to the working-place on the first car, and brought out on the last, so that it would not have to be carried by hand at all. The advantage to the operator would be material, as the men can work better in good light than in poor, and the coal would come faster and cleaner from a well-lighted place. The lighting of switches, turnouts, etc., could be easily accomplished in all parts of the mine, and the jar which would be detrimental to traction would not occur in this case. The charging could be done at night, and a clear steady light delivered during the working hours.

The Use of Electric Power-Transmission at Aspen, Colorado.

By C. E. Doolittle.

At the request of Mr. Spaulding, I give a brief statement in regard to the plant in use at this place by means of which water-power is utilized for generating electric currents and the latter are applied to mine-hoisting, etc.

This mining-town had been lighted by electricity for three years, the power being obtained from several small water-powers, when owing to the limited quantity of water flowing in the mountain streams of this region in winter, it became necessary either to put

in a steam-plant or to use water under a much greater head. It was determined to adhere to water-power; and the construction of a plant, to use water under a head of 900 feet, was begun eighteen months ago.

A dam was built on Hunter Creek, about three miles from town, at a point where a dam 12 feet in height creates a reservoir covering several acres of ground. It has been necessary, during a few of the coldest winter nights, to draw upon the supply of water in this reservoir, but the reservoir always refilled itself during the following day.

From the dam, the water is carried some two miles in a wooden flume, buried in the ground, to the head of the pipe-line. The pipe-line is 4500 feet long, and has a fall of 876 feet, giving a pressure of 380 pounds per square inch at the water-wheels. Lap-welded pipe, 14 inches in diameter and about $\frac{1}{4}$ inch in thickness, is used.

In the power-house there are eight Pelton water-wheels, each two feet in diameter, and each capable of developing 150 horse-power. The water-wheels run at the rate of 1100 revolutions per minute, and each wheel is belted directly to one or more dynamos. This power has now been in use more than a year and has proved absolutely reliable. The wheels at present furnish power to operate three arc-dynamos, sixty lights each; four incandescent dynamos, supplying 2500 incandescent lamps, and two 500 volt-power dynamos, one requiring about 60 horse-power, the other about 120 horse-power under full load. These power-generators furnish current to electric motors used principally for hoisting in mine-inclines. For this purpose, motors of the street-car type are used, so that the speed is under control of the engineer. The motors are connected to ordinary mine-hoists by spur-gearing and paper friction-wheels. These electric hoists are all underground, at points where it would be both difficult and expensive to obtain power in any other way. The distance of the motors from the dynamos is, in most cases, about 2 miles. Some of the hoists have been running more than two years, and all have proved satisfactory and reliable. The expense for motor-repairs has been very small. There are also motors in use for running blowers, rock-crushers, etc. The motors range in size from 3 to 75 horse-power.

An electric tramway for mine use is now in process of construction. The locomotive for this purpose consists of a street-car mo-

tor, mounted on a truck, with the armature-shaft at right angles to the axles. This arrangement involves the use of a set of bevel-gears. There should be a special motor for this purpose, so designed that the locomotive will not be too broad to run on the 18-inch-gauge tracks when the armature-shaft is placed parallel to the axles, thus allowing spur-gearing to be used.

While the application of electric motors to hoisting has been a pronounced success, there are still two obstacles in the way of the general utilization of electricity for all mining purposes, viz., the lack of electric drills and electric pumps. As soon as an electric drill is produced that will do the work of an air-drill, and an electric pump that will take the place of the present sinking-pump, then the time for the general application of electricity for all mining purposes, to the exclusion of other power, will be at hand.

Discussion.

R. W. Raymond, New York City: In this interesting survey of the department in which our mining engineers have done so much during the last two or three years I think the most important feature is the description, furnished by Mr. Van Depoele, of the electric-magnetic reciprocating engine. If I understand it correctly, the electric generator constituting the essential part of this engine gives a pulsating (not an alternating) current, which can be "converted" in the same manner as an alternating current, thus making it possible to do for power what is now done for electric lighting, namely, transmit electrical energy at high voltage over long distances, and transform it into induced quantity-currents at lower tension for use. The great saving of material for conductors is only one of the advantages of this method. As illustrated in the employment of the alternating current for lighting, it has achieved a rapid commercial success. But, so far as I know, the alternating current has never been converted with satisfactory results into a direct current for *power*. Since the "conversion" of a high-tension into a quantity-current is simply the production of an induced current at the cost of the energy of the primary one, and since the effect of induction depends on the electrical changes in the primary conductor, so that a direct and *perfectly uniform* current would generate no induced current at all, it seemed likely that the alternating current only, consisting as it does of a series of rapid reversals, could be effectively "converted." But the

resulting induced current is necessarily an alternating one likewise, and in that form has not been, so far as I know, successfully utilized for power. The problem, therefore, has heretofore seemed to me to be to take the alternating quantity-current derived by induction from an alternating high-potential current, and further convert it into a *direct* quantity-current. The operation of the commutator on the dynamo seemed to me to furnish a hint in this direction. But it was probable that this operation, even if practicable, would prove too cumbersome and expensive for any such general use as is now made of the ordinary transformers of the alternating system.

Now this employment of a regularly fluctuating, that is, a pulsating, direct current as the primary current, with the means of utilizing either the primary or its induced current to generate a reciprocating motion, appears to be an ingenious solution of the problem in an unexpected way. The important questions of economy and efficiency remain to be answered by careful tests and measurements. They are two-fold, relating first, to the efficiency of the engine itself as compared with the energy of the operating current, and secondly, to the economy of the conversion of a pulsating as compared with the conversion of an alternating current.

In my judgment, the second inquiry is more important, because effective motors for direct primary currents already exist; whereas, if this engine can be operated with economy by an induced current, it will command an exclusive field of its own.

On general principles, I should expect the conversion of a pulsating current to be less effective than that of an alternating one, because the changes on which induction depends are less complete. It seems reasonable to conclude, therefore, that to produce an induced current of a given electrical energy, the primary current, if pulsating, would have to carry a greater energy than if alternating.

But it must be remembered that the pulsating current is not competing in this field with the alternating, but undertaking to do something which the latter has not yet done. The real competition is with the ordinary direct primary current; and the exclusive field of the new device will begin where the considerations of economy make the usual system of electrical power-transmission disadvantageous.

MR. SPAULDING: Dr. Raymond's statement regarding the peculiarities of the reciprocating system is entirely correct, and I may say that the reason why more prominence is not given to this apparatus

in the paper just presented to you is that, while our experiments thus far have been perfectly satisfactory, we have not entirely completed them, nor have the results of the experiments already made been tabulated with sufficient comprehensiveness up to the present time to present before the Institute.

As soon as we have completed the work and have obtained accurate data regarding this apparatus I shall take great pleasure in presenting the results to the Institute for its consideration.

JAMES DOUGLASS, New York City: I may be forgiven for telling a dream that I have, which depends for its realization upon the transmission of power. Like all of us who are engaged in mining, I have been led to recognize the importance of agriculture, especially in its broad relation to that industry. Near one of the mining towns in the West with which I am intimately connected, we have a vast body of agricultural land, which is to all intents and purposes a desert, and yet it is underlain from five to thirteen feet below the surface by large bodies of water. The methods heretofore proposed for the irrigation of our western lands depend chiefly upon carrying water from our living water sources, or storing the water that falls in abundance at certain seasons of the year, and using it during dry seasons for the irrigation of desert tracts. Now, some miles from our mining town we are doing some work upon a coal-field and my dream is, if the coal-fields turn out as well as I hope they will, to use the slack-coal at the mines for the generation of power, transmit that power by electricity over this grand valley (which is about 120 miles long by 20 miles wide) and use it at a number of isolated pumping-plants. This would do away with the intricate system (attended likewise with considerable danger and risk), of large storage-reservoirs and of a very costly network of canals for conducting the water from such large storage-reservoirs to the periphery of the area which we intend to irrigate. It seems to me that such a use of the power in the West is sure to grow up and that large areas of our western land will ultimately be irrigated by isolated pumping-plants operated by electricity, transmitted from different sources where there is either water-power or slack-coal.

ECKLEY B. COXE, Drifton, Pa.: In some of our Pennsylvania coal-mines the electric locomotive has been successfully introduced. At the Lykens Valley mine, belonging to the Pennsylvania Railroad, they have had for the last two years a locomotive hauling coal about 6000 feet by electricity, and Mr. Irving A. Stearns, the engineer,

told me a short time ago that he thought he would put in a second plant. A year or so ago, when I was considering the subject for our works, he said, "Don't do it yet," but the other day informed me that he thought he would put in a second plant for a tunnel which he is constructing, some four or five thousand feet long.

In connection with the introduction of electricity into coal-mines, there is one point of which those engaged in coal-mining all over the world will probably recognize the importance, namely, the danger of a single fatal accident from its use, which would be almost certain to induce legislation absolutely forbidding the employment of electricity. A colliery manager must be a great deal more careful in making experiments than anybody else. In case the electric spark should ignite, or should be imagined to have ignited, the fire-damp, there would be trouble. In this experiment of Mr. Stearns, he did not hurt any man, but he did kill a mule. The electricity was transmitted by a regular railroad rail, which was spiked along the side of the track; and the mule, as some gentlemen who are familiar with him know, has a great habit of gnawing, and one day the mule stood in the water and undertook to gnaw this rail. The mule found out what a current was. That is the only accident they have had. The question of safety is, however, a very serious one; and I have been deterred from making any experiment, not so much because I think electric lights more dangerous in collieries than others, or do not believe that power-currents could be so installed as to secure a high degree of safety, so that the list of casualties from fire-damp might be not at all increased, or even somewhat diminished; but because one accident, in anybody's colliery, might precipitate a very natural (however unseasonable) peremptory legislative prohibition, rendering all our outlay useless. This consideration does not apply to open workings, or mines that contain no fire-damp; but it is not certain that collieries now free from this gas may not develop it at greater depths.

A MEMBER: I would like to ask Mr. Spaulding how many electric hoists are in use?

MR. SPAULDING: I am unable to state with any degree of accuracy the number of electric hoists. I think the first that I recollect were erected in the West by the Sprague Company, some two years ago. That was a small application of their regular railway type of motor to a worm-shaft; and the work since that time has been mainly in the adaptation of different shapes and different designs of motors

(series, shunt and compound-wound), to the ordinary hoist-drum. The advance has been in securing the regulation of these hoists and in getting them to work satisfactorily in the large sizes. In the Castle Gate mine, which is shown in some of the sketches in the latter part of my paper, there is to be a thirty-five horse-power hoist, to operate coal-cars, which, so far as I know, is the largest electrical hoist in the United States at the present time. This was to have been started up this week. The question of hoisting comes down really to one of regulation and safety, and indeed does not present so difficult a problem as the haulage and pumping; because the hoist is not necessarily (at least, not in all cases), subjected to constant dampness. The question of haulage seems to present greater difficulty; and several companies, the Jeffrey Company, the Sprague, the Thomson and Houston, and others are working at it all the time.

The matter of voltage and of possible accidents in mines mentioned by Mr. Coxe is one of tremendous importance. The 40 horse-power locomotive shown on page 263 was the first one of that size in the United States, so far as I know. It was put in at the Hillside Colliery at Erie, and the voltage was 220. We had a very interesting and amusing experience with the mules in that mine. The current can be taken off by a man with impunity. You feel it, but it will not injure any one. With a mule it seems to have the peculiar effect of loosening up the cords of the legs so that they don't stand steady, and the animal immediately sinks down to the ground as though he had a fainting spell. One mule especially, in the mine had been in the habit of amusing himself by kicking the miners; and they purposely took the loose end of the wire, and brought it in contact with his ears. Each time he would drop to the ground, and then get up and look at the wire, as if to try to understand what the trouble was; and the result was that ever since that time when the mule goes through the tunnel where this wire is, he will have one eye right on the wire all the time, and he has never been known to touch it accidentally since.

THE MAGNETIZATION OF IRON-ORE.

BY CLEMENS JONES, HOKENDAUQUA, PA.

(New York Meeting, September, 1890.)

THE peculiar property of the lode-stone has been known for many centuries. It was very early observed that, by contact with it, iron is magnetized; and it is needless to dwell here upon the history allied to this discovery, now constituting the wonderful science of magnetism.

Deposits of magnetic ore are distributed all over the world. The ore is sometimes very rich and comparatively free from gangue, sometimes sparsely disseminated in the native rock. These bodies of lean ore often cover large areas. Modern ingenuity has utilized the principle of the magnet, originally derived from the ore itself, and really belonging to the time of Plato and Theophrastus, to separate the magnetic ore-particles from the rock which has previously been crushed. This process recovers the ore in the state of a concentrate very rich in iron and, in some cases, desirable for Bessemer purposes. The magnetic separator is no longer a novelty. It was only at the last meeting of the Institute in New York, that Messrs. Birkinbine and Edison (*Trans.*, xvii., 728) and Mr. Robert Cook (*Ib.*, 599) described some of the principal designs of apparatus in use.

E. S. Dana classifies the natural magnetic minerals in the following order: magnetite, pyrrhotite, franklinite, almandite and minerals "containing considerable FeO." Of these, ferroso-ferric oxide, or magnetite, is the type. The proportionate amount of ferrous oxide essential to magnetize the compound, is not established. Ferrous oxide appears to be the only magnetic oxide of iron.

Tissandier (*Compt. Rend.*, lxxiv., 531) produced ferrous oxide in the anhydrous state, by the action of carbon dioxide on pure iron at a red heat, according to the equation:

$$Fe + CO_2 = FeO + CO.$$

This is black, crystalline and magnetic. Heated to redness in the air, it is converted into Fe_3O_4.

Sidot (*Compt. Rend.*, lxvii., 175) has also produced an oxide exhibiting magnetic polarity, by heating ferric oxide to a temperature

not sufficient to fuse it, in an earthen tube placed in the magnetic meridian.

A. Frenzel (*Neues Jahrb. f. Min.*, 1874, p. 685) accidentally obtained small rhombohedra, slightly magnetic, by igniting a precipitate of ferric hydrate.

The presence of ferrous oxide in magnetite, whether as a molecular constituent of the compound, or as an independent molecule, seems to communicate the property of magnetism.

Magnetic oxide of iron, as an artificial product, is considered to result from the direct action of carbon dioxide. In simple determinative blow-pipe work, the reducing-flame alone, or with the aid of alkaline carbonate, produces magnetic particles, or slag-globules, from nearly all iron-minerals.

Spathic iron-ores have long been known to exhibit, after roasting, magnetic properties—largely due, however, to the presence of metallic particles in the roasted ore. Carbon dioxide in the ore is also here considered an important factor. Of such ores, the ore-dressing at the zinc-works at Pribram, Bohemia, described by Mr. Ellis Clark, Jr. (*Trans.*, ix., 451), furnishes an illustration. The principle is here used for the separation of iron oxide from zinc-blende.

"These are separated from other material by the ordinary concentration processes, and are then in the form of a powder, the grains being less than 1 millimeter in diameter. This powder is composed of spathic iron-ore and zinc-blende, and is roasted in a small oven, with frequent stirring, for an hour, when the iron-ore is rendered sufficiently magnetic."

It is then separated by a magnetic separator, and the zinc-blende is thus recovered.

In nearly all cases of the carbonates of iron similarly treated, minute globules of metallic iron are contained in the roasted oxide. Furthermore, the expulsion of the carbon dioxide contained in the ore, with contact of carbon at a certain heat, approximates to the conditions of the blast-furnace and favors the reduction of part of the ferrous oxide to the metallic state. The roasted carbonate is always strongly magnetic. But, from the fusible nature of the ore, it is extremely difficult to prevent it from melting during the process, so that nearly all the associated minerals assume the form of "clinkers" with the ore. This circumstance, together with the expense of crushing the ore sufficiently fine, makes the magnetic separation of iron oxide from the gangue useless or unprofitable in such cases. At Pribram, zinc-blende is sought, and the oxide of iron is a secondary product.

Hematite is sometimes magnetic and, according to Dana, even

magneti-polar—of course, as an incidental occurrence, other than in crystallized specimens. On account of its rarity, as well as the feebleness of the attraction it presents, magnetic hematite is not available for practical purposes. Martite, the isometric ferric oxide which is supposed to be a pseudomorph after magnetite, is attracted by the magnet. The famous ore from Marquette, in the Lake Superior region, abounds with crystals of martite. The ore from the Juragua mines, in Cuba, is in part magnetic. Fayalite, $FeSiO_4$, is magnetic.

The anhydrous oxides of iron in general are magnetic. Of all the hydrous oxides of iron, not one is magnetic. Brown hematite or limonite (with which göthite and turgite are usually associated), as well as the sub-varieties of hydrous oxides, are non-magnetic, and have no effect upon the magnetic needle. The composition of these oxides is:

Limonite, ...	$2Fe_2O_3 + 3H_2O$, or 14.4 parts water.
Göthite, ...	$Fe_2O_3 + H_2O$, or 10.1 " "
Turgite, ...	$2Fe_2O_3 + H_2O$, or 5.6 " "

The importance of the brown hematites is indicated by the amount of such ores consumed in the manufacture of iron in this country and abroad. At one time brown ore was considered indispensable for the production of foundry-iron. Prior to the development of the Lake Superior hematite-regions, brown hematite was the chief ore in use in America. It occurs abundantly throughout Germany. "It supplies by far the greater number of the French iron-works; it also occurs in Spain, Russia,"* etc. In England the famed Forest of Dean supplied a great abundance of it. There are still vast bodies of the brown varieties unworked in England, and in this country they occur in nearly every State in the Union. Since the advent of the rich Lake Superior hematite-ores, hundreds of productive brown hematite ore-mines have been abandoned; and there are still numerous undeveloped beds too lean to be handled profitably. In Pennsylvania the mines of Lehigh and Northampton counties have furnished for quarter of a century the raw material of nearly half the iron made in the Lehigh Valley.

With the exception of the oölitic varieties, the localities named may be considered to represent all of the varieties of brown-hematite ores. At present only the better grades are in use, and of these the supply is limited. An investigation of the quality, product and yield of thirty of these mines for the year 1877, undertaken

* Crookes and Röhrig, *Treatise on Metallurgy*, ii., 332.

by the writer, showed that over 100,000 tons of this ore, represented by fully 500 samples and analyses, averaged only 37 per cent. of metallic iron. Three-quarters of this was wash-ore. The bulk of this ore, which will fairly represent the majority of this class of mines in the United States, will carry about 34 per cent. of metallic iron in the wash-ore.

Examination of the iron oxide or ore proper, shows it to contain from 45 to 60 per cent. metallic iron, which, in the wash-ore, is, in many instances, reduced below a marketable standard by the admixture of foreign material, chiefly quartz-gravel, slate and clay.

Separation of the ore has long been a problem presenting many-sided difficulties that have never been successfully overcome. Water-jigs and specific-gravity separators of innumerable designs have been invented for the purpose all over the world. But the very best results with these machines fall short of obtaining the ore entirely separated without loss in the tailings. A concentrate carrying 42 per cent. of metallic iron is a fair economical result from these machines, and, except with unusually rich ores, 45 per cent. of iron is seldom obtained. The only satisfactory solution of the problem lies in the recovery of all of the ore proper, entirely free from foreign material.

Some experience a few years ago with a method of water-separation acquainted me with the futility of attempting this result by that means or any other at that time known. With the modern demand for the richest material, it is easy to anticipate that a blast-furnace cannot exist among abandoned ore-beds. And the fact must not be overlooked that in this country nearly all of the older blast-furnace plants were located within or adjacent to the brown-hematite regions. A perfected method of concentrating would mean a survival of the brown-hematite ore as a source of supply.

On rapidly drying a small sample of limonite over a powerful Bunsen flame on one occasion, I observed that the smaller particles were magnetized. Could the ore be magnetized? A trial on a more practical scale convinced me that such was the case. I then experimented with several different ores, and found that all the ore-particles were so strongly affected as to permit their complete separation by means of a magnet.

Naturally, a series of experiments followed, in every instance giving the same results. My attention was then directed toward finding out the conditions as well as the cause of the phenomenon. This led me to the conclusion that the agency of heat in the pre-

sence of carbon or carbon dioxide magnetizes the hydrous oxides of iron.

Further on I will endeavor to qualify this statement; but before beginning its discussion, it may be interesting to allude to the practical method and the results, from a commercial standpoint, of some of the experiments.

The ore is placed in a convenient receptacle and merely roasted by the usual process, using either solid or gaseous fuel. It is undesirable to use much heat, since at a temperature of cherry-red magnetization is fully imparted. The ore is drawn as fast as it reaches this temperature, and is at once ready for magnetic separation.

In the following list, for obvious reasons, the ores are designated alphabetically.

Iron in Crude Ore. Per cent.	Iron in Concentrate. Per cent.	Iron in Tailings. Per cent.	Ore Recovered as Concentrate. Per cent.
A, 40.98	55.88	.....	...
B, 34.32	51.72	.....	...
C, 38.04	45.24	.....	68
D, 40.63	50.04	.....	80
E, 20.30	46.32	5.21	55
F, 40.00	53.71	.....	65
G, 49	60	.....	80
H, 37.06	50	.....	65
I, 41.31	52.46	.....	85
J, 35.35	55.04	.....	60
K, 33.55	48.63	2.56	70
L, 31.31	48.87	5.20	70
M, 39.84	51.29	5.00	78
N, 42.55	55.36	5.76	86
O, 43.84	55.61	.....	70
P, 42.96	54.43	.....	85

A, *B*, *C*, *D* and *E* are from Pennsylvania, *A* and *B* being wash-ore; *C*, washed and jigged ore; *D*, a fine refuse-sand; and *E*, a similar material; *F*, an ore separated by a Bradford jig; *G*, ore from a Northampton county, Pa., mine; *H*, the tailings rejected by a new jig; *I*, a Pennsylvania ore; *J*, a lot of six cars of ore "condemned" at the works; *K* and *L* are from the same mine in Pennsylvania and impossible to separate by water-jigging; *M* is a mixture of fine ore from Pennsylvania mines; *O*, an ore from Connecticut; and *P*, the fine ore from a large mine in New York.

The commercial success of treating ores in this way depends on the three items: cost of wash-ore; cost of roasting; cost of concentration. The first may be considered to range from 75 cents to $1.50 per ton, according to locality. The second item will be fully covered by the following statement, taken from the books of a large company exclusively engaged in roasting carbonate ores, and at that time under the most capable management.

Actual Cost of Roasting and Handling Ore.

PER TON OF ORE.

Roasting Ore.				Handling and Wharf Expenses.			
	1886.	1887.	1888.		1886.	1887.	1888.
Labor, including car- and top-man..	$0.0883	$0.0673	$0.0811	Labor, includ'g engineer & teamster	$0.0514	$0.0439	$0.0720
Supplies.....	0.0024	0.0032	0.0091	Supplies and coal....	0.0177	0.0132	0.0169
Coal.....	0.0725	0.0750	0.0748	Team	.....	0.0040	0.0062
				Repairs, etc.....	0.0046	0.0025	0.0029
	$0.1632	$0.1455	$0.1650		$0.0737	$0.0636	$0.0980
Culm, per ton.....	2.00	2.10	2.25				
Average cost, $0.1600. About one ton of culm used to 30 tons of ore.				Average.....$0.0700 Average total cost... 0.2300 Per ton of ore treated.			

The third item remains to be determined, namely, the cost of concentration, including royalty on the process. This would be 2 cents per ton for concentrating and 25 cents royalty. Taking as an example ore K in the foregoing list, the cost of the concentrated ore would be as follows:

Cost of 1½ tons ore, ...	$2.25
" roasting, ...	36
" concentration and royalty, ...	27
	$2.88

At a yield of 48 per cent. iron, this is 6 cents a unit, or, in round numbers, $6.00 to the ton of iron.

To determine the extent to which the other constituents of the ore were affected, I tested several of the ores, of which two examples will suffice. They gave the following results:

Ore E.	Raw.	Concentrates.	Tailings.
Iron, ...	20.30	46.32	5.21
Phosphorus, . . .	0.12	0.30	0.09
Ore L.			
Iron, ...	31.31	48.87	5.20
Silica, ...	.....	20.25	83.20
Manganese, . . .	.....	2.20	1.18

The surprisingly small amount of ferrous oxide in the magnetized ore leaves the question, whether magnetization is wholly due to its presence, very uncertain. In the list of ores given, G, carrying 60 per cent. of iron, which would be presumably high in ferrous oxide, contains but 3.07 per cent. of ferrous oxide. In some cases the raw

ore contains nearly this amount. If the reduction caused by roasting were the sole cause, it would be reasonable to expect the *exterior* of the particle to be the magnetized portion. But lumps of the roasted ore having been broken, small pieces, carefully selected from the inside of the lumps, showed in some cases even more strongly-marked magnetic properties.

A number of trials with the anhydrous oxide, both red and specular hematites, conducted in precisely the same manner, gave no evidence whatever of resulting magnetization. Crystallized specimens remained likewise unaffected. While the porous character of the brown hematite would permit permeation by reducing gases, it is difficult to understand how such minute deoxidation could magnetize it so strongly. Moreover, the magnetization appears to be permanent. Samples which I have kept for over a year in contact with air, have preserved the quality unimpaired. Cooling the red-hot oxide in water affects it somewhat, but it is still capable of responding through that medium to the attraction of a magnetic field.

The phenomenon seems to be connected with the expulsion of the combined water, and I think is largely dependent on its physical separation from the ore. During the process some of the oxides glow quite strongly. The raw oxide, when heated to redness in a closed glass tube, gives off a small amount of carbonic acid with its water, but is only feebly magnetic. There is no apparent evidence that ferrous oxide is produced by decomposition of the water. The only statement now possible is, that the hydrated oxides of iron become magnetized at a red heat in contact with carbon or carbon dioxide.

It may be worth noting that the idea of employing a magnetic separator in a process for "manufacturing cast steel and malleable iron of a superior and uniform quality directly from ores" was announced some years ago.* The curious supposition on which it was based was, that when "impure ores" are brought to a white heat in a furnace (thereby producing metallic sponge), they can be separated from "pure ores" by the "simple mechanism of revolving magnets." Some of the "noxious elements" thus to be eliminated were "titanic acid, vanadic acid, chromic acid, sulphuric acid, sulphur, phosphorus, phosphoric acid, silicic acid, silicates." The resulting product was to be simply melted.

It is of course a logical deduction, that all hydrous ores of iron become magnetic in the blast-furnace, and that at the proper zone

* In a U. S. patent by E. L. Seymour, 1867.

even anhydrous hematite does so. Ore concentrated as above described is peculiarly adapted to use in the blast-furnace. Before the ore-particles reach a red heat in the process of roasting, violent decrepitation takes place, thus breaking or splitting up into small fragments all the lumps of ore. This state is most desirable for rapid reduction in the blast-furnace, and, aided by the easy fusibility of the ore, offers the most favorable conditions for "driving" and regularity of work.

The subject-matter outlined in this paper is embraced in a process on which application for letters-patent has been made.

THE PRATT MINES OF THE TENNESSEE COAL, IRON AND RAILROAD COMPANY, ALABAMA.

BY ERSKINE RAMSAY, PRATT MINES, ALA.

(New York Meeting, September, 1890.)

THE following description of the Alabama coal-fields is taken (with slight omissions) from the report "On the Warrior Coal-Field, by Henry McCalley, A.M., C. and M. E., Assistant State Geologist:"

The coal lands of Alabama are of the great coal basin of the Ohio, or of the great Appalachian coal-field, and are estimated at 8660 square miles. They formed one connected field prior to the great Appalachian revolution, when there was pushed up through them long, narrow, anticlinal ridges which have been denuded into the present well-known anticlinal valleys, and thus this great field has been divided up into three more or less distinct parts. These parts were named by Prof. Tuomey, in 1849, the Warrior, the Cahaba and the Coosa coal-fields, respectively, from the names of the rivers which drain them. They are of very unequal size, the Warrior, with an area of 7810 square miles, being nearly ten times as large as the other two combined, and about two-thirds as large as the coal-area of Great Britain. This Warrior field is the most northwestern of the three coal-fields of Alabama, and comprises all of the coal-measures in Alabama, northwest of the Alabama Great Southern railroad, or all of those drained by the Warrior and Tennessee rivers. The greater part of this field, almost all of it away from the present railroad lines, has never received more than a surface-examination. Still, we have in the Warrior coal-field one of the richest coal-basins of this or any other country. . . . As a whole, the Warrior coal-field is a broad, shallow, tray-shaped depression, sloping towards the southwest, and with its southwest end covered by a newer formation and its southeast side, especially, complicated by folds and fractures. As is the case with all of our Coal Measures, it is composed of a series of sandstones, conglomerates, shales, slates and coal-seams, with some little limestone at several horizontal positions, but contains no thick bed of this last rock like the Measures of Pennsylvania and other States. The thickest conglomerates are near the bottom and top of the series, and hence these coals might be said to belong mainly to the

inter-conglomerate Measures. The general dip of the strata is some few degrees to the southwest and, as this dip is greater than that of the inclination of the surface of the country, the measures thicken as you go to the southwest until they become covered by the drift. In addition to this small angle of dip, which is characteristic of the Warrior coal-field, when compared with the other coal-fields of Alabama, the strata of the northwest side of the field have a general dip to the southeast, and those of the southeast side to the northwest, and thus the basin shape is given to the area. The strata of these two sides are also in long, flat waves from northwest to southeast, while those near the center of the basin are in similar waves from northeast to southwest. . . .

Provided all the coal-seams of the Warrior field will average, throughout their whole extent, a thickness equivalent to that of their most accurate and reliable measurements, they will contain, we believe, a sum total of coal, of not less than 113,119,000,000 tons, of which about 108,394,000,000 tons would be available coal, or of the seams eighteen inches and over in thickness. These figures are simply enormous, and tell us, in plain black and white, that the available coal of the Warrior coal-field is three times that of the estimated available bituminous and semi-bituminous coals of the great coal-producing State of Pennsylvania, and that, if this coal was spread out evenly over the surface, it would cover the whole State of Alabama (52,250 square miles in extent), over two feet in thickness, and that, at the present rate of consumption of coals of all kinds, it would last the whole world for over two hundred and seventy years.

The map (Plate IV.), shows the Warrior and other Alabama coal-fields, and their relation to iron-ore deposits and centers of iron manufacture.

While coal-mining in Alabama has been carried on to some extent for a considerable time, it is only within the last few years that it has taken long strides, and now ranks at the head of the Southern States, having passed Tennessee and the Virginias. The same may be said of the manufacture of coke, in which, also, the development has been rapid and steady.

The following table gives analyses of three representative coals of this field, together with that of the Connellsville coal.

	Pratt.	New Castle.	Black Creek.	Connellsville.
Specific gravity	1.299	1.33	1.36	.....
Sulphur	1.041	.64	.10	1.06
Moisture	1.025	.50	.12	1.20
Volatile matter	32.169	28.24	26.11	28.50
Fixed carbon	63.370	59.69	71.64	63.12
Ash	3.342	10.92	2.93	6.12

To indicate the actual value of the Pratt vein, for making steam or for heating-purposes, in comparison with other well-known

coals of Alabama and other States, the following table is given. These tests were made by Prof. O. H. Landreth, of Vanderbilt University, Nashville, Tenn. Ten tons of each kind of coal were used, thus making the test a measure of actual practice:

NAME OF COAL.	Pounds of water evaporated from and at 212° per lb. of coal.	Per cent. of non-combustible from combustion under boiler.	Relative heat'g or steaming values, Cumberland being 100.	Name of State.
Cumberland	8.21	11.5	100.0	Maryland.
Pratt	8.04	7.4	97.9	Alabama.
Jellico	7.45	6.3	90.7	Tennessee.
Pittsburgh	7.63	7.4	92.9	Pennsylvania.
Altmont	7.41	3.5	90.3	Kentucky.
St. Bernard	6.73	6.9	82.[illegible]	"
Warrior	7.73	4.6	94.2	Alabama.
Helena	7.58	7.7	92.3	"
Watt	7.11	13.2	86.6	"
Diamond	6 20	10.2	75.5	Kentucky.
Mud River	6.89	4.06	83.9	Illinois.
Memphis	6.45	8.05	78.6	Kentucky.
Clifton	5.74	14.6	69.9	"
Sewanee	7.37	11.3	89.8	Tennessee.
Cahaba	7.65	5.0	93.2	Alabama.
Blocton	7.37	5.7	89.8	"
Black Creek	7.63	4.0	92.9	"
Henry Ellen	7.25	5.2	88.3	"
Daisy	7.16	11.4	87.2	Tennessee.

Among the seams now opened and worked, the Pratt is the most valuable and important, being mined most extensively, and furnishing the greater part of the State's output of coal. Jefferson county, in which this seam is mined, furnishes nearly three-fourths of the coal-product of Alabama. The other coal-producing counties are Bibb, St. Clair and Walker, ranking in the order named.

That part of the Warrior field which is underlaid with the Pratt seam is undulating, yet sufficiently level to permit the building of railroads at moderate expense. The field now worked is drained by

Village and Five Mile creeks, the former of which has its source near Birmingham.

There is plenty of timber on the coal-lands for mining operations, and it is well distributed over the entire field, thus making it easy of access, and consequently cheap. The mine-timber is cut near the various mines, and delivered by teams to the pit-mouth. Oak and pine are the principal woods, although beach, walnut, gum, ash and other woods are found also.

The Tennessee Coal, Iron and Railroad Company, the Sloss Steel and Iron Company, the Woodward Iron Company and the Coal Creek Company operate in the Pratt seam. Of these, the Tennessee Coal, Iron and Railroad Company's mines, known as Pratt mines, are the most extensive. The remainder of this paper will be confined principally to Pratt mines.

William Gould was the first to open the Pratt seam, which, at that time, was called the "Gould" seam, in honor of the pioneer of the field. He began operations near what is now called Slope No. 1, and had planned to build a pole-road from the mines to Birmingham, a distance of about six miles; but, after having built but a fraction of it, he sold out to Mr. H. F. DeBardeleben and his associates, who began working the mines on a more systematic and extensive scale; and a standard-gauge railroad was built from the mines to Birmingham. The first mine was Slope No. 1, opened about 1879. It is still in operation. The next opening was Shaft No. 1, after the commencement of which, the property passed into the hands of Col Enoch Ensley and others, of Memphis, Tenn., with the title of the Pratt Coal and Coke Company. Then followed the opening of Slopes Nos. 2 and 3, during the execution of which the Pratt company consolidated with the Alice Furnace Company, changing the name to the Pratt Coal and Iron Company. During 1886 the Tennessee C. I. and Railroad company absorbed the latter company, and immediately began to develop the property extensively. The construction of the Ensley plant of four blast-furnaces was begun, about $1\frac{1}{2}$ miles from the town of Pratt Mines, and to supply these furnaces with fuel, the coke-plant at the mines was increased from 110 to 806 ovens; and Slopes Nos. 4 and 5 and Shaft No. 3 were opened to meet the increased demand for coal.

The town of Pratt Mines has a considerably scattered population of about 5000. It is located about six miles from Birmingham, with which place it has connection by the following railroads: Pratt Mines railroad; Kansas City, Memphis and Birmingham railroad; the mineral branches of the Louisville and Nashville railroad; the Alabama Great Southern railroad, and the Georgia Pacific railroad.

Under the head of Pratt mines are included Slopes Nos. 1, 2, 3, 4 and 5, Shafts Nos. 1 and 3, and the Drift mines. All these mines are situated on or near the southeastern outcrop of the Pratt seam of the Warrior coal-field. The slopes are opened usually beyond the outcrop of the coal, and begin with a smaller angle of dip than that of the coal, so that the seam is first struck from the under side, and when the height of coal is reached, the slope is simply continued on in the coal. The dip of the coal extends one or two miles into the field, although much flatter as the distance from the outcrop becomes greater. At the outcrop, the dip is in some places as high as 20 degrees; but this decreases rapidly at first, and more gradually afterwards, as the field is penetrated. These favorable conditions of dip make it possible to operate the slopes for long distances as simple engine-planes. At Slopes Nos. 2, 4 and 5, the coal is taken from the various lifts, which are usually 300 feet apart, by the slope-engine at the surface, for a distance of 2000 feet, beyond which point the cars are brought forward to the chain by either mule- or engine-power. The plan of working the coal has been until within the last two or three years an incomplete system of pillar-and-room. The idea was to take out all the coal possible, just leaving sufficient "rib" to sustain the superincumbent strata. Very naturally the tendency of this system was to go beyond the safe limit; and, as a result, a crush or creep would be brought on, with a loss of rooms which it had taken time and money to make. The small quantity of coal, left in the shape of small pillars, which would be able to support, for a time at least, the surface, was really surprising to those acquainted with the comparatively narrow rooms and wide pillars left in other regions. Another very serious objection to this plan of working was, that the coal left in these stumps or pillars was forever lost. Instead of being permanently safe, the roof was a constant source of danger, standing, as it were, on stilts, and ready to fall in at any time. Dissatisfaction with this method led to the adoption of the plan shown in Fig. 1 (Plate I.), which was introduced gradually as opportunity was presented. As will be seen from an inspection of the sketch, butt- or room-headings, usually running about water-level, are driven off the slopes or main headings at intervals of 300 feet, and the body of coal intervening is worked out by first driving rooms 40 feet wide and 60 feet from center to center, for a distance of about 275 feet. This leaves a pillar of 20 feet between rooms, which is drawn back to the heading just as soon as the rooms are finished. In this way all the

coal from the fire-clay bottom to the sandstone top is taken out between the headings, leaving the entry-stumps to protect the entries. When the room-headings have been driven to their destination, which is usually about 3000 feet, the entry-stumps are drawn back, the double-entry system providing ventilation for the miners while engaged at this work. By this means practically all the coal is taken out. It may fairly be said that not more than 5 per cent. is lost.

There are two widths of rooms, 20 feet and 40 feet, with pillars of 12 and 20 feet respectively. They are distinguished as single and double rooms, from the fact that the single rooms have one loading-track while the double rooms have two. Tee-iron rails are used for all tracks: those in the rooms weighing 12 pounds per yard; in the room- or butt-headings, 16 pounds; and in the main-headings or slopes, thirty-five pounds. The use of iron rails in the rooms is made necessary, or at least desirable, by the fact that the miners could not well handle the cars on wooden tracks upon grades so heavy as those in some parts of the mines, especially near the outcrop. From an economical standpoint they are better in the long run than wooden rails, which will only last during the lifetime of one room, while the iron will last for an indefinite period. The miners lay their own track of iron rails, just as they do in mines where wooden ones are used. In the regular working of rooms nothing is taken out but the coal, which is 4 feet 6 inches thick. The roof being usually a hard sandstone, which stands up well, requires very little propping. No mules are taken into the rooms; the miners receiving the empty cars from, and delivering the loaded cars to the driver on the heading at the mouth of the room. In the room-headings, as well as in all places where mules are intended to work, about 1½ feet of top is taken down or the same amount of bottom taken up, depending altogether on the relative ease of excavation. A height of 5 feet 3 inches from the top of the rail is given to all hauling-roads, this height being necessary for the mules. When it is not thought desirable to haul this rock to the slate-dump on the surface, the headings are driven 12 to 15 feet wide, thus giving ample room to "gob" it away on one side.

As is no doubt the case in most mining regions, the Pratt mines were operated for years without mine-maps of any kind; and many natural advantages were passed by unconsciously, with the result that the coal was taken out at greater cost, and a smaller number of tons to the acre was won, than might have been secured. But as the

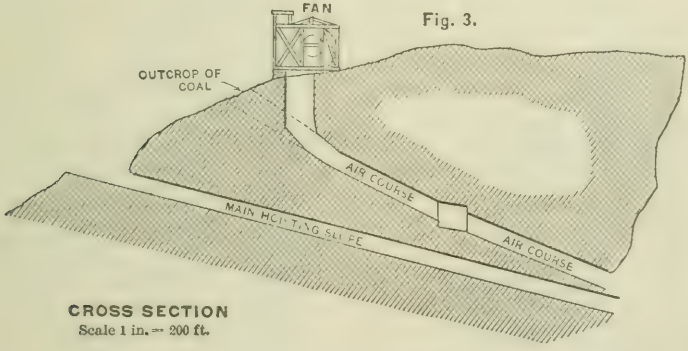
mining business grew, bringing in new competition, and requiring better economy, accurate and complete mine-maps were seen to be important; and such have been prepared. There are now maps of all the mines on a scale of 100 feet per inch, and showing all workings. The course of the air-current through the mines is shown by arrows, and the tidewater elevations are shown at all the more important stations, both inside and outside. A corp of engineers is constantly employed, to keep up the surveys and maps as close to date as possible. The time of the surveys is noted on the maps, and the coloring is changed at intervals to correspond with the surveys, thus making them chronological charts of the progress of the work. Contour-lines are run in for every 5 feet difference in elevation of the surface. The creeks, railroads, and outside improvements are shown on the maps in their relative position to the mine-workings.

Water is not encountered in large quantites in these mines, and what there is contains very little sulphuric or other free acid. Hence there is here none of the trouble incident to bad mine-water, eating out the pumps and pipes. Wrought-iron pipes and cast-iron pump-cylinders are used, and last for years. The wrought-iron pipe put in at Slope No. 1, ten years ago, did duty until a few months since, when it was abandoned because the water from this mine had been drained to Shaft No. 1. Here, together with the Shaft No. 1 water, it is elevated to the surface, some 225 feet, through a 12-inch cast-iron column-pipe, by a "Yough" pump, having 26-inch steam-cylinder, 14-inch water-cylinder and 48-inch stroke.

With the exception of Slopes Nos. 4 and 5, and (partially) No. 2, the mines are ventilated by the continuous current system. At Nos. 4 and 5 the split-current system is used, as shown in Fig. 1. By reference to this figure it will be seen that all the air enters at the mouth, and, travelling down towards the face of the slope, parts with a portion of the current at each set of lifts, where it divides right and left. The remainder continues down the slope to the next lift, where the same thing occurs; and so on to the face of the slope, where what air remains splits right and left and returns along the air-course running parallel with the slope. These currents on either side of the slope are augmented by receiving from the last pair of headings, through their respective air-courses, what air they received from the main current on its way down. The amount of air received at each of the lifts is regulated by air-regulators in the air-courses of the room-headings, and near the slope

air-course. At each of the lifts the air on its return along the main air-course crosses by means of an overcast, as shown in Fig. 1, and proceeds to the next heading, and so on, until the fan is reached near the mouth of the slope, and the air passes out. A fan of the Guibal type, 15 feet diameter by 6 feet face, is located on either side of the slope at the air-shafts, thus making the ventilation of each side of the mine independent of the other, so that, in case of a break-down, only one side would be affected, and this only for a few minutes, or until the proper doors could be opened and closed and the remaining fan caused to furnish the ventilating-power for both halves of the mine. This arrangement is shown in Figs. 2 (Plate I.), and 3, the former being a ground-plan and the latter a section. It will be seen that, should any accident occur to the fan D, the simple closing of door B and the opening of doors A and C, or, *vice versa*, should fan E become disabled, the shutting of door A and opening of doors B and C, would place both sides of the mine in communication with fan E or D respectively. Against this plan of ventilation it may be urged that it is too expensive; but the price of these two fans was but very little above that of one fan of the same aggregate capacity.

Fig. 3.



Pratt Mines, Mouth of Slope.

That the expense of an overcast is not much more than that of a simple trap-door cannot be questioned; and to offset this small excess, it must be remembered that not a single man is required to attend the overcast. At present, the saving on trappers alone amounts to ten boys at each mine.

Aside from the economical advantage of this system, it has many others, among which may be mentioned that, there being no doors,

there can be none left open, thus disarranging the entire ventilation beyond; that the drivers are not hindered in their work by having to pass through trap-doors or wait for them to be opened, but have a clear road all the time; that each heading receives its own quota of pure, fresh air direct from the slope, so that the men working therein are not compelled to re-breathe air already vitiated by man and beast; and that the gases from blasting with powder or dynamite do not affect any person beyond the heading in which the blasting occurs, so that the men driving the headings can shoot whenever they please without affecting in any way those laboring elsewhere.

The coal-seam is remarkably uniform in quality and thickness (averaging 4 feet 6 inches). One of the notable geological features of the seam is the persistency of a band of slate, about 8 inches from the top, which ranges in thickness from 1 to 2 inches. This is a distinguishing characteristic of the vein, and is the point usually looked for by prospectors, who, when they find it at the proper place, are sure they have the Pratt coal.

The usual mode of mining is first to undercut about 3 feet, and shear along the side or rib. Then holes are bored in the coal (the number depending on the width of the place and the ideas of the miners), usually by some hand-drill; the holes are charged with powder, and the coal is shot down; after which it is cleaned of the band of slate, and is ready for loading into the 25-cwt. mine-cars.

The greater portion of the coal at Pratt mines is mined by hand; but there are at Shaft No. 1 some 13 Harrison mining-machines. The coal is admirably adapted to the use of these machines, owing to the comparatively great width of the rooms, which reduces to a minimum the time lost in shifting machines from one room to another. The machine is simple in construction and has few parts to get out of order and require renewal. It may be described as a simple cylinder mounted on two wheels, with the pick, which is shaped somewhat like a fish-tail, attached to the end of the piston-rod. The reversing-mechanism consists of a very small rotary engine, which runs at about 200 revolutions per minute, and operates a common slide-valve, admitting the air into and out of the main cylinders, as is the case in common slide-valve engines. The small size, great strength and simplicity of the Harrison mining-machine adapts it to this kind of work. It can be worked around and between props, wherever a miner can swing a pick. One skilled machine-runner and one ordinary mine-laborer, to shovel away the cuttings, are required at each machine. It cuts an open channel

under the coal of any desired vertical height, usually from 8 to 10 inches in front and tapering to 2 inches in the rear, making an average of 5 inches. The depth of the undercut is made 4 feet 6 inches. The manner in which the machine attacks the coal is very much like that of a hand-miner. The machine is mounted on platforms which

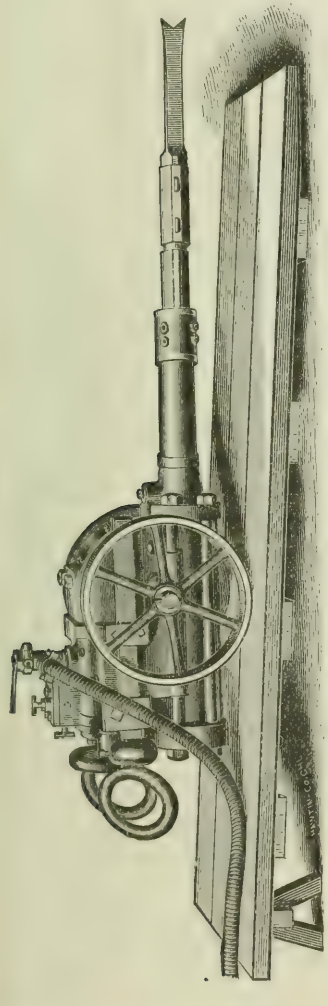


FIG. 4. The Harrison Mining-Machine—Outside View.

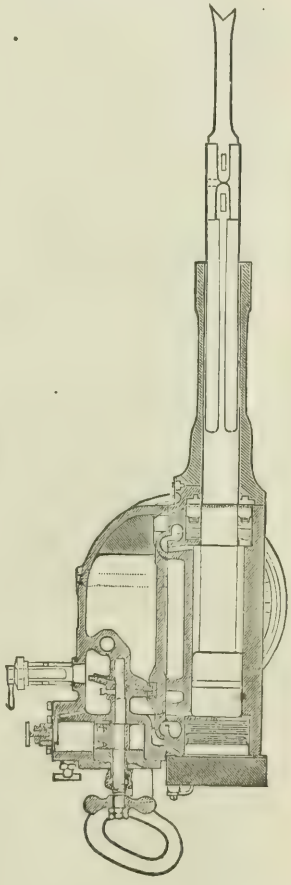


FIG. 5. The Harrison Mining-Machine—Longitudinal Sectional View.

are so pitched towards the coal that the recoil, when the pick strikes the coal, is neutralized by gravity. The pick strikes from 175 to 200 blows per minute; and as it works under the coal, the runner

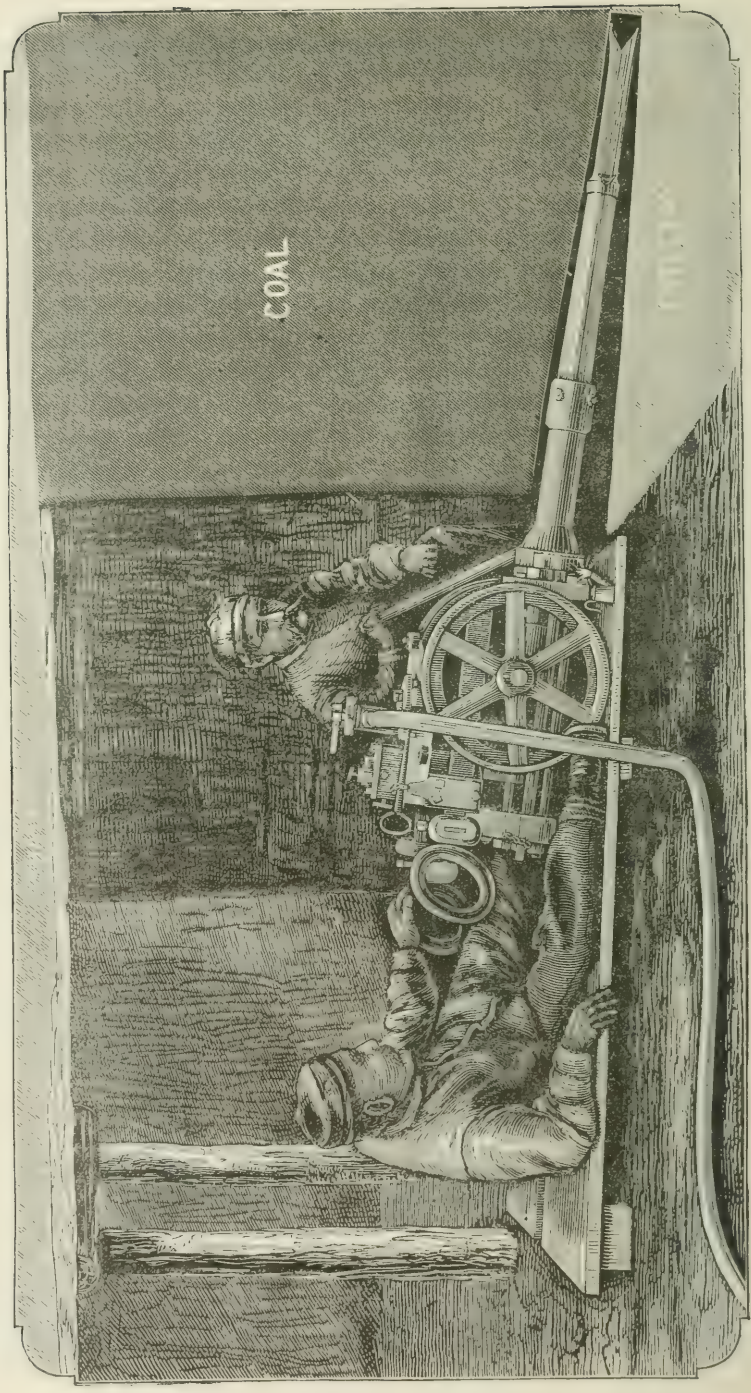


Fig. 6.

The Harrison Mining-Machine at Work.

allows the machine to run forward down the platform and swings the pick from side to side as it advances, while the helper throws back the cuttings. When the machine has made the cut, it is moved along the face and begins work on a new cut. With this machine a man undercuts along the face for a distance of 90 feet and to a depth of 4 feet 6 inches per shift, making a total area of 405 square feet of floor-surface cut over, and this produces about 60 tons of merchantable coal. To gain a better idea of the construction, form, and working of the mining-machines, the reader is referred to Figs. 4, 5 and 6, which are respectively an outside view, a longitudinal sectional view, and a view of the machine at work.

In addition to the power mining-machines, there are at this mine several power rock-drills of the Sargeant, Ingersoll and Rand make. These drills are used to do such rock-work as the blasting of top or bottom to make height for the mules in the hauling-roads, or the sinking of shafts, etc. Two Allison duplex air-compressors, having 22-inch steam-cylindres, 24-inch air-cylinder, and 42-inch stroke, furnish the compressed air to operate the mining-machines and rock-drills, as well as some of the smaller direct-acting pumps, located in remote parts of the mine. Either one of these air-compressors has sufficient capacity to do all the work singly in case of accident to the other. The use of compressed air is of great advantage in the mines, since it does away with the very objectionable heat from steam-pipes and the equally bad exhaust, which causes the roof to cave in.

From the air-compressors the air is delivered directly into a receiver, 46 inches in diameter by 24 feet long, whence it is conveyed to the bottom of the shaft through 8-inch pipes and there divides, travelling in 6-inch pipes to both sides of the mines, along the main roads. In the room-headings, $2\frac{1}{2}$-inch pipe is found large enough in most cases; but this depends to a great extent on the distance. A $1\frac{1}{4}$-inch pipe is large enough in the rooms; the connection being made from the pipe to the machines—whether mining-machines or rock-drills—by 1-inch rubber hose.

An example of one of the shaft pit-head arrangements is given in Figs. 7 to 12 (Plate II.), inclusive, which are in the main self-explanatory. The general arrangement is such as to give great rigidity to the derrick, a side view of which is shown in Fig. 7, while back and front views are given in Figs. 9 and 11. Fig. 10 is a plan of the brace extending from the top of the derrick to the stone and cement engine-foundation. It will be observed that in the con-

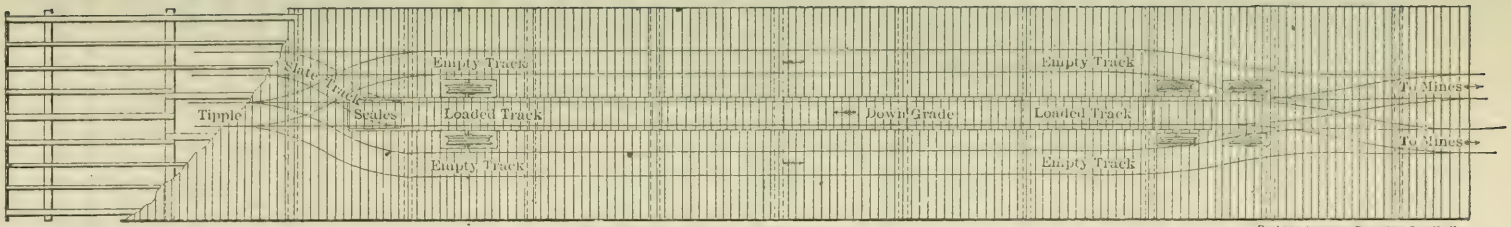
struction of the derrick and its brace, instead of having the main timbers of one single piece, they are composed of two 6- by 12-inch timbers, put together in such a way as to permit the structure to be rebuilt piece by piece, without shutting down the plant for a day. The engines are direct-acting, coupled direct to the drum-shaft, and have the crank-pins set 90 degrees apart. They are provided with cast-iron conical grooved drums, and $1\frac{1}{4}$-inch cast-steel wire rope. The cages are provided with safety-attachments and are wide enough to handle two $1\frac{1}{4}$-ton cars, side by side, at each hoist. Fig. 8 is a plan of the tipple-arrangements, showing the cast-iron plates, scales and tipples. As will be seen, all the coal is dumped on one side, the loaded cars being taken off the cages on that side and the empties put on from the opposite side. Eight-inch buntons are set in the shaft from top to bottom, 4 feet apart from center to center, to which are attached 8- by $10\frac{1}{2}$-inch yellow pine guides.

The arrangement for handling the coal at Slopes Nos. 4 and 5 is shown in Figs. 13, 14 and 15. Both these plants are constructed of native pine timber, the bents being 10 by 10 inches, and the deck floor of 3-inch pine plank. The winters being mild in this State, the tipple-house is not closed on the sides; and the roof is of corrugated sheet-iron to reduce to a minimum the danger from fire. These slopes, like all the slopes at Pratt Mines, have two tracks running into the mines, and at Slopes Nos. 4 and 5, as will be observed in Fig. 14, these tracks converge to a common central track extending from the knuckle to the dump. This track has a down-grade of 1 per cent. in favor of the loaded cars, which greatly facilitates their movement forward to be dumped. When the cars have been dumped they are switched right or left to the empty tracks, on either side of the loaded track and parallel with it. Both empty tracks are also on a down-grade of 1 per cent., but in an opposite direction to that of the loaded track. This permits the empty trip to be easily moved off at the knuckle and sent into the mines.

The hoisting-engines are double, 12 by 20 inches, and geared 1 to 4. The drums run independent of each other, being loose on the drum-shaft, and, by means of a V-clutch, are thrown in and out of gear by the engineer at will. By this arrangement the engineer is enabled to lower into the mines either trip singly, or both at the same time, at different speeds and to different lifts.

Where power is applied in the haulage of coal underground, the tail-rope system is generally used, except where, the grades being heavy enough and in one direction, the simple engine-plane takes

Fig. 14.



GROUND PLAN OF TIPPLE-FLOOR.

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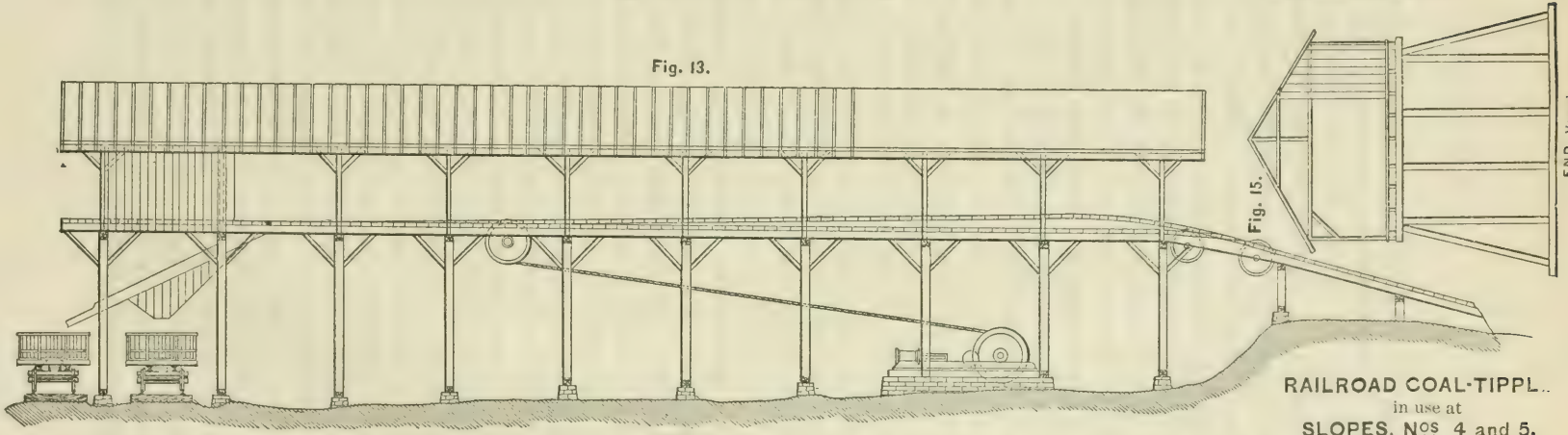
Fig. 13.



SIDE VIEW

Fig. 15.

END V

RAILROAD COAL-TIPPL..
in use at
SLOPES, Nos 4 and 5.
PRATT MINES.
Scale: 1 inch = 24 feet.

its place. An example of a tail-rope haulage, now under course of construction at Shaft No. 1, is given in Figs. 16 to 22 (Plate III.), inclusive. This haulage has but one curve in its entire length of about 3800 feet. The engines are located at the curve, 140 feet back from the hauling-road, to give sufficient lead to cause the rope to coil regularly on the drums. From the curve, or engine-location, to the point where the coal is delivered is about 2700 feet, while the distance in the opposite direction to the return-shieve is about 1100 feet. It will be observed, that the track is thrown outside of the true center-line at the curve. This is done in order that the rope may always remain in the center of the track (curve excepted), while at the same time it is running in the shieves located at one side of the track. Fig. 17 shows one of the curve-shieves, which is so constructed as to make it impossible for the rope to fail to go into the shieves automatically, the instant the trip has passed the curve in either direction. The tail-rope is supported at intervals of 30 to 50 feet on 12-inch cast-iron shieves, which are set in perfect line, both laterally and vertically, thus giving each shieve its own part of the rope to sustain, and doing away with any side-thrust caused by the shieves being out of line. Instead of having the shieves supported in a frame, a 3- by 12-inch pine timber is used, with an opening cut through it just large enough to receive the shieve, which prevents the rope from jumping off. The shieves run loose on a $\frac{3}{4}$-inch pipe, as a spindle, which is closed at one end while the other is turned up so as to contain a quantity of oil. On the under side of the pipe, and centrally located with reference to the shieve-wheel, is a small hole, immediately over which is a wad of waste to prevent the too rapid flow of the oil from the pipe to the journal. With this arrangement the shieves are oiled once a week, and it is a very rare thing for one of them to run dry. Two views of this arrangement are given in Figs. 18 and 19. A cross-section of the tracks at either terminal of the haulage is shown in Fig. 20. The difference in elevation of the two tracks is due to the grading in opposite directions, as already explained, of the empty and loaded tracks so as to facilitate the handling of the cars. To supply the steam for the haulage-engines, as well as the pumps used in the mines, there are located within a few feet of either, and near the bottom of an air-shaft 360 feet deep and 7 by 10 feet in size, two 2-boiler batteries of cylinder boilers, 46 inches diameter by 24 feet, either battery of which is able to supply sufficient steam to operate the engines and pumps. The boiler-house is neatly arched with a 13-inch hard-brick

wall, set in cement, thus making it fire-proof and perfectly safe against the spawling or falling in of the mine-roof. The boilers are supplied with coal from the regular mine cars, which are run into the boiler-house as shown. The waste heat from the boilers, which would ordinarily pass up the smoke-stack, is here utilized by being led into the air-shaft, thus heating the returning air from the mines, and aiding the current of ventilation. A brick partition is built in the shaft to prevent the flame or heated gases, from the fires under the boilers, from coming in contact with the return-air from the mines and causing an explosion, should there be from any cause sufficient gas contained therein to render it explosive. This is a very remote possibility in mines generating as little carburetted hydrogen as these mines do. By placing the boilers in the mines, a great deal is saved in the hauling of fuel from the mines to the boilers; and, in addition, there are no long steam-lines to keep up in the shaft, where it is difficult to reach them. All the water from this section of the mines is brought to the pumps, which are located near the boilers, through a canal or ditch, 10 feet deep, which tapers out to nothing about 200 feet from the pumps. It is so arranged that no inconvenience would be caused, should the water rise to the top of the canal. This gives immense storage-capacity; and should anything go wrong with the pumps or boilers, plenty of time can be taken to make repairs. The canal connects the sump with acres of old worked-out rooms, between which the room-ribs have been left standing to keep up the surface. Views of the boilers and pumps are given in Figs. 21 and 22.

The coke-plant consists of 806 ovens at Pratt Mines, 581 of which are near Slope No. 1, 225 at Shaft No. 1, and 250 at Alice Furnaces, Birmingham. They are of the bee-hive type, and, with the exception of a few, are 12 feet in diameter by 6 feet high in the clear. All these ovens are charged from "larries" containing one charge each, which run upon tracks of 5 feet gauge, extending the full length of the lines of ovens. At Alice Furnaces these larries receive the coal from a bin, supplied from the Pratt Mines railroad-cars, which, being placed on top of the bin, are dumped by means of the drop-bottoms. The larries at shaft No. 1 receive their coal from a bin which is supplied direct from the mine-cars. The plant of 581 ovens at slope No. 1 is supplied from a large new bin, having a capacity of 5000 tons, and supplied like the one at Alice Furnaces noted above. The bin is provided with 3 larrie-tracks extending its full length, thus permitting the coal to be drawn from the bins

into the larries at any point desired. The ovens are charged with 5 tons of coal, which produce 3 tons of coke, the yield of coke being about 60 per cent. After the ovens have been carefully levelled, the doors are closed up with brick and luted, space being left near the top to furnish sufficient air to carry on combustion. The drawing of the ovens is performed by men whose special duty it is. The coke produced from Pratt coal compares favorably with the typical Connellsville article, being but little higher in ash and other impurities, and, consequently, about as high in carbon. It has, to a great extent, the characteristic metallic ring, and stands without difficulty the burden in our highest furnaces (if it be admitted that heavy burden and high furnaces go together). Washing the coal would, no doubt, improve the quality of the coke; but whether this would not be offset by the additional cost is a question which will not be answered satisfactorily until it has been practically demonstrated. That it has been tried, although possibly not thoroughly, and abandoned, because it was thought the improvement in the coke did not justify the increased cost, is a well-known fact. The experiment of washing was tried by the Woodward Iron Company, which put up a substantial plant of about 500 tons capacity per day; but, after having operated it for one year, the company decided to resume the coking of its coal unwashed. Washing of Pratt coal has been tried also by the Coalburg Coal and Coke Company, which made several unsuccessful attempts. In other seams the Cahaba Coal and Railroad Company has tried washing and abandoned it because of the increased cost.

In many cases, it may be said, Southern coke has been condemned where it was not at fault. When a furnace gets to working badly, the furnaceman, to shield himself, will lay the blame on bad coke, without having any good reason for so doing. Time and again it has occurred that at the very time when the furnaces were not doing well, the coke was, notwithstanding, above the average; yet just then the complaints would be most prolific, clearly indicating that the signal for complaints was not bad coke but bad furnace-working.

The making of coke is not by any means an intricate problem, and there is no good reason why there should be bad coke made if the coal is mined properly and the coking-operation receives the attention it demands. When watering an oven, to put on no more water than is necessary, and to draw all of the coke out of the oven clean, having it re-charged as quickly as possible, thus keeping the oven hot, are the main points in making good coke.

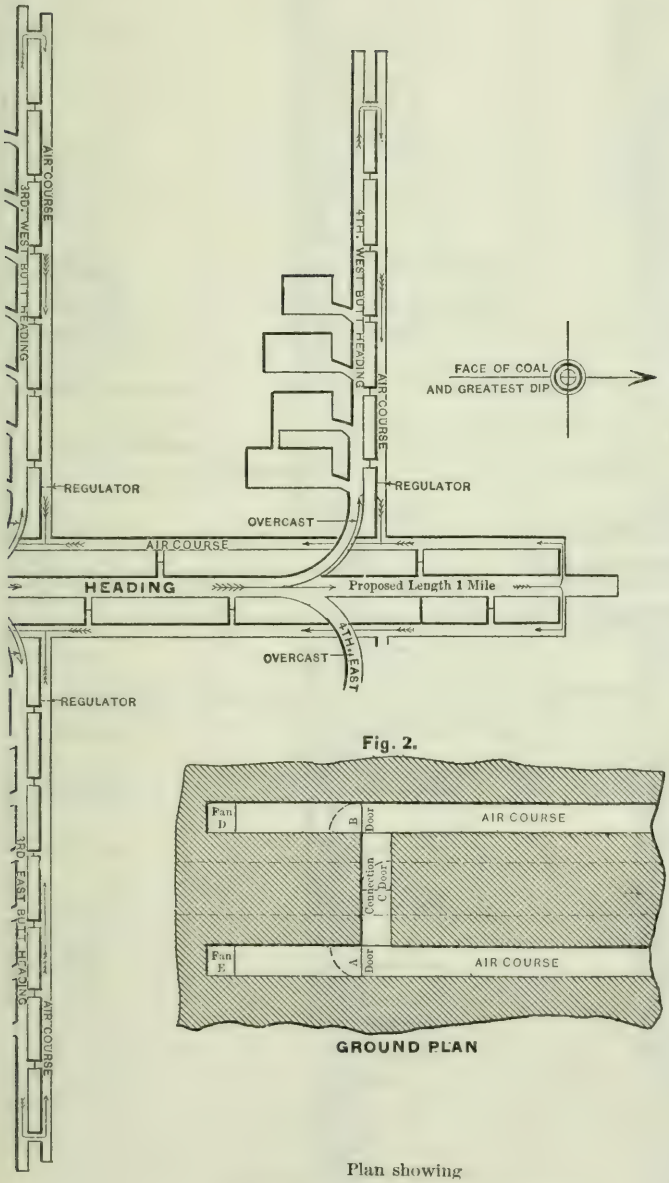


Fig. 2.

GROUND PLAN

Plan showing

VENTILATION and WORKINGS

of

SLOPES Nos. 4 and 5.

PRATT MINES DIVISION.

Scale, 1 inch = 200 feet.

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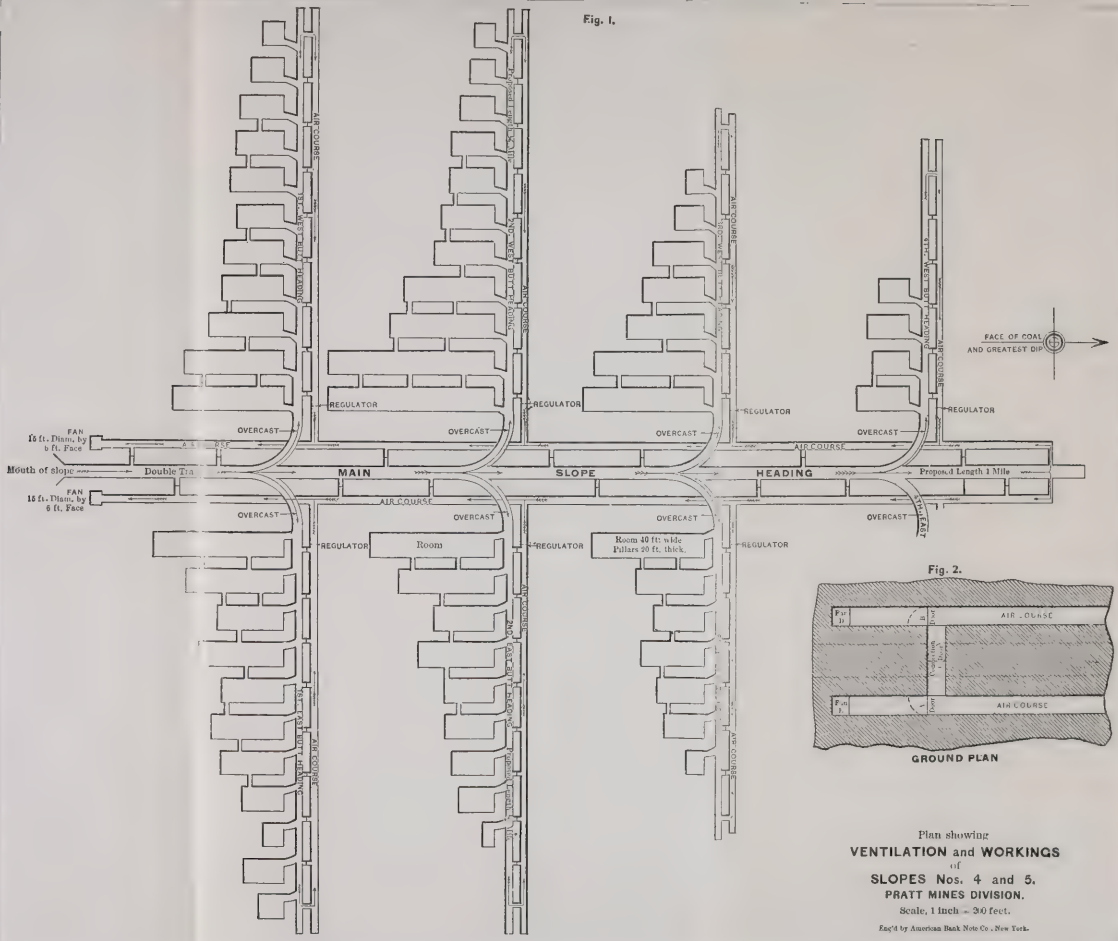


Plan showing
VENTILATION and WORKINGS
of
SLOPES Nos. 4 and 5,
PRATT MINES DIVISION.
Scale, 1 inch = 300 feet.

Eng'd by American Bank Note Co., New York.

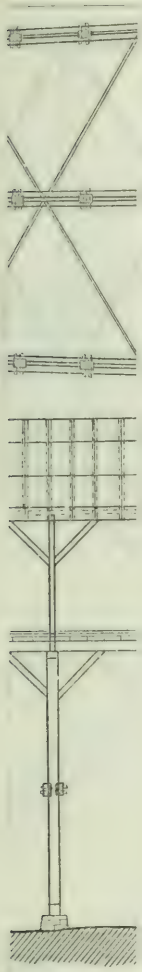
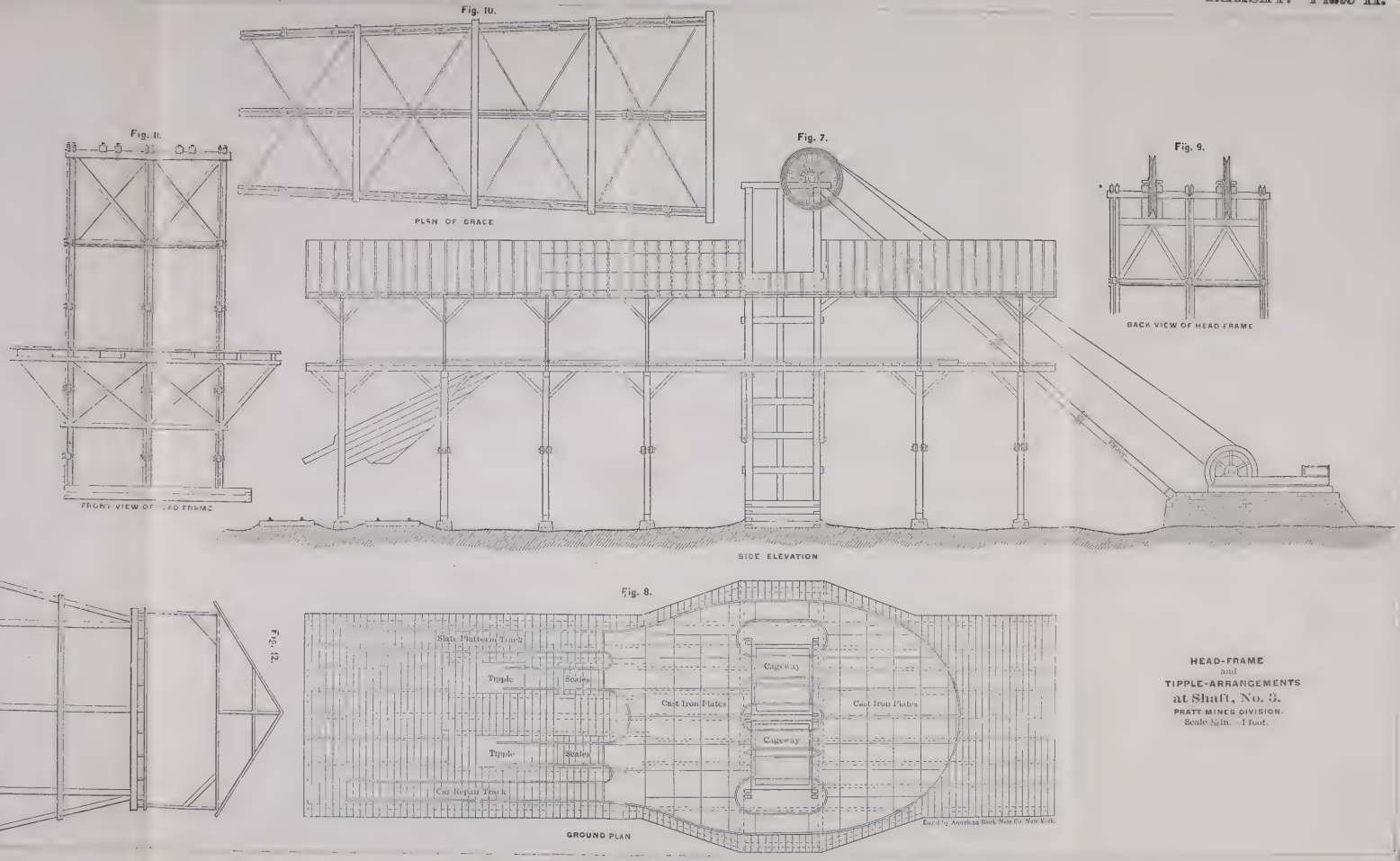
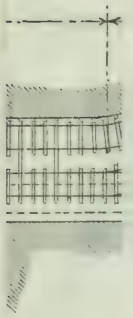
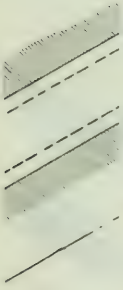


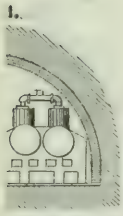
Fig. 8.



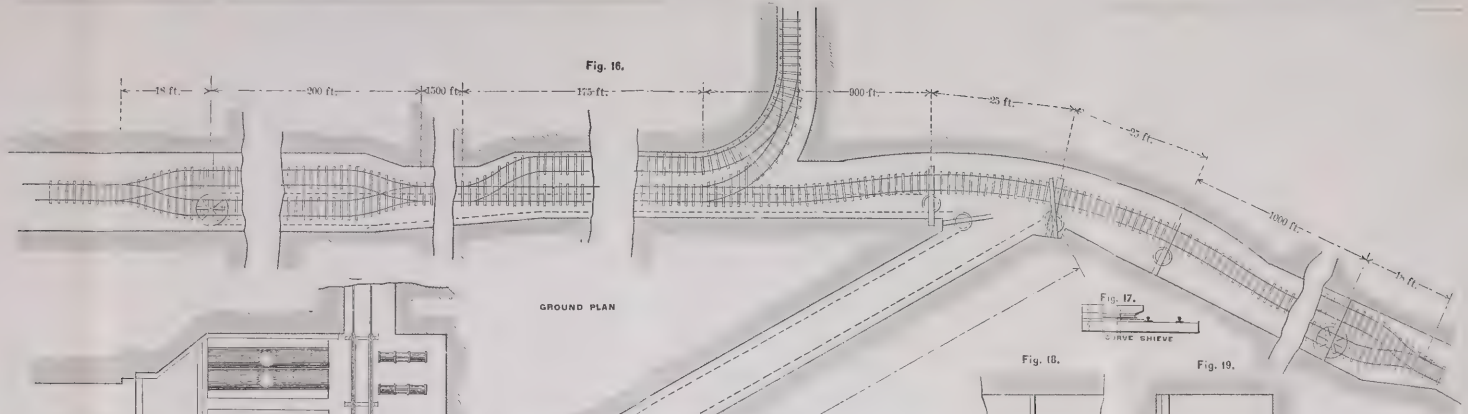
HEAD-FRAME
and
TIPPLE-ARRANGEMENTS
at Shaft, No. 3.
PRATT MINES DIVISION.
Scale ⅛in. = 1 foot.



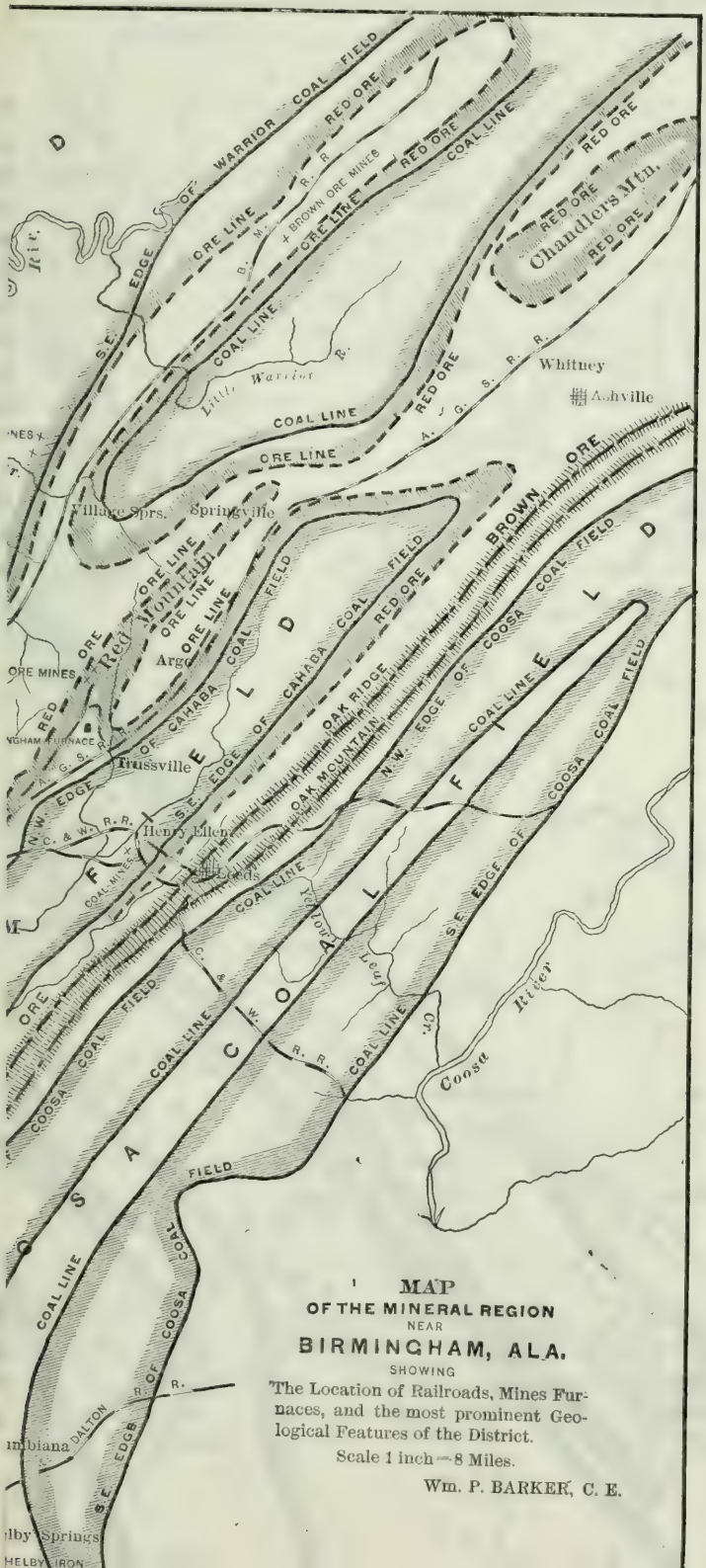


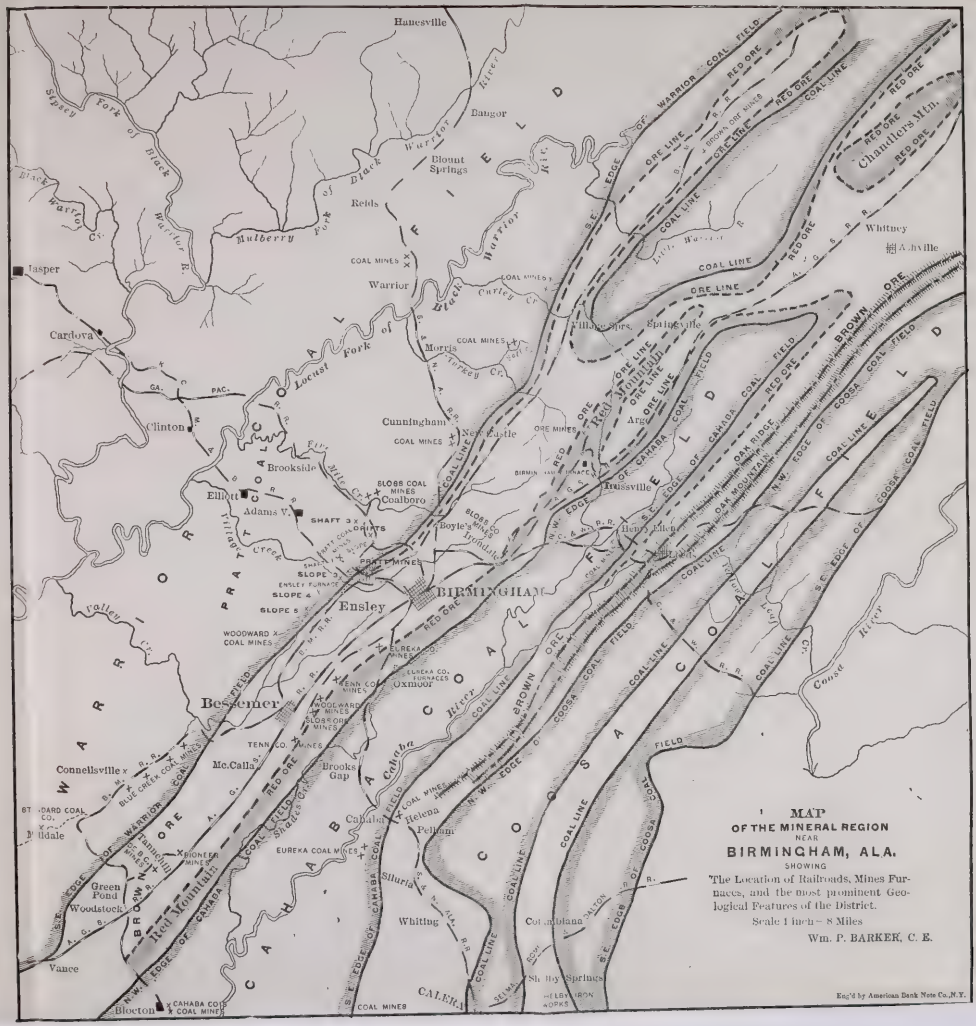


F BOILERS.



General Arrangement of
TAIL-ROPE HAULAGE,
BOILER and PUMPING-PLANTS
at SHAFT No 1.
PRATT MINES DIVISION.
Scale; 1 Inch—24 feet.


WARRIOR COAL FIELD
RED ORE
ORE LINE
COAL LINE
BROWN ORE
Chandlers Mtn.
Whitney
Ashville
Little Warrior R.
Village Spgs.
Springville
Red Mountain
Trussville
Henry Ellen
Leeds
Coosa River
Coosa R.
Big Canoe Cr.
Dalton
Rome
Shelby Springs
MAP
OF THE MINERAL REGION
NEAR
BIRMINGHAM, ALA.
SHOWING
The Location of Railroads, Mines Furnaces, and the most prominent Geological Features of the District.
Scale 1 inch = 8 Miles.
Wm. P. BARKER, C. E.



MAP
OF THE MINERAL REGION
NEAR
BIRMINGHAM, ALA.
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The Location of Railroads, Mines Furnaces, and the most prominent Geological Features of the District.
Scale 1 inch = 8 Miles
Wm. P. BARKER, C. E.

The coke made at Pratt Mines is mainly used at the Tennessee Company's Ensley plant of four 20-feet by 80-feet blast-furnaces, located about one mile from the coke-ovens. It is loaded into the Company's own 20-ton special coke-cars, hauled by its own locomotives over its own railroad, and delivered direct into the furnace stock-house. The coal from which the coke is made is in like manner moved from the mines to the coke-ovens in the Company's own rolling stock over its own system of railroads, making connection with all its mines. By a direct line of its own from Pratt Mines to Birmingham, the Company is enabled to ship coal, coke or iron by any line of railroad entering the city of Birmingham.

In the table below is given the output of Pratt Mines from 1883 to July 1st, 1890. The output for 1888 would have been, no doubt, some 75,000 tons greater, had the mines not been shut down for some time, owing to a strike.

Year.	Output—Tons.	Remarks.
1883	487,425	
1884	511,523	
1885	511,058	
1886	588,028	
1887	693,353	
1888	832,854	Strike.
1889	1,084,784	
First half of 1890	564,976	Half-year.

MACHINERY FOR THE CHARGING OF HEATING- AND MELTING-FURNACES.

BY S. T. WELLMAN, THURLOW, PA.

(New York Meeting, September, 1890.)

THE gradual change within the last few years from wrought-iron to steel for many purposes, notably for rails, plates, beams and other structural shapes, has brought the desirability and possibility of casting and rolling the steel in very large masses as compared with the

smaller piles worked and welded in the case of wrought-iron. Many advantages are gained in the way of economy by making steel ingots as large as can easily be handled. In casting the ingot it costs no more for labor to handle with a hydraulic or other power-crane an ingot or mould weighing 2 to 3 tons than if it weighed 500 pounds or less. The product at the blooming- or plate-mill is very much larger and the labor per ton very much less; and the saving in scrap in consequence of the long plates or blooms made from the heavy ingots, and the resulting smaller number of waste or crop ends, is a very important item. The improvement in quality, as a result of more work, is one of the most important effects of the increase in size and weight of ingots. Last, but not least, to the manager of the works, is the ease with which the product of any plant can be handled in and out of the pit in the case of say 3-ton ingots, compared with the trouble, delay and expense caused by the casting-pit if the same product has to go into 800- or 900-pound ingots. This can be fully appreciated only by the steel works manager who has been through the experience.

To get the full advantage of large ingots and consequent large product at the blooming- or plate-mill, proper apparatus should be provided for carrying the ingot into and taking it out of the heating-furnace. This part of modern rolling-mill machinery, it has seemed to the writer, has not kept pace with the rest, and it is the object of this paper to describe a machine which has been designed to do this work.

One of the most common methods of charging ingots into the furnace is by lifting them from the car on which they are brought into the mill by means of a long "peel," which is suspended in a crane, the opposite end being balanced and guided by a number of men. The ingot is swung into the furnace, rolled off of the peel by tilting it, and so on until the furnace is full.

Another method, and the most common one in this country for rail-ingots, is to push them from the car (which is brought immediately in front of the furnace-door) with a peel placed under and behind the ingot, a groove being made in the car immediately under the ingot for this purpose. The peel is pushed into the furnace by a chain which is fastened to the end of the peel, passing over a pulley on the car or in front of the furnace and back to a special hydraulic apparatus or crane arranged for the purpose. The ingot is generally pulled out of the furnace, with the same apparatus, by means of a pair of tongs fastened to the ingot, or an iron loop which is dropped

over it, and to which the pulling-out chain is fastened. In either case, the sand bottom of the furnace is constantly torn to pieces by the heavy ingot pulled over and ploughing through it, and a great deal of time and labor is taken up in keeping it in order.

The whole operation of charging and drawing, as commonly practiced, is slow and laborious, and takes a number of men to handle the tongs, peels, chains and other apparatus used.

The aim of the machine here described is to do all the work of taking the ingot from the car on which it comes from the converting- or melting-house, lifting it up, carrying it into the furnace and placing it wherever desired, dropping or lowering it in place, and, when the ingot is ready for rolling, lifting it from the bottom, bringing it out of the furnace, carrying it along in front of the furnaces and depositing it on the table in front of the rolling-mill. All the valves controlling the movements of the machine, as well as operating to open the furnace-doors, are worked by one operator, who rides on the machine.

In the accompanying drawings, Fig. 1 (Plate I.) is the plan of a machine for handling plate-ingots up to 5 tons weight; Fig. 2 (Plate II.) is the side elevation; Fig. 3 (Plate I.) is the longitudinal vertical section and cross-section of a furnace and car; Fig. 4 is a vertical cross-section through the tongs, *e e*, and front elevation; Fig. 5 is a vertical cross-section on line, 5 5 (Fig. 1); Fig. 6 is the rear elevation, and Fig. 13 (Plate V.) shows the general arrangement of furnaces and machine in relation to the mill.

The boiler, pump and engine for operating the machine are all carried on it. In Fig. 1, A is the boiler, B the hydraulic pump, C the engine for moving the machine along the track, and D the water-tank. The ingot is shown at E, and is grasped by the tongs. These are mounted on the end of the moving frame-work, *f f*, the forward end of which rests on the wheels, W W. These wheels are lifted, as desired, by the hydraulic cylinder, H, Figs. 3 and 4.

The rear end of the frame carries the wheels, W^1 W^1 (Fig. 3), which can move forward on the beams, *b b* (Fig. 1). This frame is moved in and out of the furnace, as desired, by the hydraulic cylinder, G (Figs. 1 and 3).

The machine is mounted on wheels, as shown, and runs on the broad-gauge track in front of the furnaces. It is moved by the small reversing steam-engine, C, which is geared to the axle of one pair of wheels, as shown by the dotted lines in Fig. 4.

The operator's seat is at O (Fig. 1), and the operating-handles for all the valves are within his reach.

The ingot is grasped by the tongs, which are closed by moving forward the piston of the cylinder, F (Figs. 2 and 3), thus forcing forward the plates, P P, carrying the rollers, r r. As the tongs are pivoted on the pins, p p (Fig. 1), this closes and holds the tongs on the ingot as long as the pressure remains.

The ingot being lifted from the car high enough to clear the fore-plate of the furnace, the operator moves the frame forward with the hydraulic cylinder, G, until the ingot is in the furnace, when it is moved to any desired place, and the tongs are lowered, opened and removed from the furnace, and are ready for another ingot.

The water is taken to the tongs-cylinder, F, on the moving frame, through the walking-pipes, x x, shown in Fig. 3.

In removing the ingots from the furnace, the operation of the machine is much the same. The ingot, after being removed from the furnace, is carried along by moving the machine on the track with the engine, C, and is deposited on the rolling-mill table, which extends out between the furnaces, as shown in Fig. 13.

As to the speed of the machine, the writer has frequently seen one ingot taken out of the furnace and carried to the mill-table, the machine going back to the furnace and charging three more ingots within one minute from the time the first door was opened.

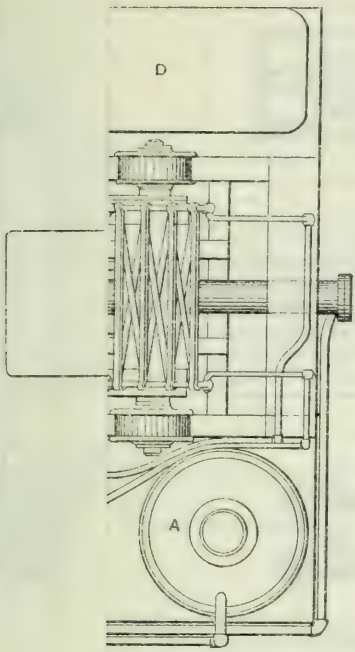
The whole crew to handle a product of 100 to 200 tons of ingots per turn, taking the ingots from the cars, putting them through the furnaces and on the mill-table, consists of a heater, a helper and the boy that runs the machine. Two Siemens furnaces heat the above product, the one heater attending to both.

In heating cold ingots, it has been found that the best way to get a large product is to heat the ingots to a red heat in one furnace, and then take them out and remove them to a furnace which is kept up to the full heat required for rolling, both furnaces being kept nearly full of ingots all the time.

This practice would only be possible with a machine capable of handling the ingots very quickly and with little labor.

An adaptation of a somewhat similar machine is shown in Figs. 7, 8, 9, 10, 11 and 12. It is designed, however, for charging the materials into open-hearth furnaces, the idea being to save not only time and labor in charging, but also one handling of all the materials that go to make up the charge.

Boxes made of plates are placed on small cars, and filled with the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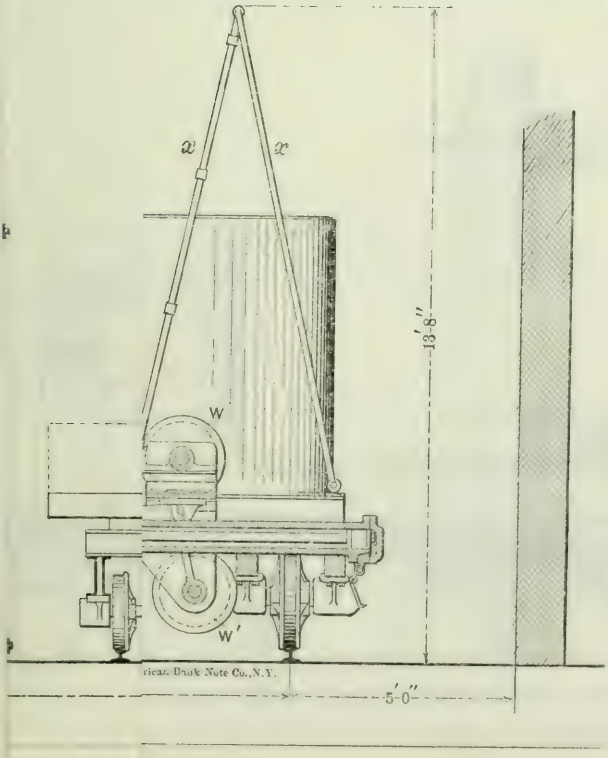
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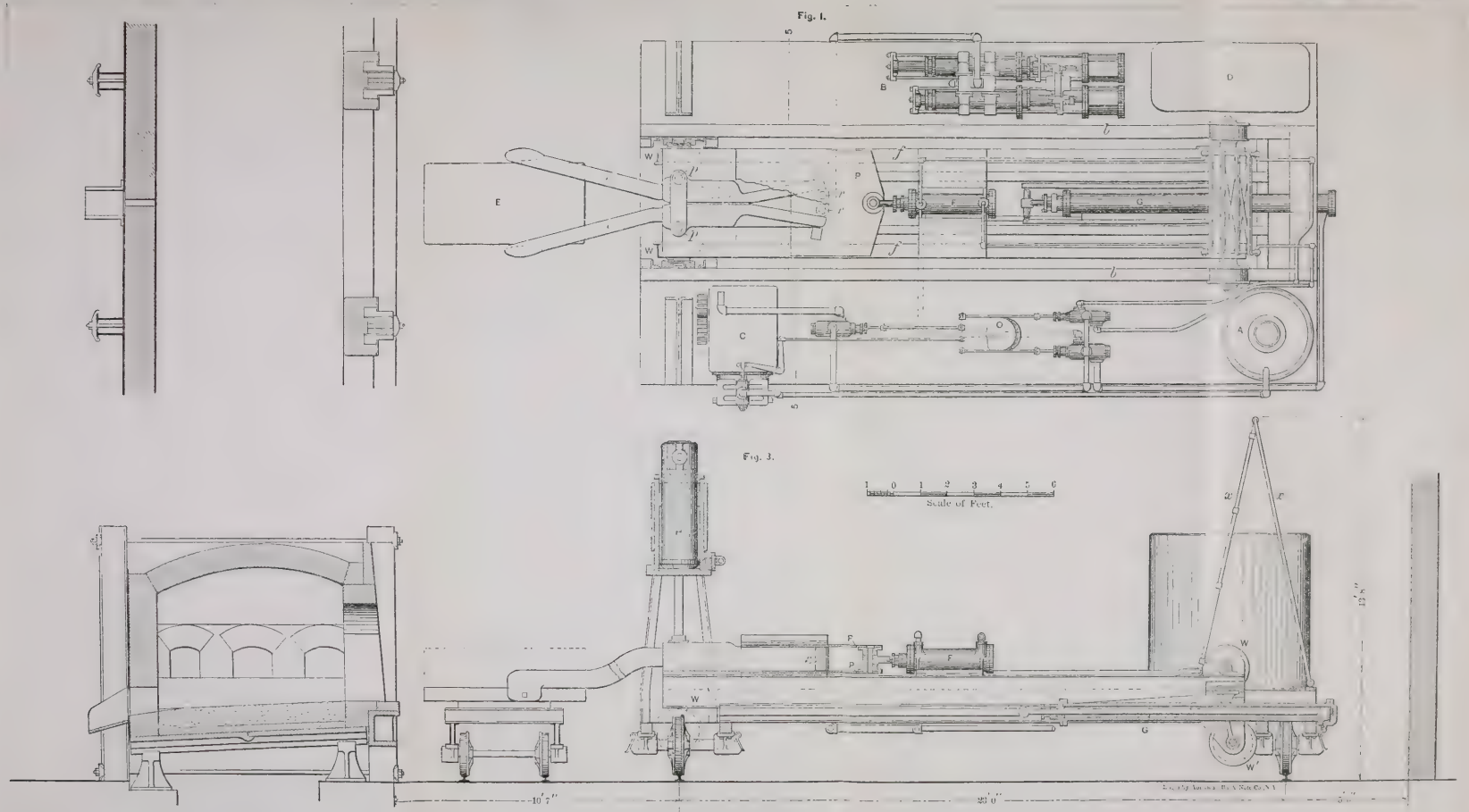


INGOT CHARGING MACHINE.

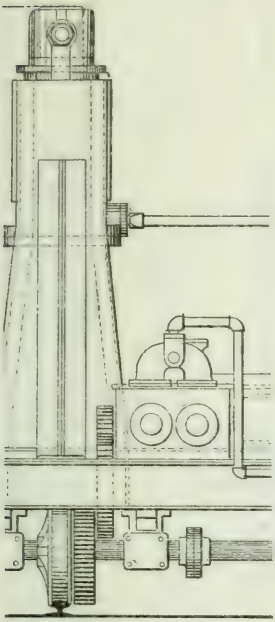


Fig. 6.

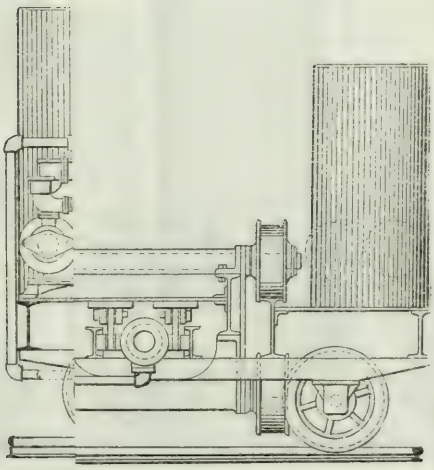


Fig. 2.

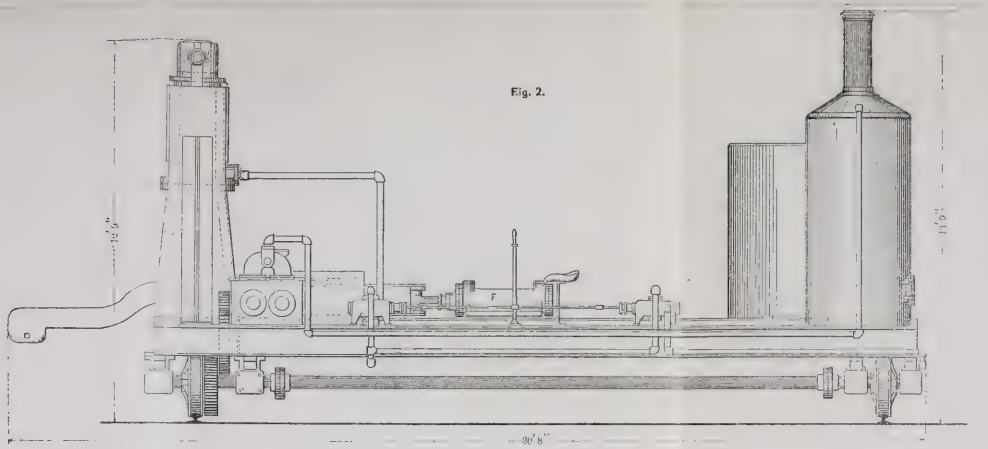


Fig. 4.

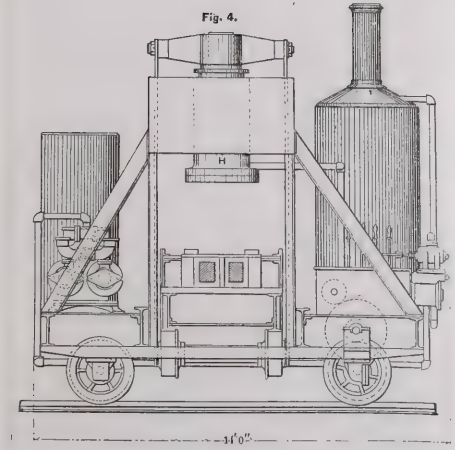


Fig. 5.

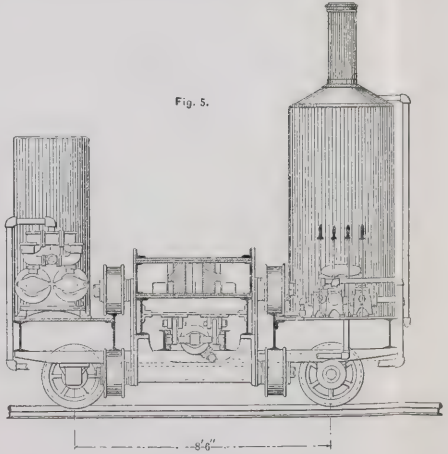
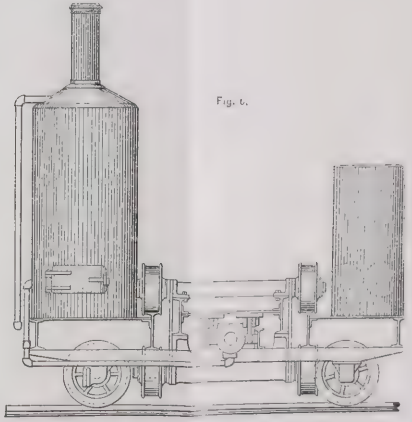
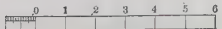
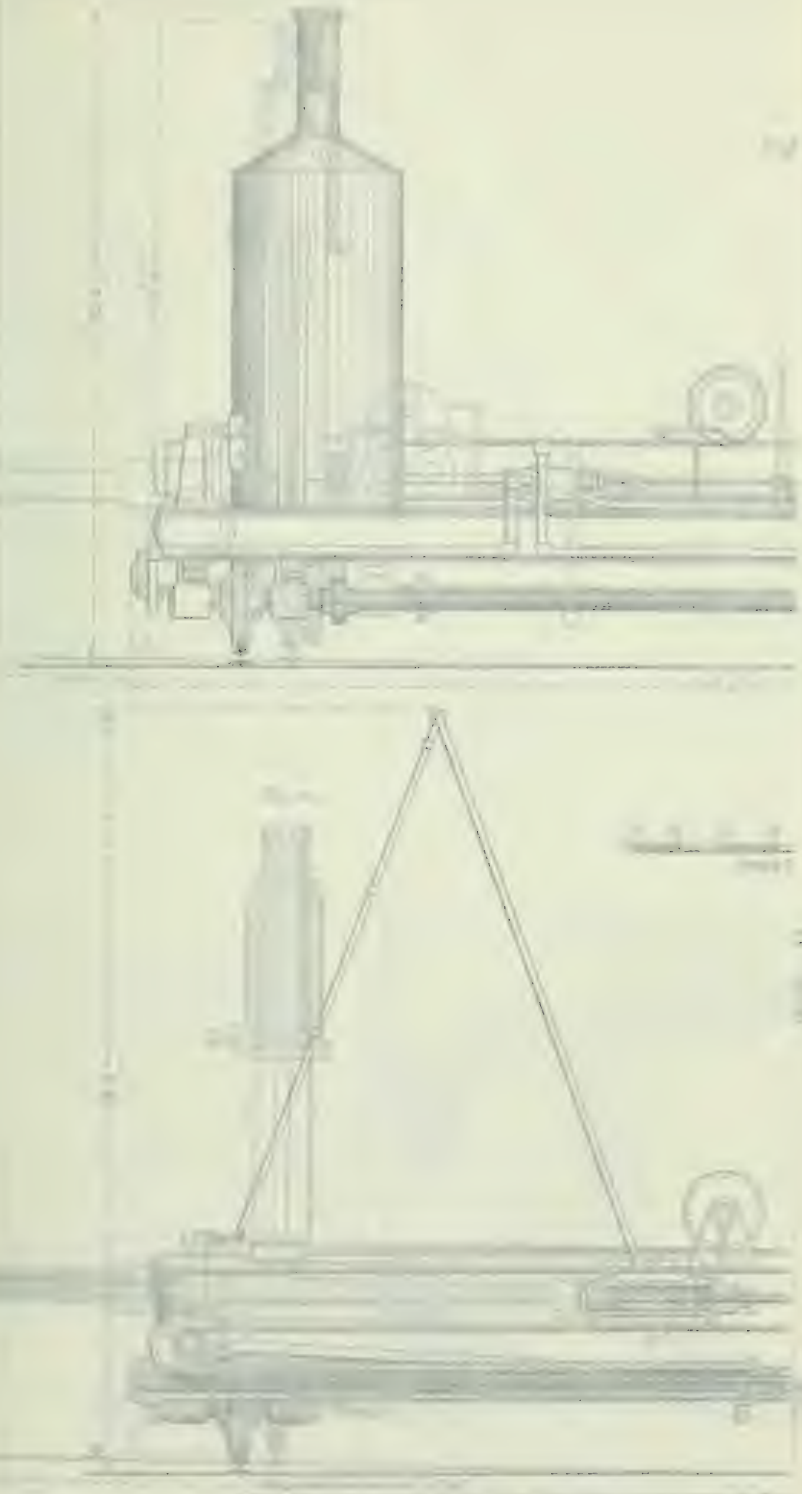


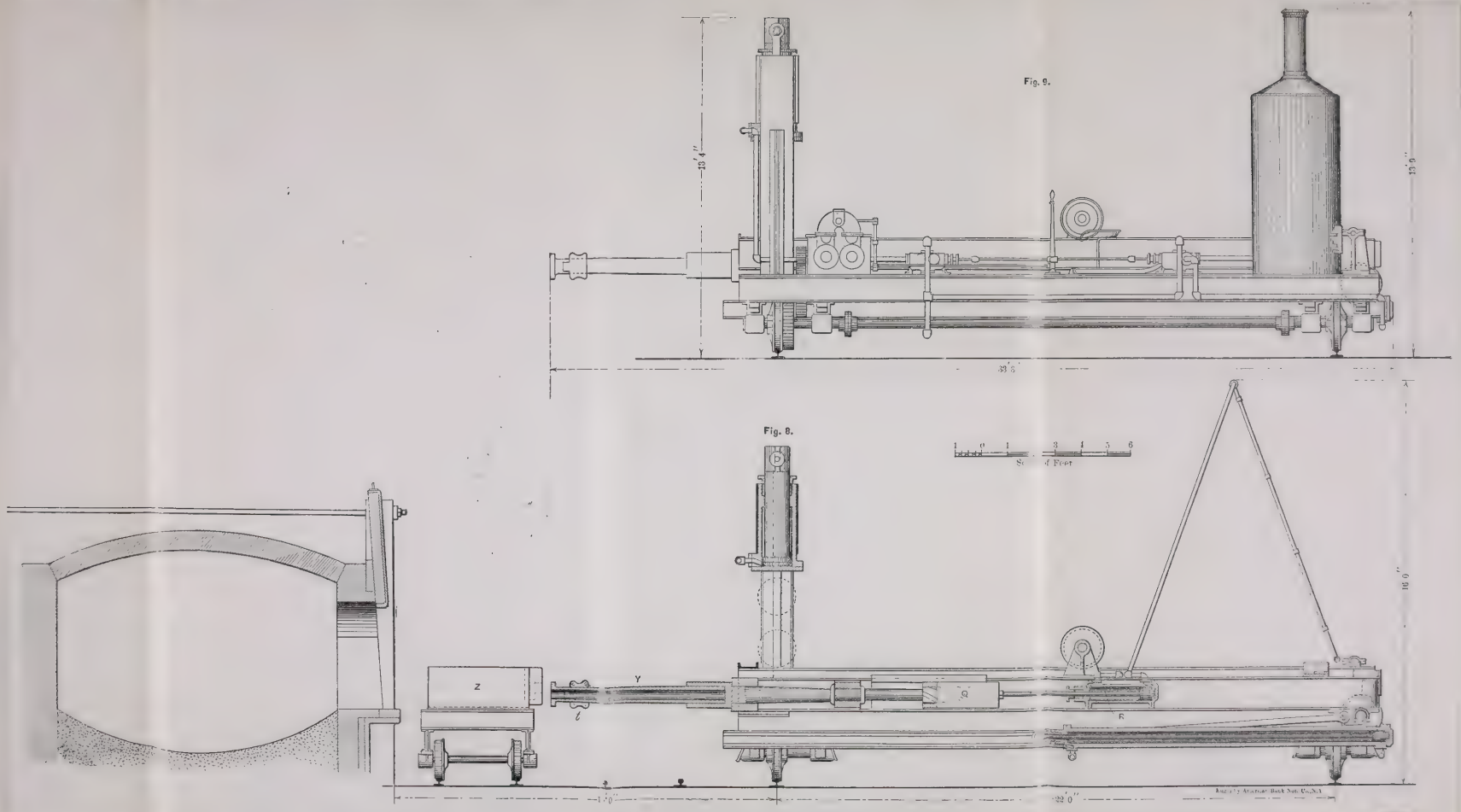
Fig. 6.



INGOT-CHARGING MACHINE.







MACHINE FOR CHARGING OPEN HEARTH STEEL MELTING FURNACES.

Fig. 10

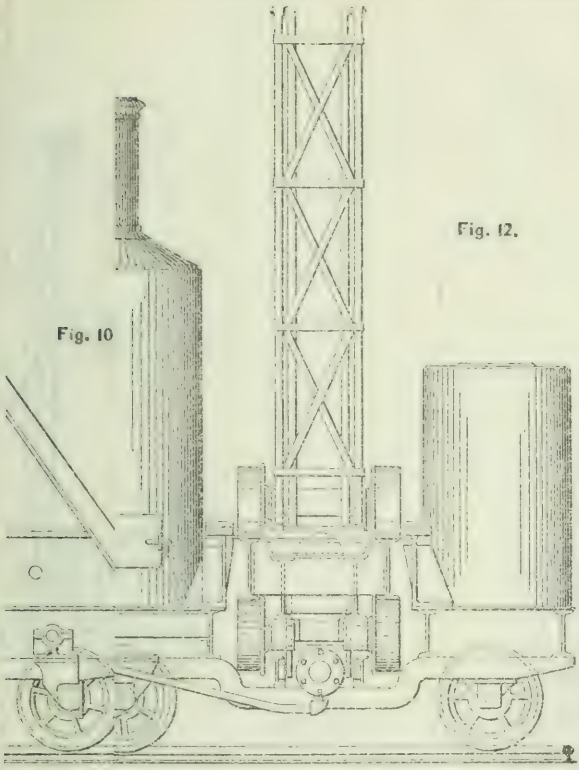
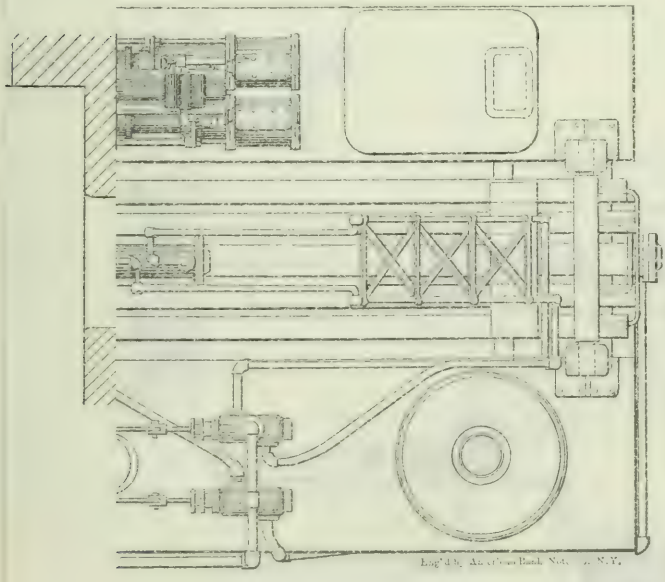


Fig. 12.



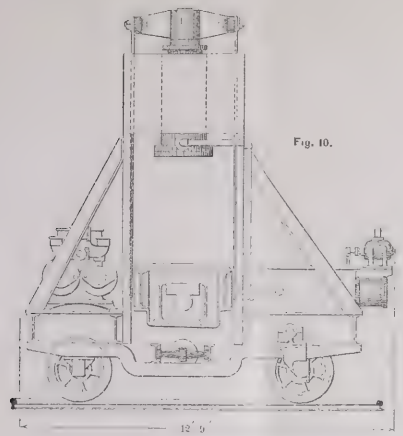


Fig. 10.

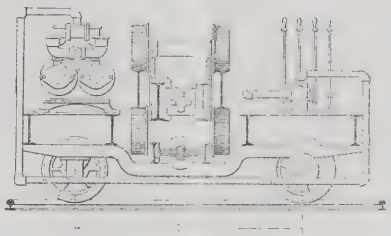


Fig. 11.

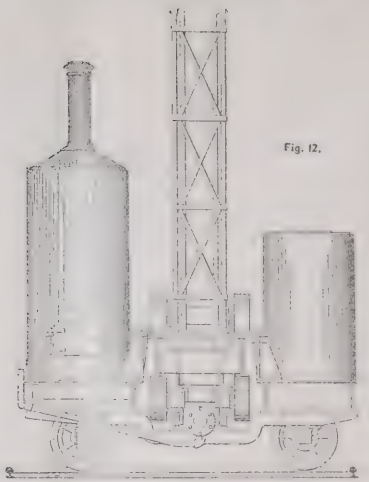
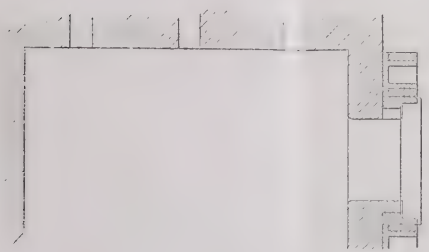
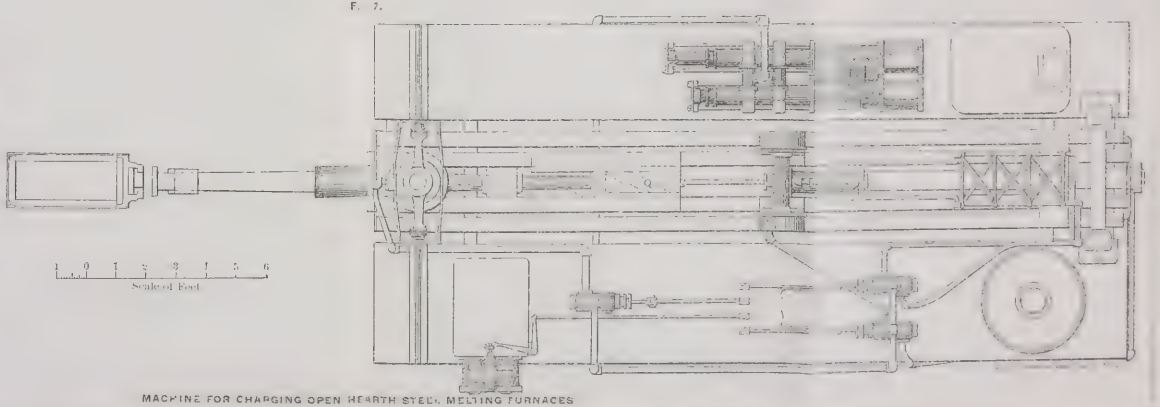


Fig. 12.

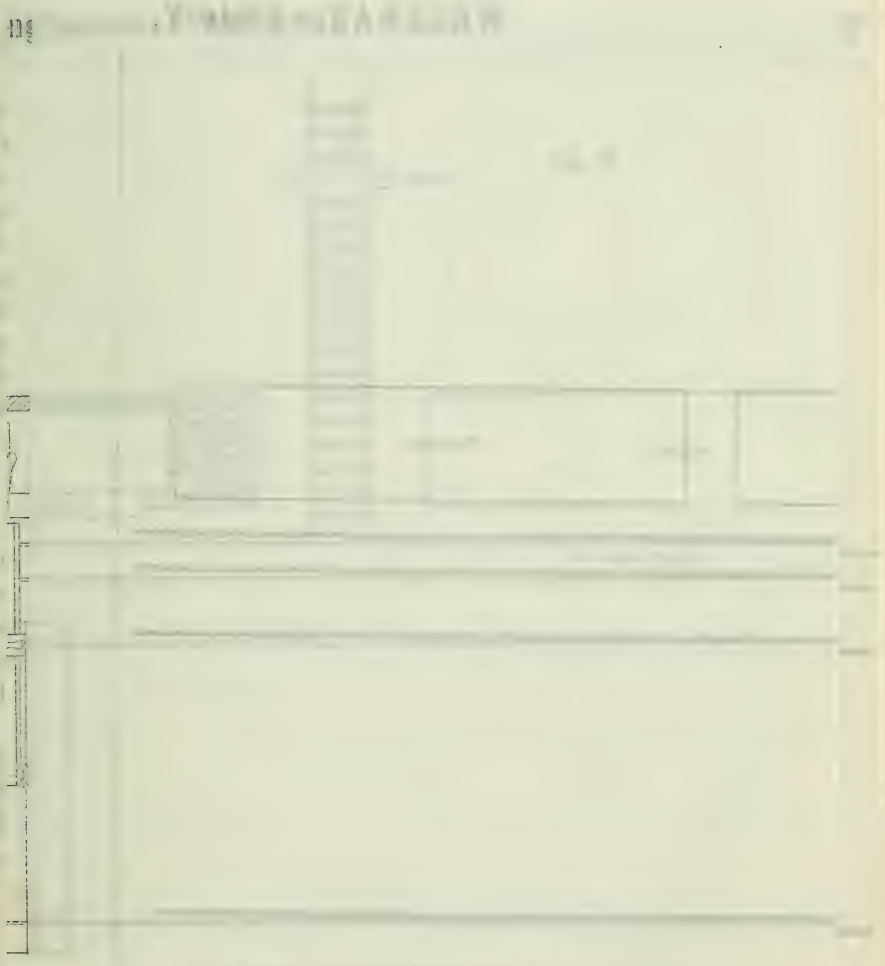




Scale of Feet

Fig. 7.

MACHINE FOR CHARGING OPEN HEARTH STEEL MELTING FURNACES



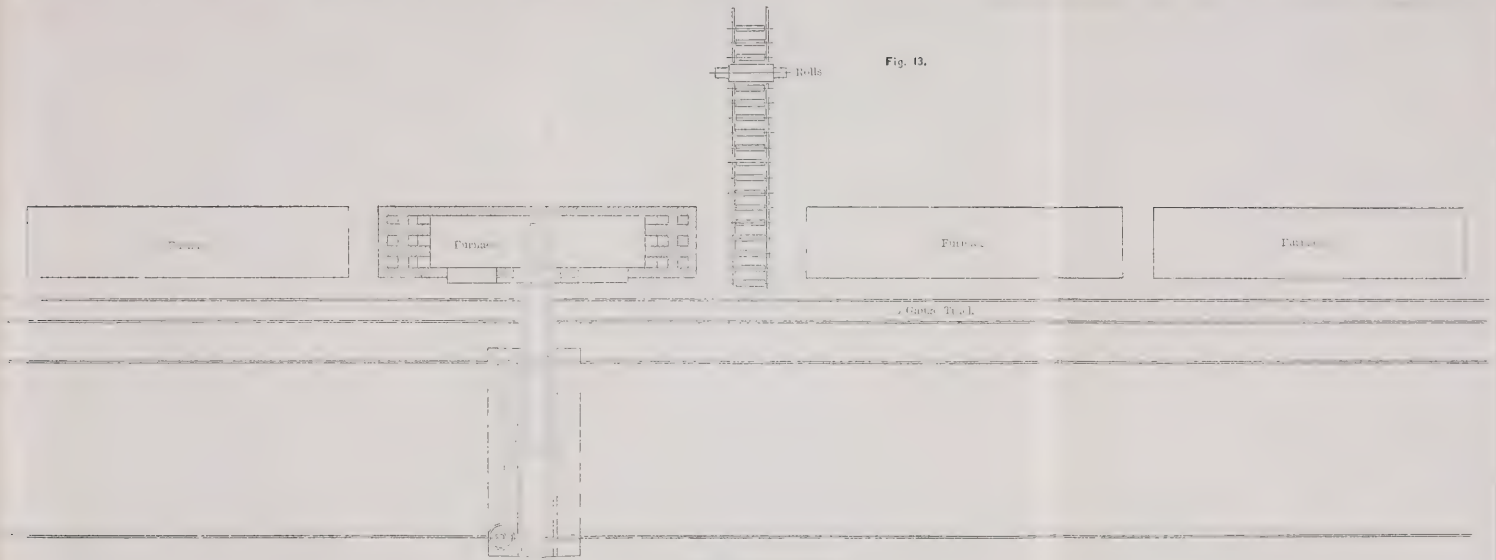


Fig. 13.

INGOT CHARGING MACHINE.

iron and steel at the stock-pile. The cars with the loaded boxes are then run into the melting-house in front of the furnace that is to be charged. The machine picks up the boxes, runs them into the furnace, dumps them and removes the empty boxes to the car, and so on until the whole charge is in.

Very little description of this machine is necessary, as its general features are the same as those of the ingot-charging machine.

The machine carries its own boiler, pump and engine. The tilting frame carries the T-headed bar, Y (Fig. 8), the head of which fits in the socket on the back end of the charging-box, Z, and is locked in place by the sliding block, *t*. The frame is lifted and moved forward in a similar manner to the ingot-charging machine, but the box is dumped, when desired, by the hydraulic cylinder, R, which is connected to the twister, Q. The whole operation of the machine is exceedingly simple. It is obvious that the full benefit of such a machine can only be obtained where several furnaces are in operation. As the time taken in charging a 15-ton furnace is only ten to fifteen minutes, one such machine is ample for several furnaces if they are arranged in line.

Discussion.

H. M. Howe, Boston, Mass.: The author speaks of putting the ingots first into a warm furnace, and then into a very hot one. I would ask if he prefers that to the German practice of using a *Rollofen*, *i.e.*, a rolling-furnace with a strongly inclined hearth—charging the ingots at the top or coal-end, and afterwards rolling them down toward the hot end?

S. T. Wellman: I do not think a *Rollofen* is applicable to very large ingots. It is difficult to turn over an ingot weighing four, five or six thousand pounds, and it requires too much manual labor. For small ingots, however, or for billets or blooms, the German system of which you speak is very good.

CAST-IRON TOOLS FOR CUTTING METALS.

BY OBERLIN SMITH, BRIDGETON, N. J.

(New York Meeting, September, 1890.)

The use of cast-iron tools, with chilled cutting-edges, for lathes, planers, boring-mills, etc., is not, as far as I can learn, very extensive in the United States, or perhaps in England and other parts of Europe. I have, however, no data to guide me in regard to foreign

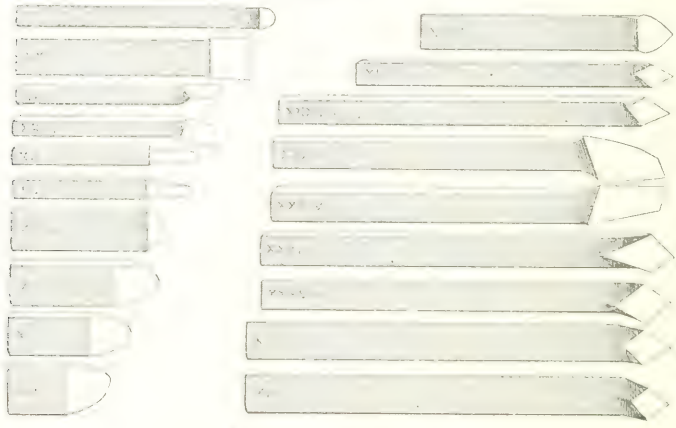
practice, and hope that some information on this point will be elicited from our transatlantic brethren.

In this country, the Pennsylvania railroad-shops at Altoona have, to the best of my knowledge, taken the lead in the use of such tools, and have very successfully employed them since they were introduced, some twenty years ago, by Mr. Peter Moore, who was then with the Pennsylvania Railroad Company.

They are in use to some extent at various other shops, including that under my own control; but there are no definite data at hand regarding their increasing employment.

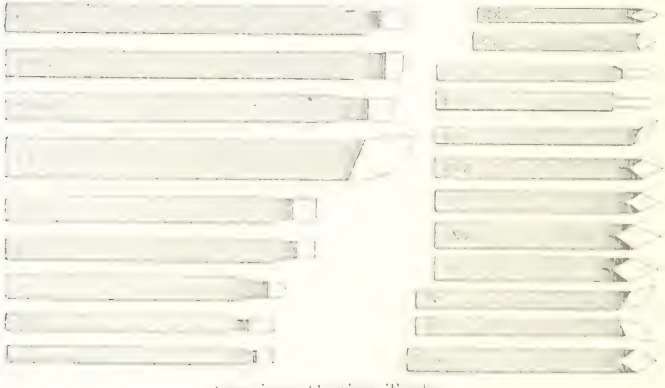
Mechanically, but little description of these tools is necessary. They are made to the ordinary standard shapes used for forged tools, as shown in the accompanying diagrams, Figs. 1 and 2, which are

Fig. 1.



Cast-iron cutting tools.

Fig. 2.



Cast-iron Cutting Tools.

copied from some kindly furnished to me from the Altoona shops. It will be noticed that the tools are numbered. The names, functions and dimensions corresponding to these numbers are given in the following:

List of Cast-Iron Tools.

Number.	Name.	Dimension, inches.
*I.	Round nose for iron and brass,	$1\frac{3}{8}$ by $\frac{5}{8}$ by $8\frac{3}{8}$
†I.	Thread tool for wrought and cast-iron,	$1\frac{3}{8}$ by $\frac{5}{8}$ by $8\frac{3}{8}$
II.	Diamond point, right-hand, for lathe,	$1\frac{3}{8}$ by $\frac{3}{4}$ by $8\frac{1}{2}$
III.	" " "	$1\frac{1}{2}$ by $\frac{7}{8}$ by $8\frac{1}{2}$
IV.	" side tool, right-hand, for lathe,	$1\frac{1}{8}$ by $\frac{5}{8}$ by 9
V.	" " left-hand, "	$1\frac{1}{8}$ by $\frac{5}{8}$ by 9
VI.	Round nose for lathe or planer,	$1\frac{1}{2}$ by $1\frac{3}{16}$ by $9\frac{3}{4}$
VII.	Square nose for planer,	$1\frac{1}{4}$ by $1\frac{1}{16}$ by $10\frac{1}{4}$
VIII.	" for lathe,	$1\frac{3}{8}$ by $\frac{3}{4}$ by 10
IX.	Round nose for lathe or planer,	$1\frac{7}{16}$ by 1 by $10\frac{5}{8}$
X.	Square " "	$1\frac{1}{2}$ by $\frac{7}{8}$ by $11\frac{1}{2}$
XI.	Diamond point for lathe or planer,	$1\frac{1}{2}$ by 1 by $11\frac{3}{4}$
XIII.	Square nose for lathe or planer,	2 by $1\frac{1}{8}$ by 15
XIV.	Diamond point for lathe or planer,	$1\frac{1}{2}$ by $1\frac{1}{2}$ by $15\frac{5}{8}$
XV.	Oyster-knife for planer,	$1\frac{1}{2}$ by $1\frac{1}{2}$ by $15\frac{1}{4}$
XVI.	Diamond point for planer or lathe,	$1\frac{1}{2}$ by $1\frac{1}{2}$ by $15\frac{1}{2}$
XVII.	" "	$1\frac{1}{2}$ by $\frac{7}{8}$ by $8\frac{1}{2}$
XVIII.	" for lathe,	$1\frac{1}{8}$ by $\frac{5}{8}$ by 7
XX.	Round nose "	1 by $\frac{5}{8}$ by 7
XXI.	Boring tool "	$1\frac{5}{16}$ by $\frac{5}{8}$ by $8\frac{3}{8}$
XXIII.	Left-hand side or facing tool for lathe,	$1\frac{1}{8}$ by $\frac{5}{8}$ by 8
XXIV.	Right " " "	$1\frac{1}{8}$ by $\frac{5}{8}$ by 8
XXV.	Square nose for planer,	$1\frac{1}{2}$ by $1\frac{1}{16}$ by $11\frac{1}{2}$
XXVI.	Diamond point for lathe,	$1\frac{1}{8}$ by $\frac{1}{2}$ by $6\frac{3}{4}$
XXVII.	" " or planer,	$1\frac{1}{2}$ by $\frac{7}{8}$ by $8\frac{1}{2}$
XXVIII.	" for planing and facing cylinders,	$1\frac{3}{8}$ by $\frac{15}{16}$ by $9\frac{3}{8}$
XXIX.	" for planer or facing-mill,	$1\frac{1}{4}$ by $1\frac{1}{4}$ by $9\frac{1}{8}$
XXX.	Square nose for turning cylinder-flanges,	$1\frac{3}{8}$ by $1\frac{3}{8}$ by 9
XXXII.	Side-tool for axle-lathe or planer,	2 by 1 by $14\frac{1}{2}$
XXXIII.	Diamond point for axle-lathe or planer,	2 by 1 by 15
XXXIV.	Side-tool " " "	2 by 1 by $14\frac{5}{8}$
XXXV.	Square nose " " "	2 by 1 by 15
XXXVI.	" " " "	2 by 1 by $14\frac{1}{2}$
XXXVII.	Diamond point for planer,	$1\frac{1}{2}$ by $\frac{7}{8}$ by $14\frac{1}{2}$
XXXVIII.	" for axle-lathe,	2 by 1 by 15
XL.	Tools for boring cylinders (Baldwin),	$1\frac{1}{2}$ by $1\frac{1}{2}$ by $5\frac{3}{4}$
XLI.	" " "	$1\frac{1}{2}$ by $1\frac{1}{8}$ by $4\frac{3}{4}$
XLII.	" " "	$1\frac{1}{2}$ by $1\frac{1}{2}$ by $3\frac{5}{8}$
XLIII.	Square nose for lathe,	$1\frac{1}{4}$ by $\frac{3}{4}$ by $6\frac{3}{4}$
XLIV.	Boring tools for driving-boxes,	$1\frac{5}{8}$ by $\frac{11}{16}$ by 6

* The lower of the two tools thus numbered.

† The upper of the two tools thus numbered.

For a given size of cut, the shank of one of these tools should be somewhat larger than in the case of a forging, in order to give the required lateral strength where it is fastened in the tool-post. Otherwise the shapes and sizes may be exactly the same.

In general, heavier cuts and probably somewhat higher speeds can be taken with these tools than with forged-steel ones, for the reason that there is no danger of drawing temper by the heat due to cutting-friction.

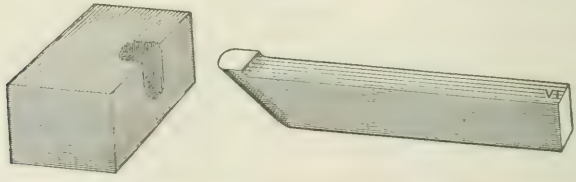
The experience of the Pennsylvania Railroad Company shows that, on the whole, these tools, cheaply made as they may be, are superior to steel tools for roughing-cuts, but that they are not desirable for finishing-cuts. This is because they will not hold a fine edge, on account of tiny pieces breaking off all along the cutting edge—thus leaving them too rough for producing a smooth surface upon the work. This breaking occurs to a still greater extent when the work is serrated, or otherwise broken into a non-continuous surface, as in the teeth of gearing, etc. The trouble in such cases is caused by the "back-lash," giving frequent and violent blows to the edge of the tool.

That, on the whole, they are superior tools for roughing-cuts upon wrought-iron, mild-steel, steel-castings, cast-iron and bronze, is shown conclusively by the circumstance that men working by contract elect to use them in preference to steel tools.

With the above conclusions my own somewhat limited experience entirely agrees, although we have been using the cast-iron tools but a comparatively short time at the shops of the Ferracute Machine Company.

The construction of these tools is obviously of the simplest description. An ordinary wooden pattern of the exact shape desired is molded in the usual way, with a small portion of its cutting-point in a cast-iron chill of about the relative size shown in Fig. 3. The

Fig. 3.



Cast-iron tool, and Chill for its Cutting-point.

exact proportions are, however, of no consequence. These tools, of course, cannot be repaired in the blacksmith-shop, but must be melted

up, when worn out by breakage or frequent grinding. The latter by the way, is best performed on emery-wheels rather than grindstones. Such tools, however, can be so cheaply recast that their maintenance, as well as their original cost, is much less than that of the ordinary forgings. The best composition of metal, as far as has been ascertained, is the same as for car-wheels, and no particular care is necessary in regard to the method of pouring, or the heat of the melted metal. The table given below contains some data regarding the standard metal at present used by the Pennsylvania Railroad Company for its car-wheels, and also for these tools:

Mixture of Iron and Steel-Rail for Making Cast-Iron Chilled Car-Wheels and Cutting-Tools.

Brand.	Kind.	Per cent. used of each brand.	Chemical Analysis.				
			C.	Mn.	P.	Si.	S.
Greenwood.....	Charcoal.....	5	3.61	0.41	0.43	0.74	.....
Lime Rock.....	"	10	3.60	0.85	0.33	0.98	.....
Richmond.....	"	5	3.52	1.09	0.28	0.92	.....
Shelby.....	"	5	3.61	0.23	0.46	0.70	.....
Raven's Cliff.....	"	5	3.89	0.27	0.16	0.50	.....
Glendon.....	Anthracite...	10	3.65	0.41	0.64	1.23	0.03
Steel-rail.....	" ...	5	0.40	0.85	0.09	0.03	0.03
Old wheels.....	" ...	55	3.50	0.52	0.42	0.75	0.08

I am indebted to my friend Mr. Theo. N. Ely, of the Pennsylvania Railroad Company, for the greater part of three trifling factors which enter into the composition of this paper, namely, the facts, figures and pictures. These, in answer to my numerous questions, he has so fully given me that it is with some hesitation I allow my name to appear as the author, instead of his. I take pleasure, also, in acknowledging the kindness shown me by Dr. C. B. Dudley and Mr. W. E. Hall, of Altoona, in sending me specimen-tools, and giving me information regarding the use thereof.

THE PAINT-ORE MINES AT LEHIGH GAP.

BY CONRAD E. HESSE, MOUNT CARMEL, PA.

(New York Meeting, September, 1890.)

To increase the durability of paint exposed to the weather, it is necessary to protect the oil with a substance that is itself unaffected by the elements. The so-called "metallic" paints, containing ochre,

possess this property, as the small particles of ochre are unaffected and penetrate the pores of the wood or iron, covering the oil and preventing it from evaporating. For the paint manufactured at Lehigh Gap another advantage is claimed, namely, that no artificial drier is needed in applying it, since it contains about twenty-eight per cent. of hydraulic cement, which hastens the drying and causes the paint to set, fitting it thereby for use in all out-door work.

The "paint-bed" worked at Lehigh Gap crops out along the southern border of Carbon county, about twenty-seven miles north of Bethlehem, in a well-defined ridge of Oriskany sandstone, known as Stony Ridge, north of and parallel to the Blue Ridge.

Fig. 1 shows the line of the outcrop. (In this illustration the lower half is the continuation eastward—to the right—of the upper half.)

The bed is of unknown extent, as it has been explored eastward only as far as Little Gap, and is but slightly developed west of the Lehigh river, though it has been traced to a point a short distance west of Pennsville.

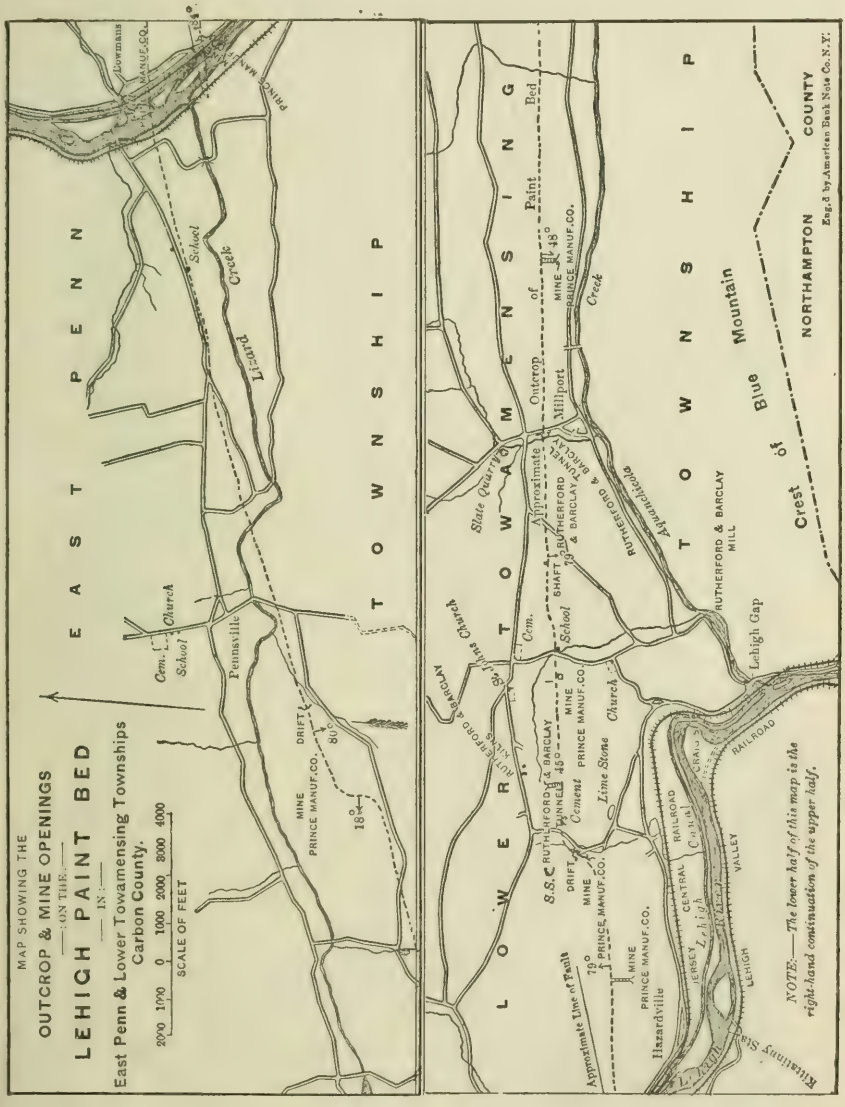
There appears to be some doubt as to the discoverer of the paint-ore bed. According to the Second Geological Survey of Pennsylvania, Mr. Robert Prince is credited with finding it in 1856, while exploring the lower portion of the Marcellus slate, in an effort to discover a good roofing-slate. He found an iron-ore, an analysis of which convinced him that it could be used in the manufacture of metallic paint. But, according to Mr. H. Rutherford and his Superintendent, Mr. Klein, Mr. Henry Bowman discovered the bed and sent samples of it to an iron-works in the vicinity, in order to determine its value for smelting. Its low melting-point rendered it worthless for metallurgical purposes, but the properties of some of the calcined material indicated its value for the manufacture of metallic paint.

Mr. Prince undertook to work the deposit, and the good prices which he obtained for the paint led several other firms into the business. Mr. A. R. Bass succeeded Mr. Prince after the latter's death, and manufactured "Prince's Metallic Paint." The principal firms now engaged are the Prince Brothers and Rutherford & Barclay, who control the paint-supply of this region.

Along the outcrop of the paint, the beds are covered by a cap, or overburden, of clay, and by the decomposed lower portion of the Marcellus slate, which is fifty feet thick at the Rutherford shaft.

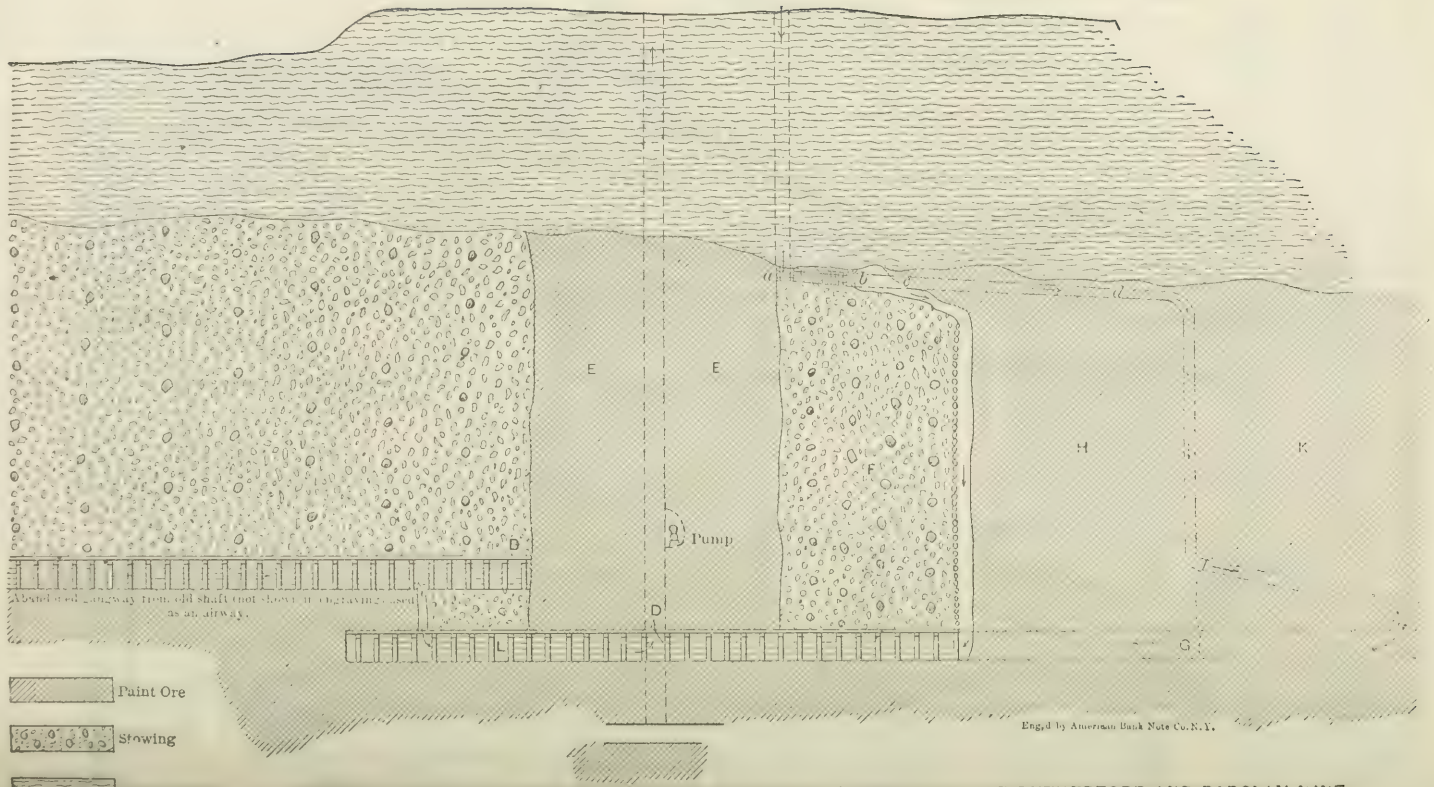
This slate is very hard and is quarried at a few places, notably near Millport.

Beginning with the Marcellus slate, the measures occur in the following descending order:



a. Hydraulic cement (probably Upper Helderberg), very hard and compact;

FIG. 2.



SECTION ALONG THE PAINT BED SHOWING METHOD OF WORKING AT THE RUTHERFORD AND BARCLAY MINE.

b. Blue clay, about six inches thick;

c. Paint-ore, varying from six inches to six feet in thickness;

d. Yellow clay, six feet thick;

e. Oriskany sandstone, forming the crest and southern side of the ridge. It is extremely friable and disintegrates so readily that it is worked for sand at many points.

East of the Rutherford shaft the sandstone forms the top-rock of the bed. This is due to an overthrow occurring between the Rutherford tunnel and shaft.

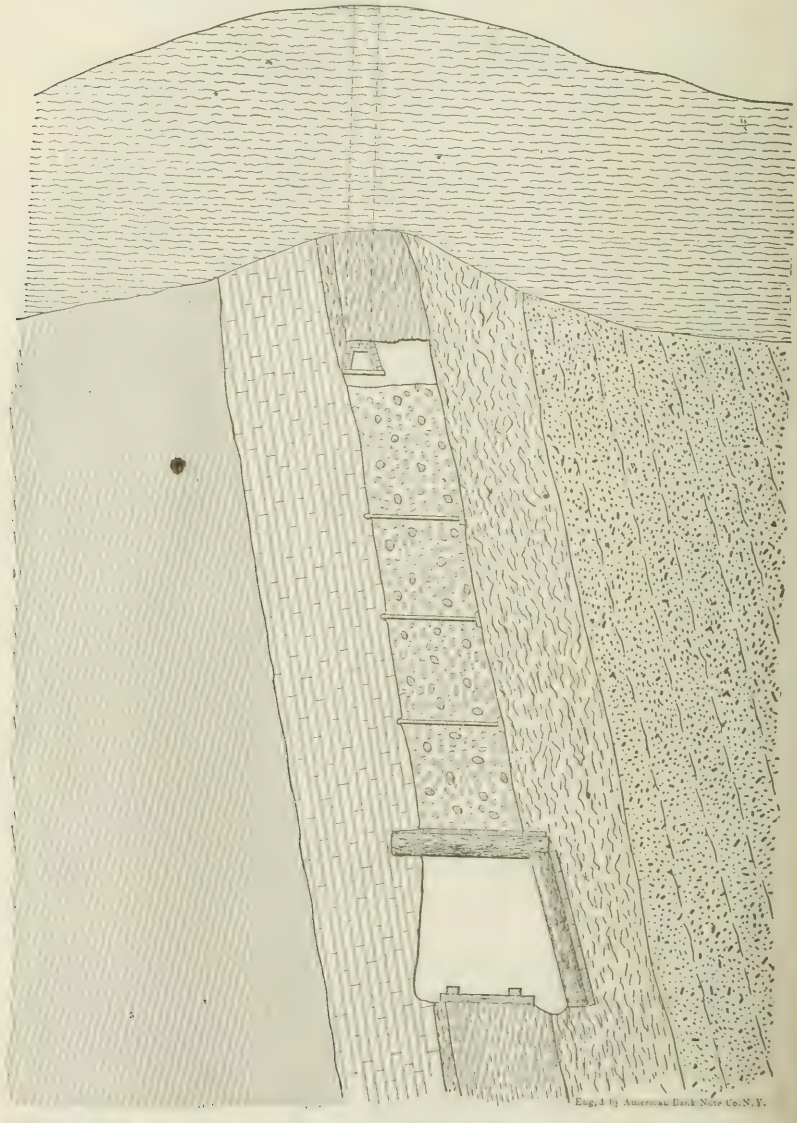
The paint-bed is not continuous throughout its extent. It is faulted at several places; sometimes it is pinched out to a few inches and again increases in width to six feet. A short distance south of Bowman's there is a fault striking northeast in the Marcellus slate, which has produced a throw of about two hundred feet. The measures dip from ten to ninety degrees. The dip at the Rutherford shaft is about seventy-nine degrees south, whereas at the tunnel it is forty-five degrees north. The ore is bluish-gray, resembling limestone, and is very hard and compact. The bed is of a lighter tint, however, in the upper than in the lower part, and this is probably due to its containing more hydraulic cement in the upper strata. The paint-ore contains partings of clay and slate at various places. At the Rutherford shaft there are fine bands of ore, alternating with clay and slate, as follows: sandstone (hanging-wall); clay; ore; slate; ore; clay; ore; clay; ore; slate; ore; cement; slate (foot-wall). These partings, however, are not continuous, but pinch out, leaving the ore without the admixture of clay and slate. Near the outcrop the bed becomes brown hematite, due to the leaching out of the lime and to complete oxidation. Occasionally, streaks of hematite are interleaved with the paint-ore. In driving up the breasts toward the outcrop, the ore is found at the top in rounded, partially oxidized and weathered masses, called "bombshells," covered with iron-oxide and surrounded by a bluish clay. In large pieces the ore shows a decided cleavage.

The method used in mining is a variation of panel-work. Nearly the same system of working is employed by all of the companies who have developed their mines either by means of tunnels or shafts. Tunnels are preferred whenever equally convenient, because they involve no expenses for pumping and hoisting machinery, fuel, repairs to machinery, etc.

The following description of the operation of the Rutherford mines is typical of all the workings in the vicinity.

The Rutherford tunnel is 6 feet high and 600 feet long. The gangways are driven along the foot-wall on the cement side, 6 feet high, and are heavily timbered and lagged at the top and on the

FIG. 3.



SECTION ACROSS THE BED, RUTHERFORD AND BARCLAY MINE.

clay side. The sets of timbers are 3½ feet apart, and usually of 9-inch timber. The width at the top is 3½ feet, with a spread of 5 feet at the bottom, the extra width being cut from the clay. Where the cement-rock is firm, the collar is hitched six inches into it and supported by a leg on the clay side. The cost of the timber is 54 cents per set, including the lagging. The monkey-gangway, which carries the air along the top of the breast from the air-shaft, is 2½ feet high, 1½ feet wide at the top, with a spread of 2½ feet at the bottom. Wooden rails with a gauge of 18 inches are spiked to the cross-ties. Fig. 2, representing a section along the bed, and Fig. 3, a section taken across the bed, show the method of timbering the gangways. The old shaft now used as an air-shaft, and the connecting gangway to B, Fig. 2, were driven some years ago; the ore above this gangway was taken out and the mine was abandoned. The shaft D was afterwards sunk and the gangway connecting with it was driven.

The gangway is not driven continuously, but after being driven about 55 feet on either side of the shaft, the breasts are started 25 feet from the shaft, a pillar, *E*, being left to protect it. The breast is then opened up to the face of the gangway, and when one ore-breast, *F*, is worked out, the gangway is driven ahead about 30 feet to *G*, and a new breast, *H*, is opened and worked out before commencing a third. The air-hole is first driven to the surface, then the breast is opened to its full width of 6 feet. The thickness of the bed of ore here varies from 4 to 6 feet, depending upon the thickness of the partings of clay and slate. The clay and slate are left on the bottom, which is left sloping to allow the ore to roll down to the shute, which is 6 feet wide and 4 feet long and heavily timbered. Small props or sprags are hitched into the cement and wedged with a lid on the clay side to prevent falls of rock.

The holes are drilled by hand in the clay-partings. They vary in depth from 1 to 4 feet, and the charge of dynamite is varied correspondingly, according to the amount of ore it is desired to throw down. The loose ore is wedged down with crowbars and picks, and is then freed from any adhering clay and thrown down the shute. It is there loaded into boxes holding about half a ton each, which are pushed to the shaft on a truck. The ore-boxes have four rings at the corners, to which are attached four chains, suspended from the wire hoisting-rope. At the top of the shaft the boxes are detached and placed on a truck, which is run to the dump. Thirty cars, averaging 15 tons, are extracted in a day of two shifts, the day-shift working nine hours and the night-shift eleven. The pay of the

miners is $1.25 per shift. The cost of mining the ore averages $1.75 per ton.

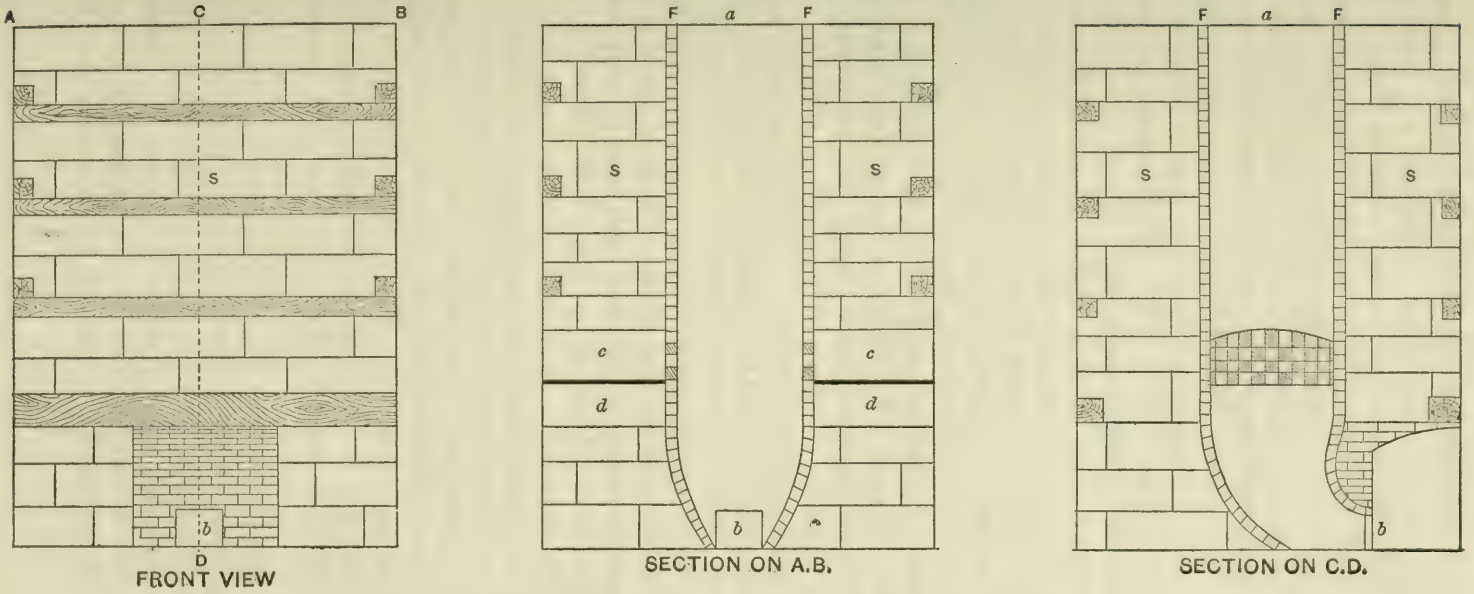
The single-drum hoisting-engine has a 14 x 8-inch cylinder, taking steam from a twenty-five horse-power boiler, which also supplies the pump. The average pressure is 80 pounds. The boiler consumes about 1 ton of coal per day. The boiler receives its supply of water from an iron tank, into which the water from the mine is pumped by a Cameron No. 8 pump. It discharges about 67 gallons per minute through a 3-inch column-pipe. The water is allowed to settle twenty-four hours before being used for making steam.

Natural ventilation is relied upon. The air-hole is timbered, and has an area of 2 by $2\frac{1}{2}$ feet. In every case the monkey-gangway, *a*, *b*, *c*, Fig. 2, is carried along the face from the air-shaft. The manner in which the third breast, *k*, would be ventilated, is shown by means of the lines, *a*, *b*, *c*, *d*, *e*, *f*, *g*. The stowing is packed so closely that there is little leakage after the roof settles. The air-shaft is the in-take, and, after passing through the breasts and gangway, the air goes, as shown by the arrows, to the hoisting-shaft, where it is warmed by the steam-pipes, and a strong upward current is caused. The east gangway, *L*, is ventilated by the old gangway leading from *B* to an abandoned shaft. The amount of air passing through the mine varies with the seasons, but is sufficient to keep the mine clear. When better ventilation is desired, a small blacksmith's blower is attached to the hoisting-machinery, and air is blown into the bottom of the shaft through a pipe. The blower was also used when the air-hole was driven.

The ore, as it comes from the mines, is free from refuse, great care having been taken to separate slate and clay from it in the working places. It is hauled in two-ton wagons to kilns which are situated on a hill-side, for convenience in charging. The platform upon which the ore is dumped is built from the top of the kiln to the side of the hill. The ore is first spalled to fist-size and freed from slate, and is then carried in buggies to the charging-hole of the kiln.

The slate, when burned, has a light yellowish color, which would change the color of the product. Fig. 4 represents a front elevation of the kiln and two sections at right angles to each other. The only kiln of the Rutherford and Barclay Company which I found in operation at the time of my visit is 22 feet high and 16 feet square on the outside. The interior is cylindrical, 5 feet in diameter, with a fire-brick lining of the best quality. The interior lining slopes from the

FIG. 4.



KILN USED BY RUTHERFORD AND BARCLAY.

S. Sandstone Casing. F. Firebrick Lining. *a*. Charging Hole. *b*. Door by which charge is drawn.
c. Fire place. *d*. Ash pit.

fire-place to the door by which the charges are withdrawn ; this facilitates the removal of the calcined ore. The casing is of sandstone, 5½ feet thick, and tied together with the best white-oak timber. When charged, a kiln holds 16 tons of ore, and the kiln is kept constantly full. The heat passes from the fire-places—of which there are two placed diametrically opposite each other—through a checker-work of brick into the center of the charge. The charge is withdrawn by a door in the front wall 2 feet long and 18 inches high. The fire is kept at a cherry-red heat, and about one cord of wood is burned every twenty-four hours. The kiln works continuously, calcined ore being withdrawn and fresh charges made without interruption. The ore is subjected for forty-eight hours to the heat, which expels the moisture, sulphur and carbon-dioxide. About 1½ tons of calcined ore are withdrawn every three hours during the day. The outside of the lumps of calcined ore has a light brown color, while the interior shows upon fracture a darker brown. Great care is necessary to regulate the heat so that the ore is not over-burnt. When this happens, the product has a black scoriaceous appearance, and is unfit for the manufacture of metallic paint, as it is extremely hard to grind.

The calcined ore is carried from the kiln in wagons to the mill, where it is broken to the size of grains of corn in a rotating crusher manufactured by W. F. Mosser, Allentown, Pa. The broken ore is carried by elevators to the stock-bins at the top of the building, and thence by shutes to the hoppers of the mills, which grind it to the necessary degree of fineness. Elevators again carry it to the packing-machine by a spout, and it is packed into barrels holding 500, 300 or 100 pounds each.

The power to run the machinery is derived from Aquanchicola creek. A dam gives a head of 8 feet to the 54-inch turbine which is geared to the machinery. Six runner-stones are used continually, one always standing idle, to be dressed. From 12 to 14 horse-power is required for each stone. The total cost of preparing the ore, including mining, transportation, burning and grinding, averages $5.75 per ton. The total waste is between 600 and 700 pounds in a long ton.

In concluding I desire to acknowledge my indebtedness to Mr. H. Rutherford, senior member of the firm of Rutherford & Barclay, for his courtesy in affording me every facility for observing the working of the mine and the method of preparing the ore. My thanks are also due to Mr. Wilson Klein, the Superintendent, and to the other employees, for the valuable information I have received from them.

THE PROGRESS OF GERMAN PRACTICE IN THE METALLURGY OF IRON AND STEEL SINCE 1876, *WITH SPECIAL REFERENCE TO THE BASIC PROCESSES.*

BY DR. HERMANN WEDDING, BERLIN, GERMANY.

(Pittsburgh International Session, October, 1890.)

Introduction.

It is now fourteen years since we German ironmasters, in considerable number, visited the United States on the occasion of the Philadelphia Exposition, and found the iron metallurgy of this country, as must be frankly confessed, in an advanced stage of development not previously suspected by us. At that time the United States already occupied, as now, the second rank, with Germany in the third, the product of pig-iron for that year being, in Great Britain, 6550, in this country, 2093, and in Germany, 1850 kilotons.* This relative position has not been changed. The present annual product of the three countries is, in round numbers, Great Britain, 8380, the United States, 7790, and Germany, 4530 kilotons. But the proportion of the United States, which was, in 1876, but 15 per cent. of the world's product, has now advanced to 30 per cent.

When we left America in 1876, it was not only with gratitude for cordial hospitality and for the liberality with which our professional colleagues here had opened to our inspection whatever was worthy of notice in our special field, but also with the conviction that we had collected a great treasure of new and interesting information which we could utilize for the benefit of our German Fatherland. Particularly the achievements of the never-to-be-forgotten Holley in the field of Bessemer practice, set us an inspiring example for imitation in our own country.

As an expression of our thanks in more practical fashion, I attempt in this paper to sketch what we have done in Germany since

* The kiloton is 1,000,000 kilograms, or (very nearly) 1000 tons avoirdupois, the metric ton of 1000 kilos weighing 2204.6 lbs. avoirdupois.

that time in the metallurgy of iron and steel, venturing to hope that some portion of what I bring may prove useful to our American brethren.

Germany (including Luxemburg, which is included in the Customs-Union with the Empire), produced, in 1876 and 1889:

	1876. Kilotons.	1889. Kilotons.
Iron-ore,	4,712	11,001
Coal and lignite,	49,550	84,892
Pig-iron,	1,846	4,525
Castings,	436	940
Malleable iron, Ingot-iron,	249	2,046
Malleable iron, Weld-iron,	943	1,655

I use the words ingot-iron (*Flusseisen*) and weld-iron (*Schweisseisen*) which are employed by us in official documents and in the custom-house, and the first of which practically comprises all that is called "steel." It is not my purpose to urge again the international harmony of nomenclature which was planned in 1876, but, unfortunately, not completely achieved. I would only remark that we Germans get along very comfortably with the names then proposed, which we adopted at once.

The figures given above may serve to prove that we have taken some pains to maintain our position as the third nation in the competition of the iron-producing world.

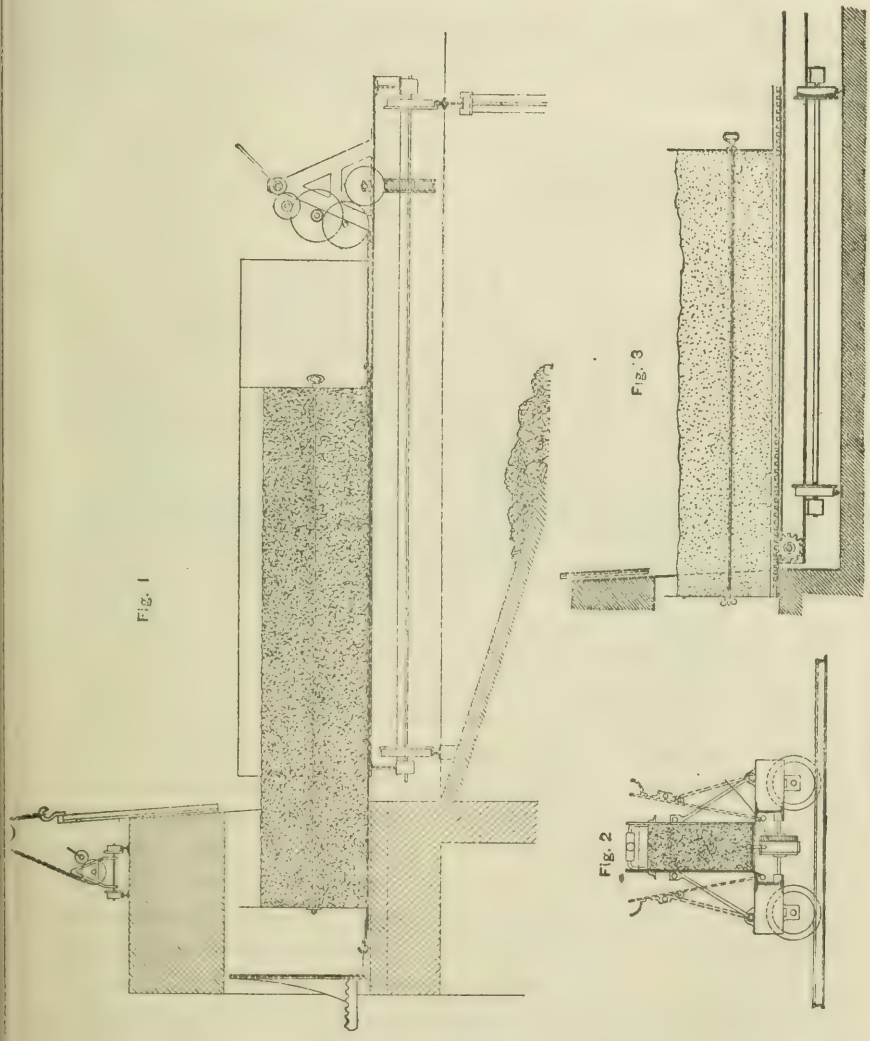
I. Ores and Fuels.

Like the United States, and unlike England, France and Belgium, Germany is fortunate in possessing within its own borders the raw materials for its iron industry. There is an importation of ores from Spain, Sweden and Hungary, it is true, but it is not strictly necessary, and, in fact, is more than offset by the exportation of German ores. In other words, we mine considerably more than enough ore for all the iron we make. Since 1876, indeed, the center of gravity of the German iron-ore production has moved. It now lies in Lorraine and Luxemburg, on the western edge of the Empire, where 57 per cent. of our ore-product is mined.

We possess very few ores that are free from phosphorus, and those which we have are so manganiferous that they cannot, generally speaking, be used to make pig-iron for the acid Bessemer process. On the other hand, we have an abundance of phosphoric ores, the greater part being so rich in phosphorus as to be suitable for

the manufacture of pig for the basic Bessemer or Thomas process. The remainder is composed of ores of medium tenor in phosphorus, and suited only to the manufacture of forge and foundry irons.

So far as the acid Bessemer practice is still maintained among us,



Quaglio's Apparatus for Charging Stamped Coal into Coke-Ovens.

it requires foreign ores; and this need is supplied chiefly from Spain.

The proportion among the different classes of ores in the product mined is about as follows :

Ores which can be used by themselves to make Thomas pig,	64.6
Ores suitable for forge and foundry irons,	22.2
"Acid Bessemer" ores,	2.8
Manganiferous ores (used to make white pig and spiegel),	10.4
	100.0

But if we add the foreign ores imported and deduct the domestic ores exported, we have as the proportional consumption of each class in Germany:

Ores for Thomas pig,	52.
" " forge- and foundry-irons,	22.
" " acid Bessemer pig,	15.
" " spiegel, etc.,	11.
	100.

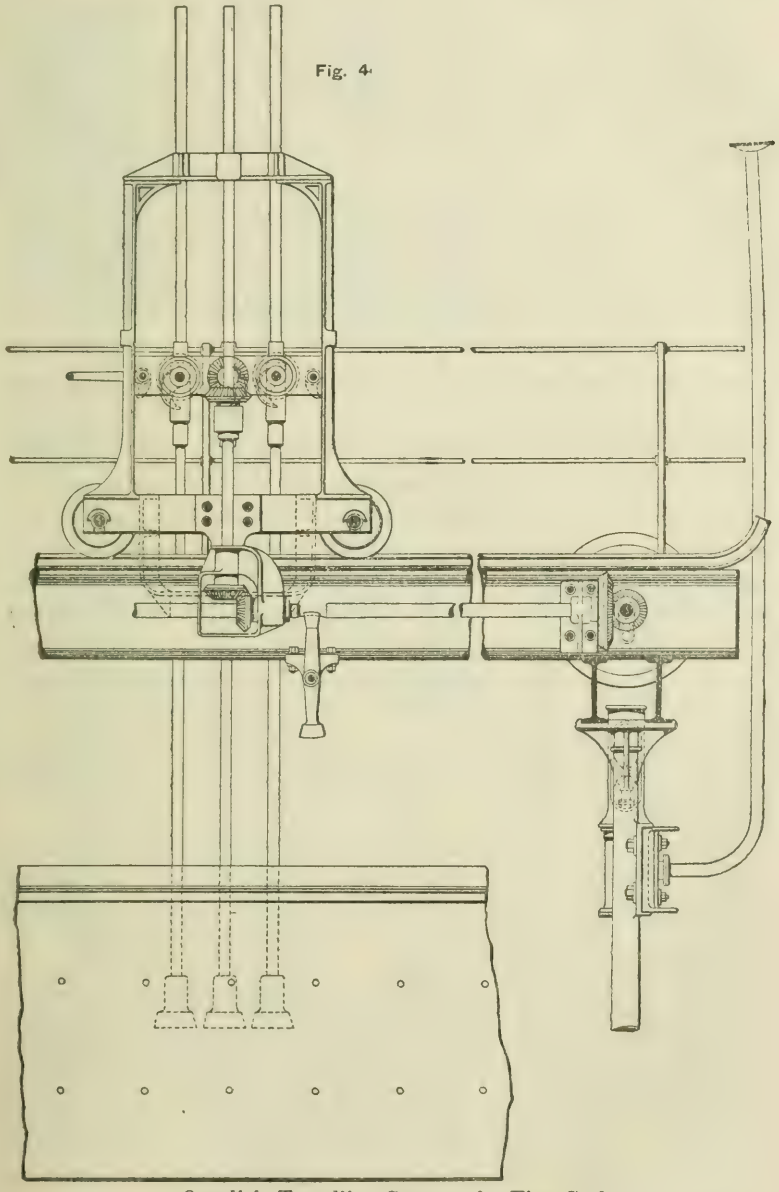
But this statement does not correctly represent the actual use of the different classes; for it frequently happens that commercial conditions make it advisable to use, in the production of one kind of pig, ores which would be, of themselves, better adapted to some other kind. (For instance, high- and low-phosphorus ores are often mixed to produce a forge-iron.)

As to our fuels, the figures already given show that we have an abundant supply for all industrial purposes, including, of course, the iron manufacture, for which, with rare exceptions, only bituminous coal and coke made from it are employed. Moreover, our coal-fields will not be exhausted in many centuries. Unfortunately, not all our coal-basins furnish the material for coke suitable for the blast-furnace. Strictly speaking, there is but one basin—the Westphalian or Ruhr coal-basin—which satisfies this condition to the satisfaction of the ironmaster. In other districts, as, for instance, that of the Saar, coke is indeed made, but of greatly inferior quality as compared with the Westphalian. In Upper Silesia, the coke made with the greatest pains from domestic coals would seem, nevertheless, to a stranger an utterly impracticable element in the regular conduct of a blast-furnace. This difficulty has caused in Germany, perhaps more than in any other country, the careful testing of all means of improvement, and the adoption of those which stood the test, in order that the quality of the coke made from poor coal might be made serviceable.

Dr. Otto estimates the coke-production of Germany at 26,313 kilotons, of which 13,401 should be credited to the Ruhr basin, 4168 to Upper Silesia and 2092 to the basin of the Saar.

The difficulty with German coking-coals is usually to be found, not so much in an excess of ash as in the lack of "coking-quality,"

Fig. 4.



Quaglio's Travelling Stamper for Fine Coal.

even when the ash is low. So far as the ash is concerned, an adequate

remedy has been found in careful washing after sizing. Our coal-washing apparatus leaves little to be desired now that the principle of a preliminary sizing has been fully recognized.

But, as already observed, some of our coals—especially, for instance, those of Upper Silesia—have already so small a percentage of ash, that washing will not sensibly reduce it, so that, although the preliminary crushing is performed as an advantageous preparation for the coking-process, it is not worth while to proceed to wash the crushed coal.

There is a decided advantage of crushing the coal before coking in the more intimate contact of particles thus secured. Experience has shown that the smaller the quantity of heavy hydrocarbons in the coal (these being the well-known determinants of its coking-quality) the closer must the solid particles be brought together in order to form a cohesive coke. To this end many means have been proposed and tried. The attempt has been made, without practical success, to press the coal together in the coke-oven. Again, the coal has been made into bricks, which, by reason of their regular shape, could be packed tightly together in the oven; but this practice also has failed of general adoption. Finally, resort has been had to the method of stamping the coal in a box, and charging into the oven the whole of the thick slab thus produced. This procedure has given satisfaction at German works, and is now in somewhat extensive use. The apparatus shown in Figs. 1, 2 and 3, from the German patent of J. Quaglio, No. 36,097, is used especially by Upper Silesian coke-makers (Julienhütte and Friedländer), who, however, everywhere stamp by hand instead of using a stamping-machine, as shown in Fig. 4.

In proportion to the amount of by-products obtainable by condensation from the gases of coking (which is not always in proportion to the total amount of the gases themselves), is the inducement to include in plants and practice the saving of such by-products. At many works tar and ammonia (which latter is usually converted at once into sulphate) are separated from the coke-oven gases by condensation. But this is not by any means the last word of progress in this direction. It has been found that, as in gas-distillations, so in the condensation of the coke-oven gases, the proper regulation of pressure and temperature permits the formation of a whole series of useful by-products. This system is now employed in ever-increasing extent, particularly in Upper Silesia.

The arrangement of such a condensing-works is shown in Figs.

5 to 14, from a communication of Dr. Otto, of Dahlhausen, to the Society of German Ironmasters. The ovens here represented have

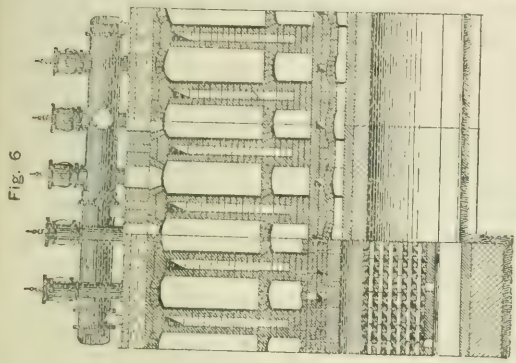


Fig. 6

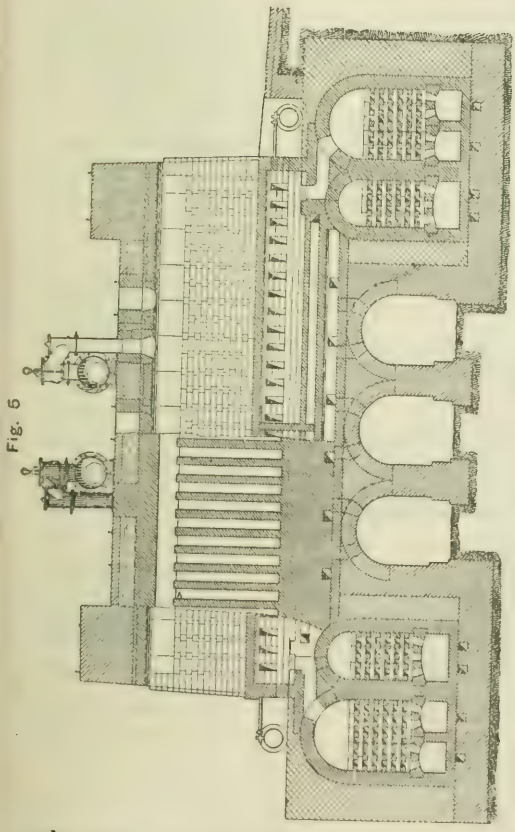


Fig. 5

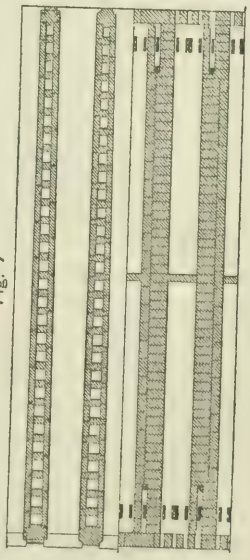


Fig. 7

Regenerative Coke-Ovens at Pluto Mine, Westphalia.

each a capacity of 5.75 metric tons of dry coal, and the coking re-

quires forty-eight hours. The gases which remain after the condensation consist essentially of light hydrocarbons, and are approximately free from oxygen and nitrogen, especially if the ovens are so well built and maintained that the entrance of atmospheric air into the coking-spaces is completely prevented. These gases are further utilized in heating—primarily the coke-ovens themselves, under the floors and between the walls of which they are conducted. But the gases are, after the condensation is over, so far cooled that they will scarcely ignite spontaneously, and once ignited do not continue to burn freely. Hence, either the gases or the air which is to burn them, or both, must be pre-heated. The practice had been introduced, at one time, of providing regenerators (see Figs. 5, 6 and 7) to be heated by the burned gases (after these, by their combustion, had heated the coke-ovens), and to heat in turn the freshly-entering gases and the air for their combustion. But it has been demonstrated, especially by the experiments of the well-known constructor of coke-ovens, Dr. Otto, of Dahlhausen, that this pre-heating of the gases may very well be omitted if the air is sufficiently heated. The resulting oven-construction is shown in Figs. 8 and 9, while Figs. 10 to 14 give the condensing-works, and Fig. 15 the general plan.

If the ammonia from the first condensation is too weak, it is enriched on gas-washers until the ammonia-water shows 3° to 3½° Beaumé.

The gases, the temperature of which in the floor-flues of the ovens is 1200° to 1400° C., and in the side walls 1100° to 1200° C., show, as a rule, at the foot of the heating-flues, over 700° C. still. When they finally escape, therefore, from the flues, they carry so much heat that they have been further used under steam-boilers with other fuels. Often (and especially in Upper Silesia) the quantity of gases is so great that a small part suffices to support the coking, while the rest can be otherwise utilized. Mr. Bremme, an engineer at Julienhütte, has succeeded in finding, after the separation of the gases, a very advantageous use for the latter portion in the heating of blast. The gases, which are free from dust and ashes and give a high calorific effect, are burned in brick heating-stoves (Cowper stoves). This has furnished a remedy for the difficulty encountered in Upper Silesia by reason of the large amount of zinc-dust in the blast-furnace gases usually employed to heat the hot-blast stoves.

II. Blast-Furnace Practice.

The conditions of the German blast-furnace practice vary greatly. The native ores are nowhere nearly as rich as the more important

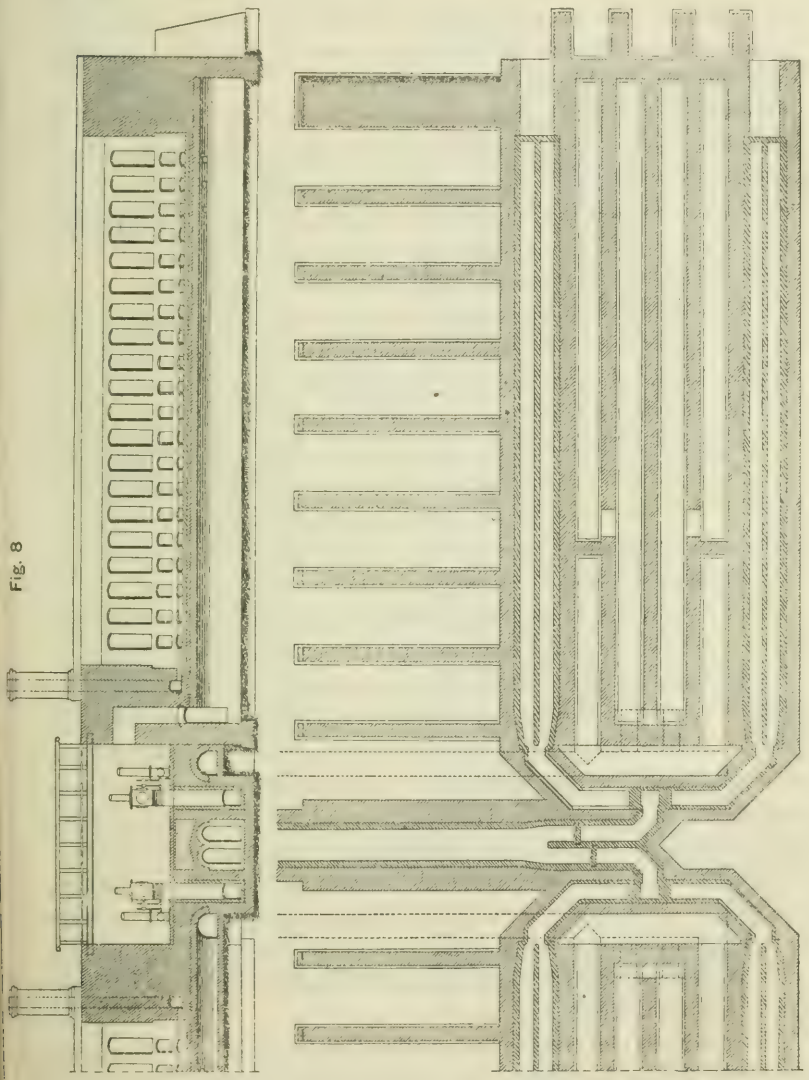


Fig. 8

Section and Plan of a Double-Bench of Coke-Ovens with Condensers.

ores employed in America; for instance, the specular ores of Lake Superior and the magnetites of the Northeast. Perhaps there is no other district in the world where the furnace-charge carries so little iron as in Upper Silesia, where it sometimes falls below 30 per

cent. It is not merely the low tenor of the ores in iron, but also its usual accompaniment, a high tenor in silica, which causes this character of charge. Concentration of the ores is, with a few exceptions, impracticable financially. The finely-divided condition of the ore in the gangue prevents any separation by gravity without a previous fine-crushing; and this, again, would introduce new difficulties into the working of the blast-furnace.

One disadvantage we have in common with the ironmasters of the United States—the necessity, namely, as a general rule, of railway-transportation over considerable distances for ores or coke, or both. But we lack the magnificent water-ways, like the great lakes of the North; and the development of internal canals, as a remedy for this natural disadvantage, is still delayed in Germany.

To give instances of our transportation of materials: most of the iron-ores smelted in Westphalia (where, as already observed, the best coke is made), must be brought from the Siegen district, from the Lahn and from Lorraine. On the other hand, notwithstanding the abundance of iron-ore in Lorraine, the ironmaster there must import not only Westphalian coke, but also (by reason of the sulphury character of the native ores) manganiferous ores from the Lahn as an admixture. And in all this transportation there is no water-route except a short piece of the Rhine.

The most favorably located furnaces, with reference to the nearness of the raw materials, are those of Upper Silesia.

In blast-furnace construction, practice inclines more and more to the entirely free stack without exterior masonry (*Rauhgemäuer*). Figs. 16 (Plate I.), 17, 18, 19 and 20 from the Friedenshütte in Upper Silesia, and Figs. 21 and 22 from Hörde in Westphalia, show the general proportions and arrangement. Lürmann's cinder-notch is universally used. Generally the furnace is closed in front, as a crucible-furnace (Fig. 16). The old-fashioned fore-hearth (*Sumpfofen*) occurs, however, when (in the production of Bessemer or Thomas pig) casts are to be made at brief intervals. But in this case also the cinder-notch is used, since the interspace of the fire-hearth is entirely closed by tamping (see Fig. 23).

The removal from the furnace-walls of the weight of the tunnel-head, bell, etc., is becoming more and more complete. The closed top (for lump-ores, usually the Parry cone, shown in Fig. 24, from Lorraine; for fine ores, the Langen bell-charger, frequently provided also with an interior suspended gas-pipe, contracting downwards, as shown in Fig. 21), is always hung from the top platform

and carried by special supports, mostly columns of iron pipe. The closed top with interior suspended pipe at the Hörde furnaces does

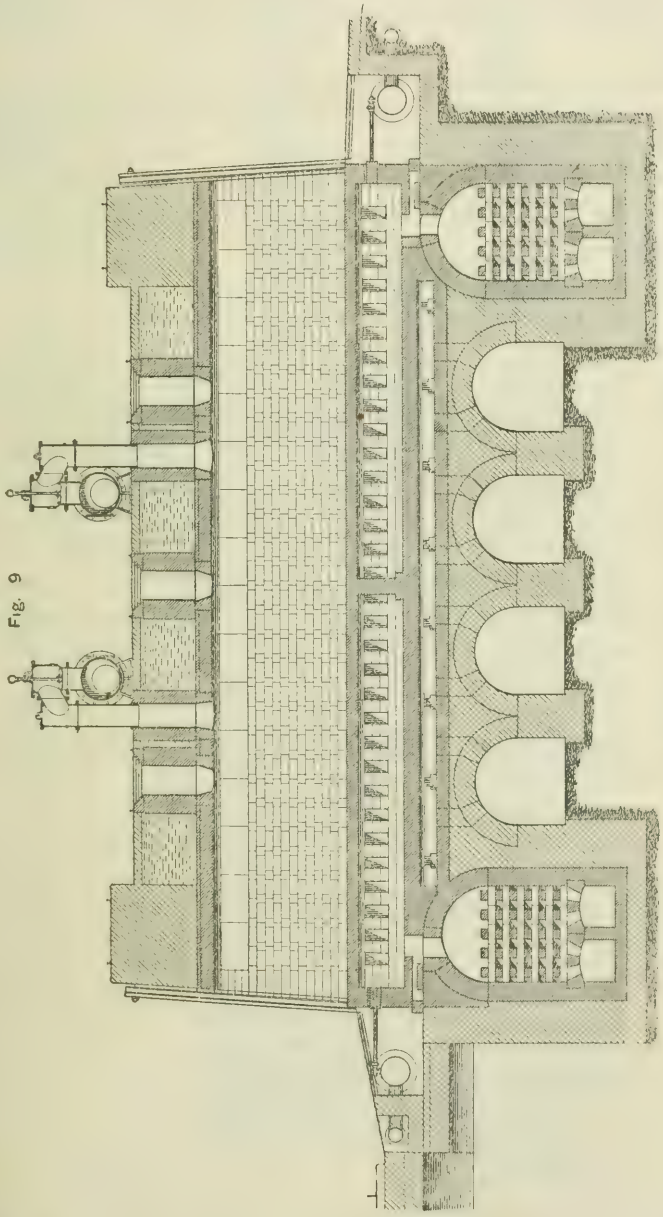


Fig. 9

Longitudinal Section of Regenerative Coke-Oven for Saving By-Products.

not, indeed, satisfy the theoretical principle that the gases should

be collected either only above or only below the stock-level; but it has proved itself the best *charging*-apparatus, probably because, by means of the suspended pipe, the center of the stock-column is most effectively kept loose. At these furnaces the bell is operated by steam.

Lürmann was the first to emphasize the desirability of making the different parts of the blast-furnace more independent of one another than formerly, and of relieving the lower parts from the weight of the upper. Fig. 25 shows such a construction. But some have gone still further and separated the parts of the furnace-walls above the bosh, connecting each by means of a ring and brackets with the supports of the top-platform.

The cooling of the hearth is generally very complete, from the bosh to the bottom. Yet it seems to have been rendered in part or wholly unnecessary by an arrangement which has been devised by Director Burgers at Gelsenkirchen, namely, the employment of carbon bricks made of retort-graphite or of coke-dust (containing 1 to 2 per cent. of ash), moulded with tar and glow-heated (*geglüht*). These are substituted for the ordinary fire-bricks of the hearth, and seem to be well-adapted even for the bosh- and belly-walls. They are likely to find increasing use wherever the blast-furnace manager is not troubled with lead and zinc in the ores. The tuyeres are always cooled; but the form first introduced by Hilgenstock at Hörde (see Fig. 26) is frequently used instead of the ordinary closed form.

Brick hot-blast stoves are so nearly universal that iron stoves can be found at few works. The former have proved economical wherever the blast is to have at least 600° C. of temperature. With well-built iron-pipe stoves, 500° to 550° C. can be maintained without difficulty, as is proved at Gleiwitz. The opposition to brick stoves lasted longest in Upper Silesia, where the immense quantity of zinc-dust (oxide) carried by the tunnel-head gas rendered the economy, and even the operation, of such stoves problematical. When they were finally adopted, their employment required a careful pre-cleansing of the gas, which was performed first in the dry way, afterwards by means of water-spray. And it is still necessary, in cases (frequent in Upper Silesia) where the ores carry zinc, to blow out the stoves by letting the blast pass through them for a brief period (often not more than a second) before admitting it to the blast-furnace.

Iron stoves have been, and will be with good reason, retained where a low-silicon gray or mottled iron is made, especially for chill-castings, to which end, it is true, a *cold* blast is not, according to the

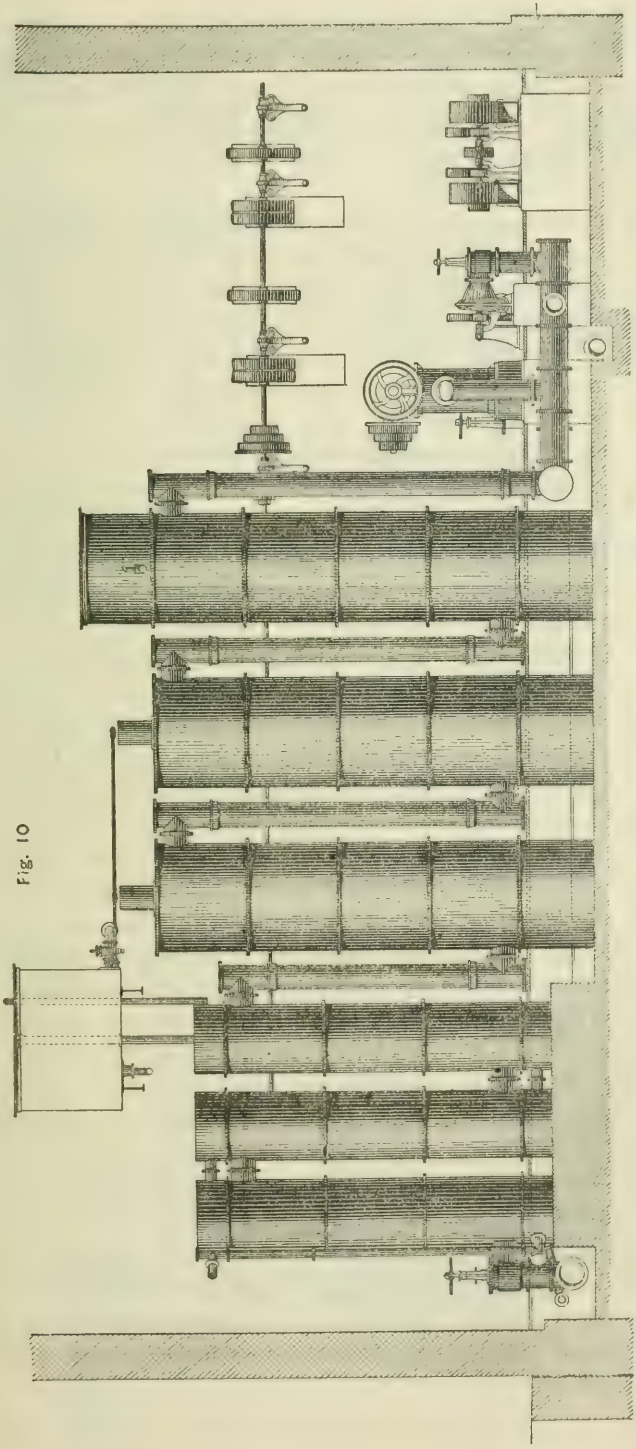


Fig. 10

Elevation of Condensing and Purifying Apparatus for By-Products from Coke-Ovens.

ancient prejudice, necessary, but the blast should not be hotter than 300° C. It is certainly an error to employ brick stoves for low temperatures. They give their best results at a blast-temperature of 900° C. In form, these stoves have followed the well-known line of development, from the original checker-work to vertical cylindrical or prismatic flues.

For a long time, difficulty was encountered in so conducting the burned gases as to heat uniformly the entire cross-area of the apparatus; and Engineer Böcker, at Friedenshütte, near Morgenroth, in Upper Silesia, effected an important improvement when, instead of giving a uniform cross-area to his gas-flues, he made those in the center, where the gases traverse the shortest path, smaller than those at the sides, through which the path is longer. Fig. 27 shows his construction. The same end has been sought of late in Luxemburg by means of slide-valves, apparently a less simple and effective device.

To prevent the gas from striking back into the blast-pipe, the simple Steffen "spectacles" (*Brillen*), shown in Figs. 28–31, are largely employed.

The blast-furnace gases are, of course, primarily used for heating the blast, but secondarily, also, in raising steam, particularly for the blowing-engines. A disaster at Friedenshütte in Upper Silesia, which spread consternation through the whole district, namely, the simultaneous explosion of a set of 22 boilers, heated with blast-furnace gas, gave rise to a thorough re-examination of the subject of firing under boilers with gas. While it was proved that even a serious gas-explosion under such circumstances could not possibly produce of itself such frightful effects as this instance presented, still it was not quite disproved that the impulse to the beginning of a steam-explosion might possibly be occasioned by the accumulation and subsequent explosion of unburned gas in the flues beneath and between the boilers. All the witnesses of the Friedenshütte catastrophe became its victims, and a complete explanation of it can never be given. But in rebuilding the boiler-department of the works, great care was exercised to secure, by means of regenerators in front of the gas-jets, the spontaneous and certain ignition of the gas-current whenever, after any brief intermission, it may be again directed under the boilers.

Fig. 32 shows this arrangement at Friedenshütte. The following statement shows the economic gain from a similar arrangement at Hayingen, in Lorraine: Before the use of regenerator-firing, in fifteen boilers, of 100 square meters heating-surface, there was pro-

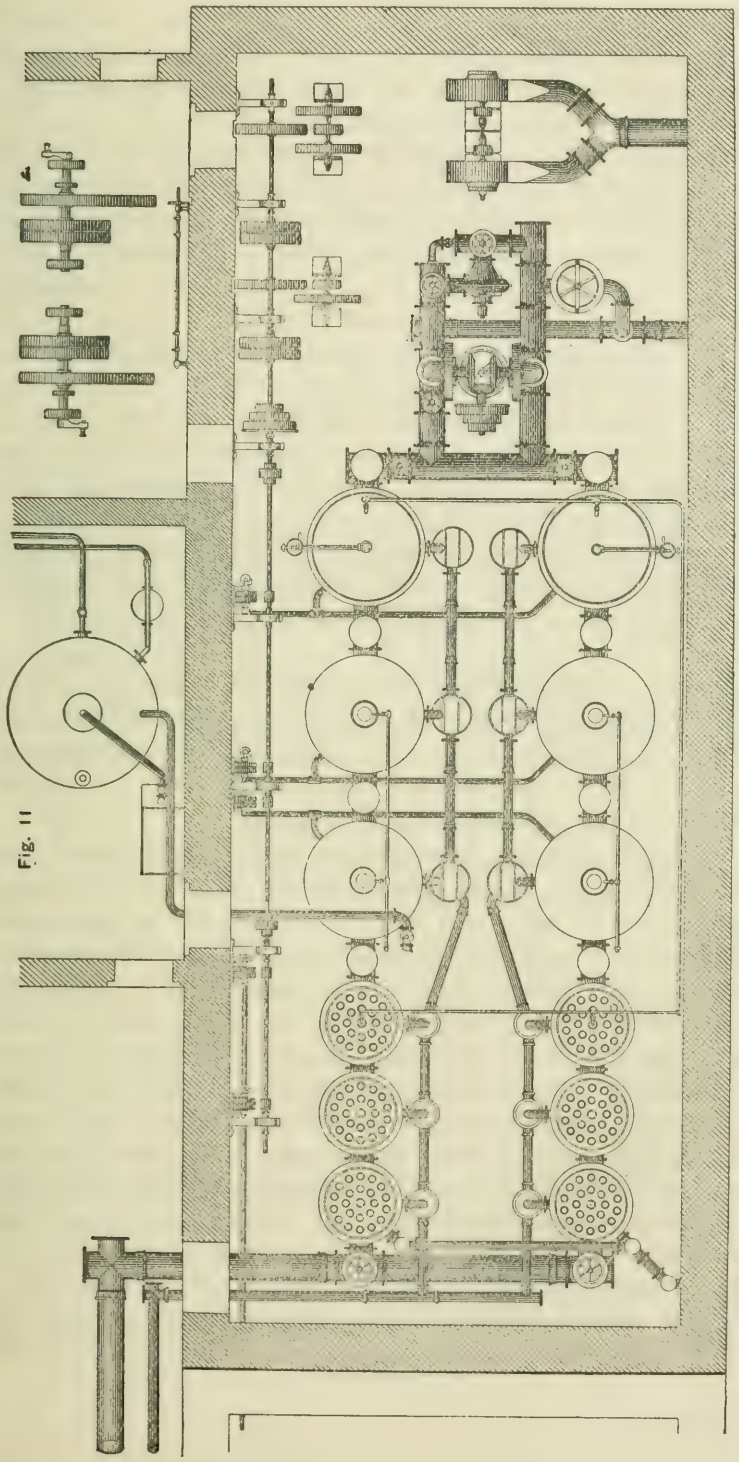


Fig. 11

Plan of Condensing and Purifying Apparatus for By-Products from Coke-Ovens.

duced, per square meter, per hour, 16 kilos of steam; afterwards, 20 kilos. At Friedenshütte the gain is still greater, 14 kilos of water, per square meter, per hour, being evaporated, as against 8 to 8.5 kilos before the regenerators were used.

Pig-iron is made, according to the market demand for its further use in the cupola, puddling-furnace, Bessemer or Thomas converter, or open-hearth furnace, or in casting direct, without remelting. The distribution of the total product among the several classes is given on a succeeding page. There is a remainder of *Brucheisen* and *Wascheisen* (furnace-scrap sorted or washed from sand, cinder, etc.). That we are, as a rule, far behind American practice in the amount of daily product, is easily explained by the leanness of our ores. If, upon such lean ores, we should blow faster, the result would be an imperfect reduction and a scouring cinder. Nevertheless, our performance is not contemptible, as the following examples may show:

At Gleiwitz, a furnace of 255 cubic meters capacity, with blast heated to only 500° C., in iron stoves, produces daily, and with regularity, from a charge carrying 37 per cent. of iron, 61 metric tons of pig, upon a fuel consumption of 95 kilos of coke to 100 of pig.

At other works in Upper Silesia, where the blast is heated in brick stoves to 700° or 800° C., 72 tons, per twenty-four hours, are produced in furnaces, of 300 cubic meters capacity, from a charge containing 30 per cent. of iron; but the consumption of coke per 100 kilos of pig, amounts to 120 to 130 kilos.

At a few works, the conditions are much more favorable. At Friedenshütte, the newest furnace, of 278 cubic meters capacity, charged with 70 per cent. Upper Silesian ore, 25 per cent. cinder and 5 per cent. Hungarian spathic ore, produces 80 to 90, sometimes 100, tons of Thomas pig per 24 hours, consuming 120 to 130 kilos of coke (containing 12 per cent. of ash) per 100 kilos of pig. The ore-mixture contains 33 per cent. of iron, 0.4 of lead and 0.4 of zinc. The dolomitic limestone flux contains from 1 to 2 per cent. of zinc.

At Ilsede, where a very uniform mixture of phosphoric brown-hematite kidney-ore (*Bohnerze*) is used without fluxes, and yields 35.7 per cent. of iron, the coke-consumption per 100 kilos of pig is 89.7 kilos (sometimes running down to 79.1 kilos), and the average product is 181.4 metric tons in twenty-four hours, while occasionally 190, and even 200, tons have been reached.

At the Rhenish steel-works, where the blast-furnace charge consists of one-third *minette* (oölitic brown hematite) and two-thirds

domestic and foreign ores, the average daily product from two furnaces is 320 metric tons.

In contrast with the relatively simple nature of the charge in the works above-named, the conditions as to ore-mixture are, in Westphalia, highly complicated. At Hörde, it is necessary to mix from twelve to fifteen different kinds of ore; and at Oberhausen, the largest German blast-furnace establishment, up to thirty kinds are mixed for the charge. The following table illustrates the practice at Hörde:

Composition of a Bed prepared for Charging, at Hörde.

Parts.	Ores, etc.	Fe. P'r ct.	Mn. P'r ct.	Al_2O_3 P'r ct.	CaO. P'r ct.	MgO. P'r ct.	SiO_2. P'r ct.	S. P'r ct.	P. P'r ct.	Lossby Heating. P'r ct.
2,500	Roasted black-band, mined by the Company	45.62	1.78	10.00	5.58	2.52	12.54	1.17	0.70	.....
1,000	Crude black-band, mined by the Company	29.12	1.34	4.33	2.74	1.56	8.97	0.23	0.40	40.25
2,000	Westphalian puddle-cinder	53.87	5.00	.....	.....	.....	10.50	.....	3.10	.....
500	" hammer-cinder	60.01	.....	.....	.....	.....	12.37	.....	3.04	.....
500	Siegerland manganese-slag	45.77	13.62	.....	.....	.....	16.74	.....	0.99	.....
500	Puddle-cinder from the Saar	59.56	1.07	.....	.....	.....	8.44	.....	4.89	.....
500	Forge-cinder, produced by the Company	50.23	.....	.....	.....	.....	32.61	.....	.....	.....
750	Brown hematite from the Lahn	42.85	1.22	4.54	1.74	1.00	24.20	.....	0.53	9.13
500	Red hematite from Sauerland	24.15	0.24	1.23	31.65	1.11	5.47	0.02	0.15	25.05
1,250	Manganiferous iron-ore from Greece	32.65	16.89	2.29	6.27	3.04	5.79	0.19	0.35	14.18
*2,600	Limestone from the Lenne	0.20	.....	.....	51.14	2.18	0.73	.....	.....	43.54
12,600										

These conditions and, consequently, the proportions of the mixture are constantly changing, so that analyses of the several ores must be continually made.

The ores of Lorraine and Luxemburg carry a proportion of phosphorus which fits them for the manufacture of Thomas pig, containing about 2½ per cent. of phosphorus; but they carry also an amount of sulphur which requires, as I have already intimated, the addition of manganiferous ores. In the northern part of Lorraine, it is true, a self-fluxing ore-mixture can be made out of suitable proportions of the different varieties; in the southern part, the addition of limestone is necessary. It is the general rule in Germany, that limestone is required; the only exception, outside of Luxemburg and part of Lorraine, being the works at Ilsede.

The relative proportion, in the pig-iron product, of the different

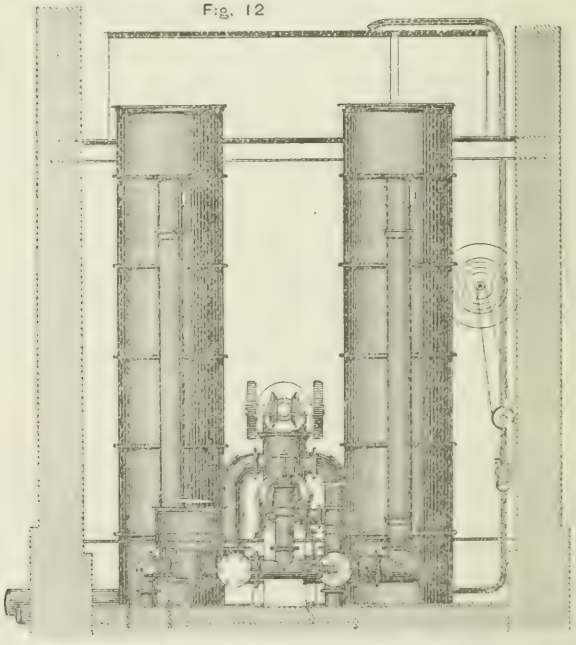
* The proportion of limestone is 26 per cent. of the total ore.

classes of pig, has greatly changed since 1876, as the following statement will show:

Classes of Pig.	Production in Kilotons. 1876.	1889.
Foundry,	117	610
Forge (for puddling),	1343	1965
For ingot-iron (including cast-steel),	341	1907
For direct casting (including *Brucheisen* and *Wascheisen*),	45	43
Total, pig-iron,	1846	4525*

That is to say, the production of pig for ingot-iron, which was, in

Fig. 12



End-View of Gas-Washers in Condensing-Plant (Figs. 10, 11).

1876, only one-fourth as great as that of pig for puddling, now equals the latter; and the product of direct castings from the blast-furnace without smelting has decreased not only relatively but absolutely.

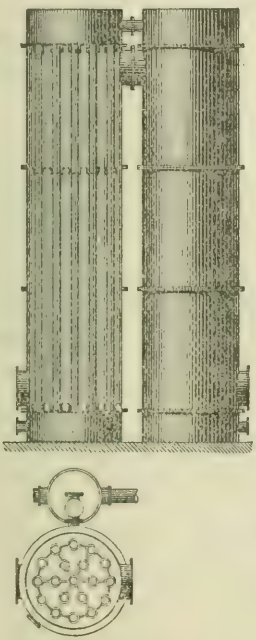
Ferromanganese is regularly made in Germany in the blast-furnace, but exclusively at the Oberhausen works (the most extensive

* Of which only 25 kilotons were made with charcoal as fuel.

of the kind in the country), where all other kinds of pig-iron are also produced.

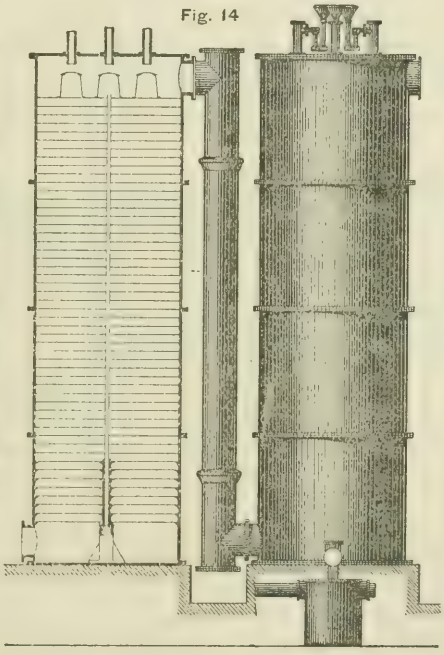
The practice of subjecting the liquid iron direct from the blast-furnace to the fining (oxidizing) process which it is to undergo, is continually extending. The liquid pig is sometimes conveyed a long distance. At Hörde, for instance, it is carried by railway 2 kilometres (say $1\frac{1}{4}$ English miles) from the blast-furnace to the Thomas apparatus. The tapping of small casts at regular periods makes no trouble. The workmen quickly become accustomed to

Fig. 13



Section and Elevation of Gas-Cooler.

Fig. 14



Section and Elevation of Gas-Washer.

tap from the furnace into the ladle (which usually stands on a scale) the exact quantity required. Figs. 33–35 (Plate II.) show the form of ladle-car at Hörde.

The iron-notch or tapping-hole is kept during the cast as small as practicable, and when sufficient iron has run out it is stopped with a chamotte or clay plug on the end of a long rod. An adequate supply of such plugs, previously prepared by the keeper, is always on hand. When the tapping-hole has become gradually too short, it is opened

and repaired; the space made for packing in the new material being partly filled with loam, and a conical wooden block being laid in it and tamped with a mixture of coke-dust and tar, over (in front of) which loam is pounded in. The tar-coke mass stands excellently, like the bricks of similar composition already described.

The ladle for the molten iron is run in upon a depressed track between two furnaces.

Such use of the liquid pig presupposes, it is true, a definite uniformity of operation in the blast-furnace. Where this cannot be maintained, the only resort (provided several blast-furnaces are running at once on a given class of pig) is to collect the liquid iron in a large reservoir or tank, in which the different grades can be mixed. These tanks are built solid—not tilting (as contemplated in Jones's German patent, No. 50,250)—and usually hold at least 80 metric tons of pig-iron.

The utilization of blast-furnace slags has long been a problem of great practical difficulty, but the increased production of slags rich in lime has led at last to a commercial application far more extensive than all others, namely, the manufacture of cement—the so-called *Schlackenpuzzolanicement*. As a general rule, the slags designed for this use must be cooled by running them into water. The slag thus granulated can be easily ground with slacked lime, and yields a cement of strength sufficient for most purposes and generally possessed also of hydraulic properties.

For some time it was attempted (and with reported success) to add to Portland cement, without injuring its quality, a considerable proportion of ground blast-furnace cinder. To this the manufacturers of Portland cement very properly objected. Now the cement made from slag is frankly called by its own name, and is not less esteemed in the building trade on that account.

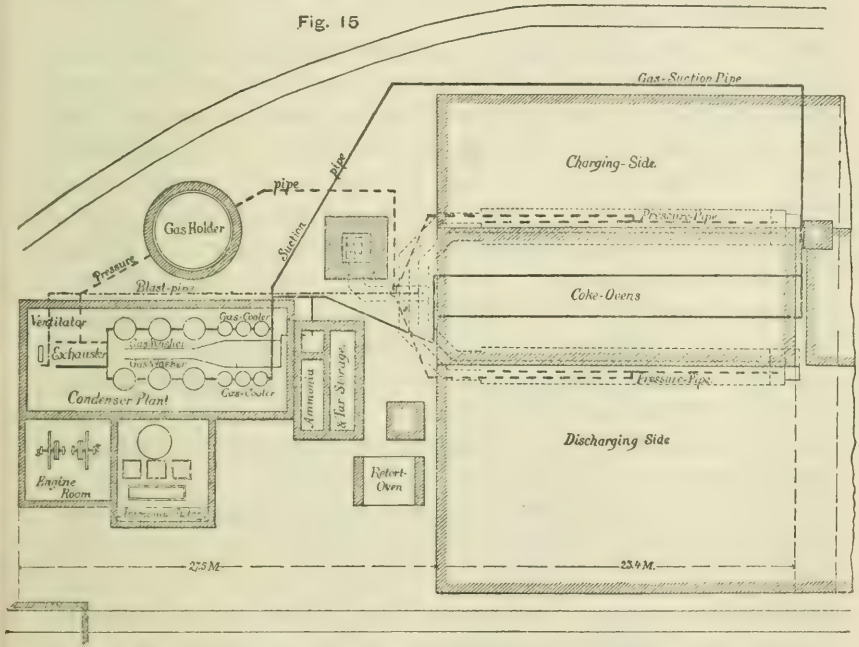
Nothing new has been added during the period under consideration to the German practice of other methods of utilizing blast-furnace cinder, such as the ballasting of railways, road-making, paving, the manufacture of mineral wool, etc.

I give below in detail an example of practice in the manufacture of Thomas pig and one in the manufacture of ferromanganese.

Manufacture of Basic Pig at Hörde.—As a type of the manufacture of pig-iron for the basic Bessemer process, we may take the works at Hörde, where, as already observed, the liquid pig is directly converted into ingot-iron. The Hörde furnaces (shown in Figs. 21 and 22) are especially arranged for the production of

Thomas pig; and the furnace-lines and other features of the four stacks devoted to this manufacture have been determined with reference to it. Each furnace produces in 24 hours from 100 to 130 metric tons of pig-iron, using in coke from 85 to 90 per cent. of its weight of product. A fundamental requirement of the first importance is uniformity of quality in the pig. The object is to obtain a pig-iron containing up to 3 per cent. of phosphorus, 2 of manganese, not over 0.15 of sulphur and not over 0.5 of silicon. The

Fig. 15



General Plan of Coking- and Condensing-Works at the Pluto Mine, Westphalia. (Details in Figs. 10 to 14.)

blast-furnace practice completely fulfils this requirement, although, as already observed, it has to contend with great obstacles in the number and variety of the ores used, necessitating very difficult mixing. Derangements of the furnace-working by the leakage of water into the hearth have not occurred for years, that is to say, not since the open tuyeres were adopted. Highly important, moreover, is the avoidance of an evil often very serious in the manufacture of Thomas pig, namely, the so-called lime-trouble or excessive basicity of the cinder. As soon as the influence of an excess of lime begins to appear in the slightest degree, it is counteracted by the timely

addition of highly fusible silicates. The presence of manganese is notoriously advantageous to the running of the furnace, but superfluous during the heat in the converter. The high proportion of manganese in the furnace-charge is carried, therefore, partly to facilitate the furnace-work, but chiefly to counteract the sulphur. Experience has shown that 2 per cent. of manganese is generally an adequate protection against an excess of sulphur in the pig. In using the liquid pig direct from the converter, the presence of superfluous manganese is specially felt, because in that case (unlike the case of pre-melting in the cupola) all the manganese has to be oxidized in the blow. For this reason the practice of treating high-sulphur Thomas pig by mixing with high-manganese pig has been introduced, in order to remove sulphur from the one and manganese from the other. Manganese, as is well-known, acts upon the sulphide of iron, liberating metallic iron and forming sulphide of manganese, which separates as a slag. Thus the excess of manganese is removed by means of the sulphur which would otherwise render the pig unsuitable for the converter. In order to facilitate the reaction of manganese and sulphur, an arrangement is employed permitting considerable quantities of the two classes of pig to be treated together for a sufficient period. It is found that two parts of pig containing 0.5 per cent. of sulphur and one part of manganiferous pig containing only 0.02 per cent. of sulphur, produce a pig with only 0.1 per cent. of sulphur, and still containing 1.5 per cent. of manganese. The separated manganese sulphides, which carry also silicates from the oxidization of silicon in the bath and from the lining of the vessel, are utilized as manganiferous additions to the blast-furnace charge. They contain up to 40 per cent. of manganese and 10 to 15 per cent. of sulphur; but the sulphur which enters the blast-furnace as manganese sulphide has proved to be entirely harmless.

The flue-dust of the blast-furnace gas-firing at Hörde contains a surprising amount of sulphate of potash, which is saved by leaching and evaporation, yielding a sulphate containing about 40 per cent. of potassa.

Manufacture of Ferromanganese at Oberhausen.—As already observed, ferromanganese is made in the blast-furnace at Oberhausen. The ores come chiefly from the Caucasus, Greece and Chili. They are pyrolusites, carrying 50 to 55 per cent. of manganese. The variable ores of Weilburg, on the Lahn, are used also to some extent. As admixture, brown hematites with 20 to 30 per cent. of

iron and 14 to 21 per cent. of manganese are utilized; they come from the Lahn and Cartagena.

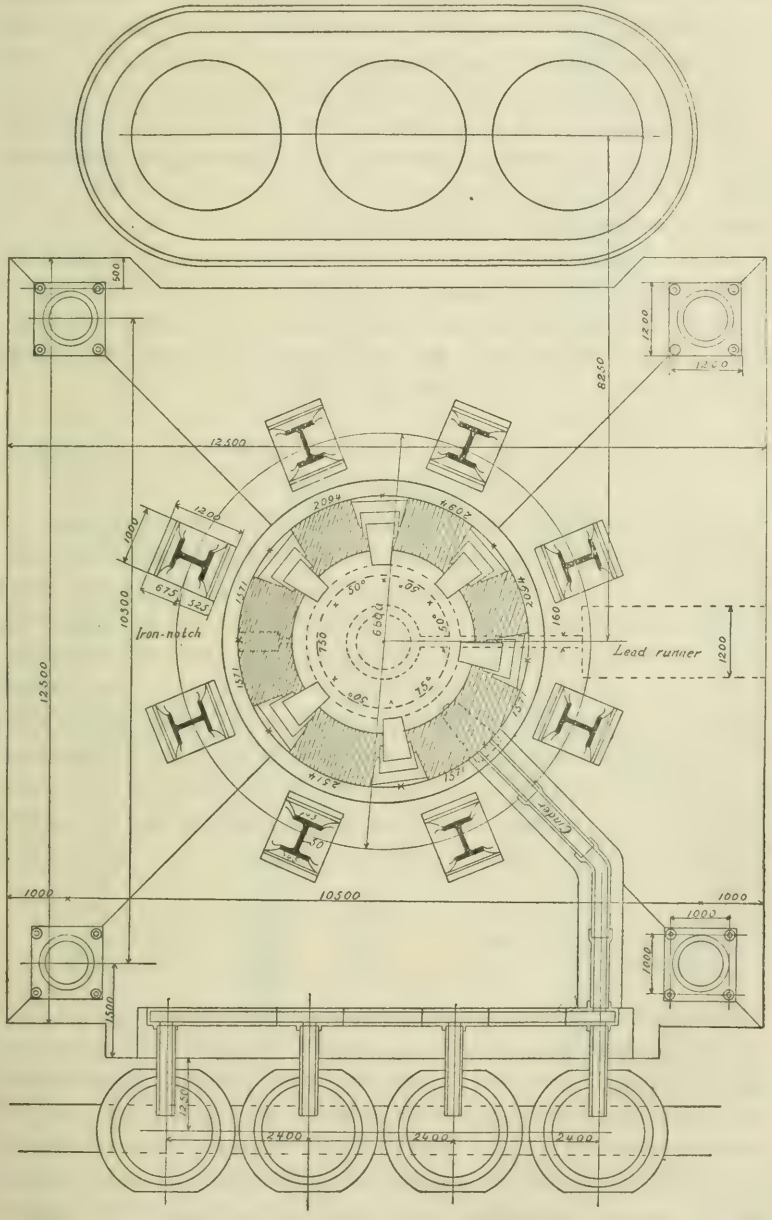


Fig. 17

Plan of No. 1 Blast-Furnace at Friedenshütte, Upper Silesia.

The ferromanganese produced contains from 30 to 87 per cent.,

averaging 66 per cent. of manganese. When it carries over 82 per cent. it disintegrates very soon into small pieces and powder, and can, therefore, be shipped in tight boxes only. Of the manganese in the charge, 60 to 70 per cent. goes into the iron. The slag contains, in percentages: silica, 25; alumina, 15; lime (with calcium and barium sulphides), 45; and protoxide of manganese, 15.

The consumption of coke ranges from 190 to 210 kilos. per 100 kilos. of ferromanganese—according to the percentage of manganese in the latter—the high-manganese ferro requiring most fuel. The average product is 40 metric tons per 24 hours.

A very interesting feature is the immense quantity of graphite which, during the cast, separates from the metal and fills the air—evidently because the molten metal possesses at the high temperature of the furnace-hearth, the capacity to alloy itself with more carbon than it can retain as the heat is lost.

III.—Malleable Iron.

A. *Wrought-Iron.*

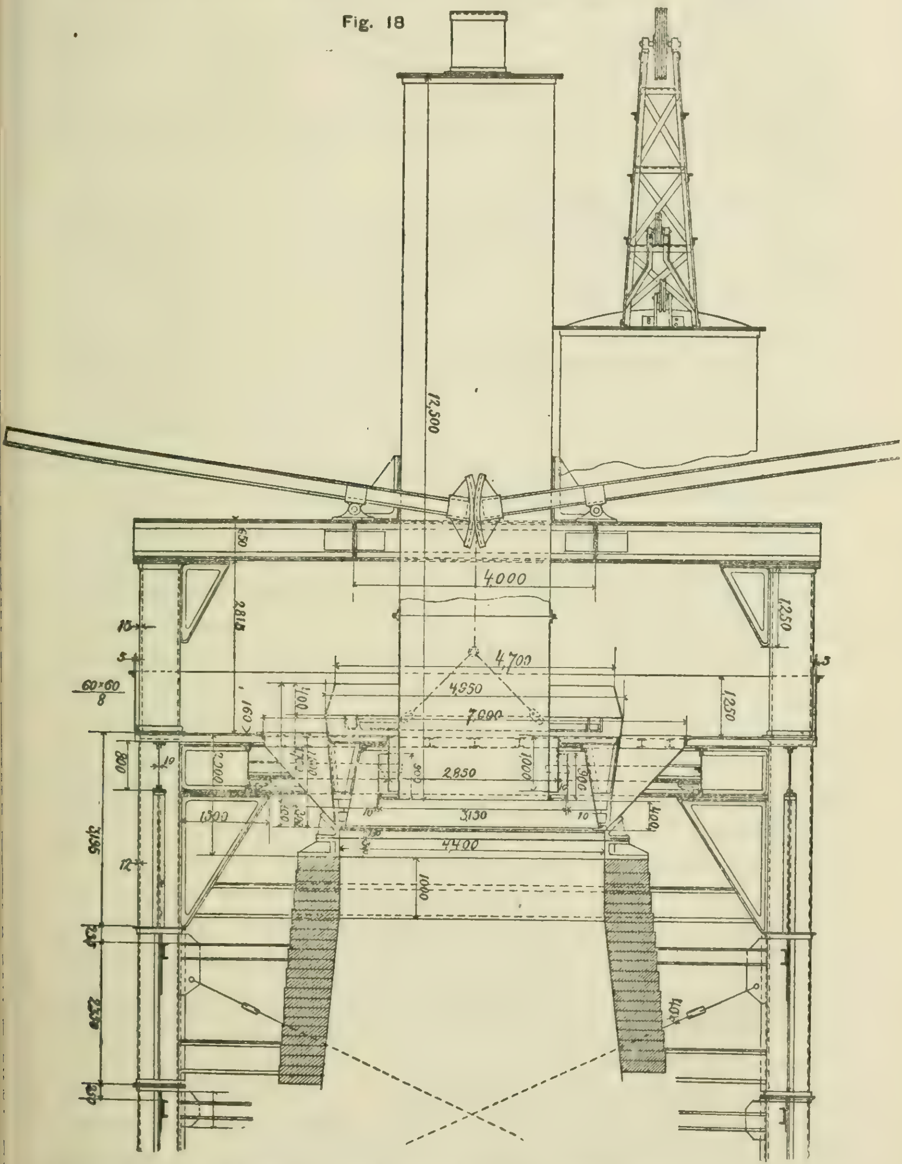
Puddling, which, after the invention of the Bessemer process, and still more after the introduction of the Thomas process, was supposed to be rapidly approaching its extinction, has been in Germany not only maintained, but, in some respects, extended and improved. True, puddled-iron relatively retreats, year by year, before the advance of ingot-iron, but the change is a slow one. The annual product of wrought-iron has increased since 1876 from 948 to 1655 kilotons, and the increase has continued in the last year, as the statement given below will show; while the product of ingot-iron, which has, indeed, had a more rapid development, has advanced from 249 kilotons in 1876 to 2046 in 1889, having passed its elder rival in 1887:

	1886.	1887.	1888.	1889.
Wrought-iron (kilotons),	1416	1625	1547	1655
Ingot-iron, "	1376	1738	1785	2046

The reasons for this striking persistency in puddling lie partly in the obstinate prejudices of consumers, but partly also in certain real advantages possessed by wrought-iron—namely, its relatively great ductility, the facility with which it may be welded, and the circumstance that, under excessive strain, it breaks gradually rather than suddenly, while its strength, though inferior to that of ingot-iron, is generally uniform throughout to a trustworthy degree. While the use of ingot-iron for large constructions, such as railways, buildings, boilers and bridges, is constantly increasing, the

small operator—blacksmith or machinist—does not seem to be able, as yet, to get along without his wrought-iron.

Fig. 18



Vertical Section of Closed Top and Charger of No. 1 Blast-Furnace at Friedenshütte.

Ingot-iron, however, is already used, not only for all rolled shapes, such as angles, tees and beams, but for all varieties of mer-

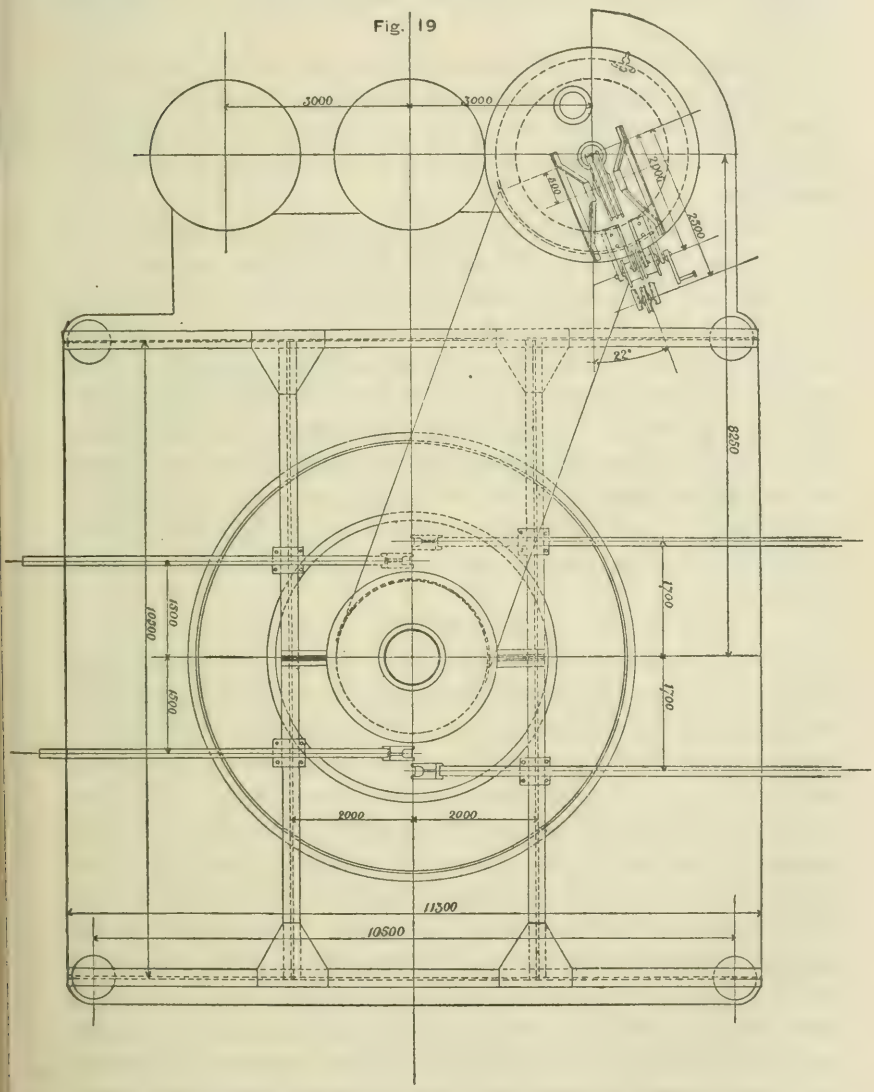
chant-bar, which is furnished, when desired, in quality suitable for welding. The ingot-iron of Peine is so pure that it can be used instead of Swedish iron in making horse-shoe nails.

On the other hand, less valuable ingot-iron is rolled into cheap beams, which still are stronger than those of puddled-iron, and have completely beaten the latter in the market. Such beams are rolled at Oberhausen and Neunkirchen in sizes up to 400 mm. ($15\frac{3}{4}$ in.) in height.

Weld-iron (wrought-iron) is produced in Germany by puddling exclusively, and almost invariably by that variety of the puddling process (*Luftfrischen*) which depends upon the air for oxidation. The other variety (*Erzfrischen*), in which oxidation is effected by means of a fettling of ore, and which is so common in America, finds among us a very subordinate application only.

The essential improvements in puddling have not been in mechanical rabbling (though this has been done in the district of Saarbrücken and in Lorraine, without, however, changing the old construction), nor in the use of the rotary mechanical puddler (which, apart from a few small experiments, has never obtained a footing in Germany), but rather in the more complete utilization of fuel, by means of gas-firing and double furnaces. Two furnace-designs deserve special mention in this connection: that of Springer, with fixed hearth and reversible flame; and that of Pietzka, with constant direction of flame and reversible hearth. Springer's furnace has two hearths, placed end to end, one of which is charged with pig, as the iron begins to come to nature in the other. The flame, brought to a very high temperature in the regenerators, passes from the puddling-hearth to the melting-hearth. The regenerators are in front of or under the furnace, as usual, and vertical or horizontal. This furnace may be used with direct firing or with gas-firing, and is shown in Figs. 36–40 (Plate III.), as constructed for gas-firing. It has done excellent work, even with lignite as a fuel. It has but a single disadvantage, namely, the dependence of the two hearths on one another; and this led to the invention of the Pietzka furnace. While in operating the Springer furnace the flame is reversed only when the puddling in one hearth and the melting in the other are finished (or in other words, must go in one direction as long as the puddling process lasts, even though, by reason of any delay or prolongation of the process, the regenerators may be more or less cooled), this objection is obviated in the Pietzka furnace, the latest form of which, as seen at Zawadsky, in Upper Silesia, is shown in Figs. 41–44. Here the flame maintains

one direction, yet the hottest flame always strikes the hearth in which puddling is going on. The peculiarity of the construction is that both hearths are carried on a hydraulic piston which stands under-



Plan of Closed Top and Charger of No. 1 Blast-Furnace at Friedenshütte.

neath, between them. The connections between the side-walls and the fixed fire-chamber or flue are made with inclined conical surfaces. The hydraulic piston lifts the hearths a little before turning them;

and hence they turn freely and without friction, until the reversed position has been reached, when they are lowered into place, and the connection is again complete. The results of practice with the Springer and Pietzka furnaces may be judged from the following examples:

A Springer furnace at Königin-Marienhütte, Saxony, turned out in 1889 an average product of 8,815 kilos per 12-hour shift, and in some weeks 10,000 kilos, with a coal-consumption of 40 to 50 kilos. In Styria the consumption of lignite is 60 to 65 kilos (per 100 kilos muck-bar).

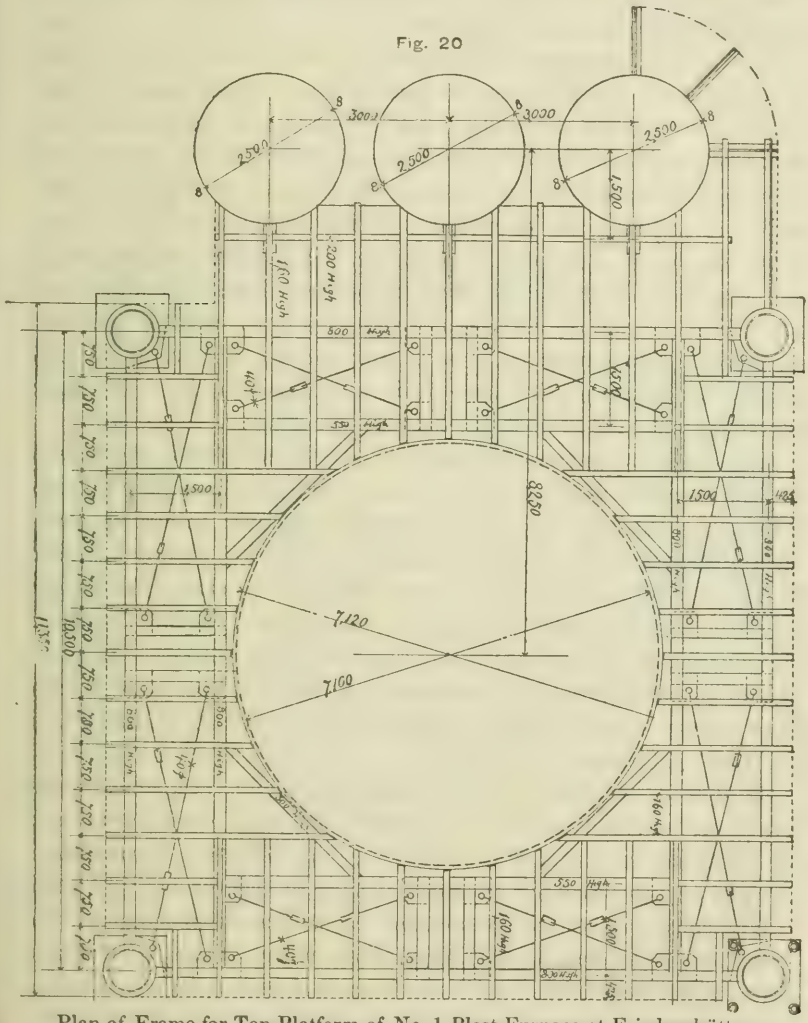
A Pietzka furnace at Zawadsky, Upper Silesia, holding a charge of 500 kilos, produced, by direct firing, 6280 kilos; by gas-firing, 6674 kilos of muck-bar, as the average per shift—in single weeks, however, an average of 10,000 kilos of muck-bar, with a coal-consumption per 100 of 66 kilos, with direct firing, and 42 kilos with gas-firing. The escaping gases were here utilized further in the production of steam. The latter record is especially favorable, since, deducting the amount of coal practically burned for the steam-power required in the operation of the furnaces, it shows only 28 to 29 kilos of coal used in puddling proper, while if all the gas were used to heat the regenerators, this figure would be further reduced to 20 to 25 kilos.

B. *Ingot-Iron.*

1. *Converter-Metal.*—The production of ingot-iron in the converter has experienced in Germany twice since 1876 a revolutionary change. In that year it was the admirable example of American works, far surpassing ours in their results, which led to a transformation of German establishments. It would be superfluous here to enumerate all the arrangements which we adopted from the United States: the continuous pre-melting in the cupola, the continuous operation of the converter, and, above all, the Holley movable converter-bottom, may be mentioned as the three most important. With regard to the movable bottom, Holley's original construction was abandoned, little by little, in favor of the draw-bottom, of which Fig. 45 shows the usual form, and which has the great advantage, that without breaking the blast-connections, the wind-box only being removed, the bottom can be drawn out and replaced with a fresh one. With regard to the air-passages, some have used bottoms with grouped tuyere holes, and others the tuyere-bottom (*Düsenboden*) of Lilienstern, but neither has shown, in the acid Bessemer process, any marked superiority to the other.

Much more extensive was the second transformation of practice, after the discovery by Thomas of the dephosphorizing process. The most important discovery of this eminent man (whose too early departure the German technical world mourned with his fellow-mem-

Fig. 20



Plan of Frame for Top-Platform of No. 1 Blast-Furnace at Friedenshütte.

bers in this Institute) owes its practical development to the perseverance of the two German licensees under the patent, namely, Hörde and the Rhenish Steel-works.

The great importance to Germany of the basic process is indicated

by the product, which between 1883, the first year of its complete success, and 1889, has increased more than fourfold, and now constitutes more than half the world's aggregate product of basic metal. The chief reason is that most of the German ores, as has been observed already, are high in phosphorus, and hence entirely unsuited for the acid Bessemer process.

The original attempts of Thomas to make a basic lining from dolomite, with the addition of water-glass, small amounts of clay and other siliceous substances, speedily came to an end, as did the numerous experiments of others who stood upon his shoulders, and who tried to substitute various materials, such as bauxite, for dolomite or magnesite, or to replace the binding-material with something else, such as gypsum, starch, etc. All came back at last to the simple basic lining of burnt dolomite, mixed with anhydrous tar. The German converters, as a rule, are lined with this mixture by stamping, except at the throat, where bricks of it are used. All repairs of lining are likewise made with tar-dolomite. In single instances, machine-pressed bricks have stood well.

The bottom of the basic converter is almost universally the tuyere-bottom, stamped of tar and dolomite and arranged as a draw-bottom. The use of acid tuyeres (tuyere-groups) in a basic lining has practically disappeared. Nor is the preparation of these bottoms difficult. It requires only special care, upon which, rather than upon small variations in composition, the durability depends. Suffice it to say that at present a bottom generally lasts through 25 heats, though instances are by no means rare in which 30 to 50 heats have been blown without removing the bottom. The lining itself lasts for 150 to 160 heats.

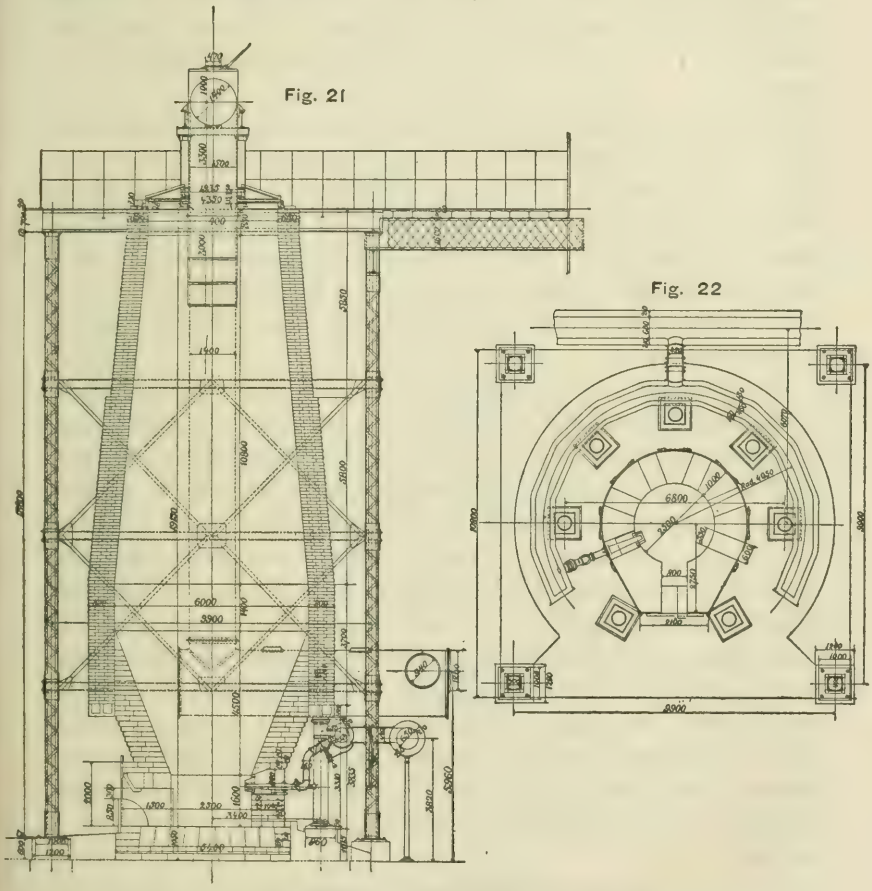
It should be added that the attempt to make by machinery the converter-bottoms, which had always been very carefully stamped by hand, has not met with approval. At least the arrangement devised for this purpose at Bochum has not been extensively adopted.

The converter is usually lined *in loco*, without displacing it. An exception is presented at Neunkirchen, where it has been found better, after loosening half of the journal-ring, to remove the converter by means of a car, and reline it in a separate building.

The liquid pig from the blast-furnace is charged into the converter wherever this course is at all practicable. Of course, not all localities will permit it. Sometimes, as, for instance, at Peine, special reasons (in that case the location of a trunk railway) seem to render it desirable to separate the converting-department from the

blast-furnace. On the other hand, it has been found (as already related of Hörde, where the blast-furnaces and the converters are 2 kilometers apart) that the transfer of the liquid pig over considerable distances exerts no unfavorable influence.

It has also been found that the most suitable pig is one which contains 2, 2.5 and up to 3 per cent. of phosphorus. This has



Blast-Furnace at Hörde, Westphalia.

necessitated in Upper Silesia, for instance, where the ores carry too little phosphorus, the introduction of phosphorus into the blast-furnace in the form of Thomas slag. In other places, as at Ilsede, the phosphorites formerly separated from the ores are added, to bring up the percentage of phosphorus.

The pre-melting of the pig, when it is not taken direct from the

blast-furnace, is never done in reverberatories, but always in cupolas. Even in these, however, a heavy loss of manganese is unavoidable.

The period of the blow for all kinds of pig used in the basic process is proportionally much shorter than in the acid process, and never exceeds fourteen or fifteen minutes. It is, therefore, easy to make forty-five or more blows in twelve hours. This is, however, not done, because there is no necessity for such fast running at any of the works, save perhaps for a short period, when it is desired to make good the loss of product through a previous interruption of work.

The quantity of blast used in the after-blow, *i.e.*, in dephosphorization, is universally regulated by engine-measurement, *i.e.*, by revolutions. It is found that in the basic process the blast-pressure which was usual in the acid process is too low, and that it is better at the outset to arrange the blowing-engines for a pressure rising, if necessary, to 40 or, still better, up to 45 pounds gauge-pressure.

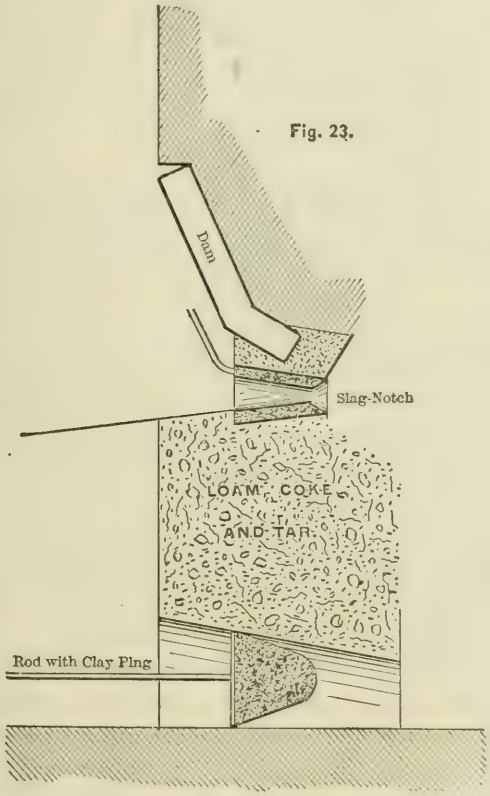
The belly of the basic converter holds generally from 7 to 10 metric tons of pig.

The basic slag (except where it is required, as already described, as a phosphoric addition to the blast-furnace charge) is sold for agricultural use. The farmers demand more of it than can be produced, since it has been found, if only ground to sufficient fineness, to yield without chemical treatment an excellent fertilizer for all kinds of soil, though best adapted to sandy, siliceous and peaty soils. In most of the works, the slag is simply poured off from the bath and collected before the addition of ferromanganese or spiegeleisen for deoxidation or recarbonization. In some works, on the other hand, as for instance at Hörde, the slag is divided according to Scheibler's method; the slag, rich in phosphorus and very poor in iron, being first decanted, and the blow being then resumed with the production of a slag poorer in phosphorus and richer in iron. The latter goes back to the blast-furnace, while only the former is sold as a fertilizer.

For a long time, many difficulties were encountered in the grinding of this slag to the extreme fineness required. The process created an enormous amount of dust; moreover, there was a constant segregation of the particles of iron mechanically contained in the slag. The slag-mills proved highly injurious to the health of the workmen, for whose protection against the dust very complicated arrangements consequently had to be introduced. Since it was proved, however, that properly constructed ball-pulverizers would remedy

completely the evil of the formation of dust, there has been no further difficulty from this source, and nothing now hinders the general use of this valuable product.

After deoxidation and recarburization in the converter, the iron is tapped into a ladle, which in basic works is attached, as a rule, to a crane carried upon a locomotive. The fluid metal is conveyed to



Fore-Hearth, Cinder- and Iron-Notches of a Blast-Furnace at Hörde.

another room, where it is poured into ingot-moulds. In some cases, these arrangements are impracticable, perhaps because old acid plants have been adapted for the basic process; perhaps for lack of the necessary space in the locality itself. The latter is the case with the handsome works at Phœnixhütte, near Ruhrort. Here it is necessary to pass the ladle from one crane to another, as is shown in Figs. 46 and 47 (Plates IV. and V.). The same crane here serves to bring the ladle

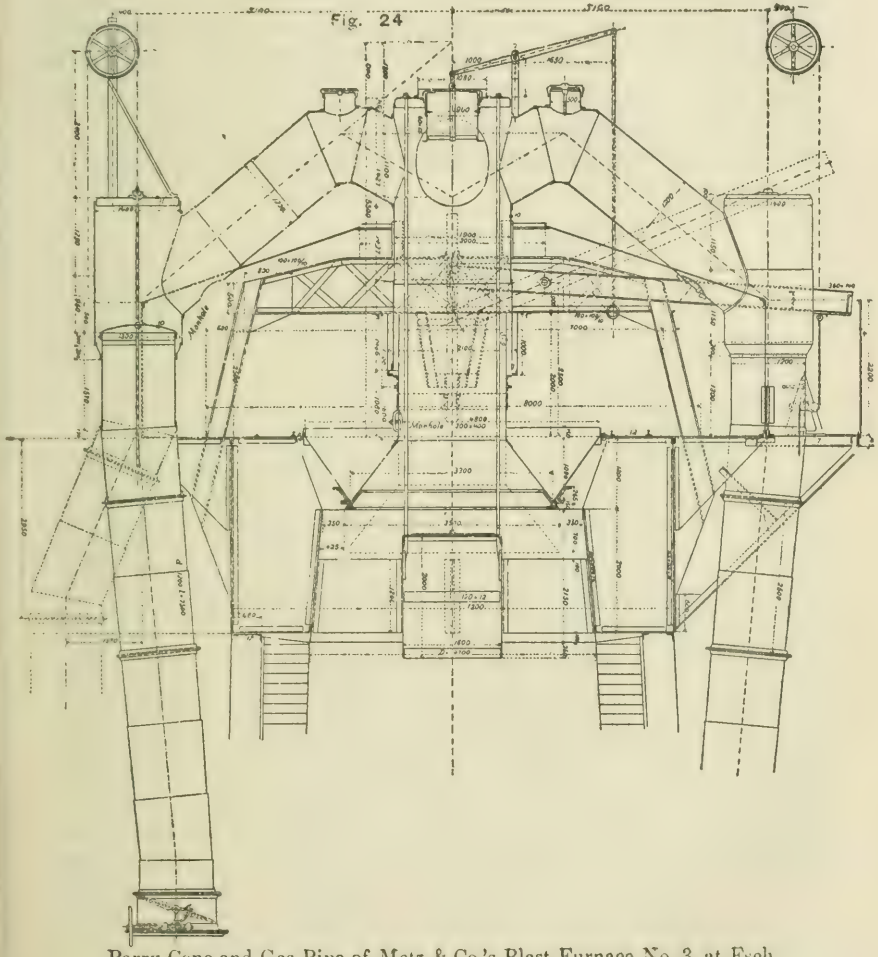
containing liquid pig from the blast-furnace and to receive the ladle containing ingot-iron from the converter. The latter ladle is delivered to a second crane, which stands in the center of the casting-pit. Fig. 48 shows the arrangement here employed to prevent disaster occurring through a sudden failure of hydraulic pressure in the cylinder. It will be seen, that if the crane, *a*, should suddenly sink, while in the position indicated by the dotted lines, the ladle could not strike the mould, *c*, because of the leg, *d*, attached to the lower side of the crane-arm, which would be stopped in its descent immediately by the masonry ring, *e*. To lower the converter further, the foot, *d*, must be brought round to the recess, *i*. This is the position shown by full lines in the figure.

It may be remarked here, that there are many other devices for the protection of the workmen in this department, particularly at Neunkirchen, where there is a gutter beneath the floor, which serves in case of a breakdown to direct the flow of molten metal to a pot; here, also, use is made of a blocking-tooth, which can be shoved into the rotating-gear on the axis of the converter so as to stop its turning, if, by any chance, the ram and rack of the hydraulic tilting device should fail to act. The vessel is generally so designed, however, that in every position its center of gravity lies below its axis of suspension; in case of mishap it will, therefore, naturally return to its upright position.

Fruitless efforts have been made to overcome the imperfect desulphurization in the basic, as contrasted with the acid converter. Rollet's proposals have been carefully tested without being found practicable—at least from an economic standpoint. Endeavors have, therefore, been directed to the elimination of the sulphur in the blast-furnace. The method which has been operated successfully at Hörde reduces the proportion of sulphur by utilizing the manganese, which eases the running of the blast-furnace while it is unnecessary for the basic process. This practice has already been described. In order to illustrate converter-practice under given special conditions, the following details of work are set forth.

Hörde.—Basic-converter practice at Hörde is carried on in the following manner: The pig-metal, weighed off in 10-ton charges, is brought to the converter in a tipping-ladle, of the form shown in Figs. 33 to 35 (Plate II.). Three converters are used in alternation. The lining consists of burnt dolomite brick for the throat, and rammed dolomite for the rest of the vessel. Formerly, the whole vessel was walled up with burnt tar-dolomite brick; but the ramming of the sides has

proved preferable, and better also than laying up with unburnt tar-dolomite brick. The bottoms have seventy-two cylindrical holes, 17 to 18 mm. in diameter. With a blast-pressure of 30 pounds, the duration of a blow is nine minutes before decarbonization, and five minutes after it. The phenomena, during the blow, do not differ



Parry Cone and Gas-Pipe of Metz & Co.'s Blast-Furnace No. 3, at Esch.

essentially from those exhibited in blowing cupola-metal; at most the flame may be slightly delayed. Generally speaking, direct-metal works hotter than cupola-metal.

The necessary lime-addition amounts to 14 per cent. of the pig-metal charged. At Hörde, two-thirds of it are introduced at the

start; three minutes after dephosphorization has set in, the metal-bath contains only 0.3 per cent. of phosphorus, while the slag, which is quite fluid, runs as high as 20 to 30 per cent. phosphorus and carries but little iron. This slag is separately poured off by Scheibler's method, and the balance of the lime is then charged. The iron that is unavoidably oxidized in the elimination of the residual phosphorus, is taken up by the new slag which, therefore, becomes excellent material for the blast-furnace, while the first slag is applied in agriculture. The operation of the Thomas process is shown in the accompanying tabulation of analytical results obtained from a series of gas-, metal- and slag-tests, which were taken at equal intervals throughout a complete blow.

Thomas Heat. No. 882.

	Si. Per cent.	P. Per cent.	Mn. Per cent.	C. Per cent.	S. Per cent.
Pig metal	0.58	2.75	1.57	3.60	0.079
Sample 1*	0.28	2.57	2.50	2.81	0.079
" 2	0.07	2.08	0.30	2.02	0.073
" 3	0.07	2.25	0.34	1.33	0.074
" 4	0.06	1.55	0.26	0.71	0.079
" 5	0.02	0.061	0.21	0.105	0.054
Steel	0.02	0.084	0.55	0.136	0.046

	SiO_2. Per cent.	FeO. Per cent.	MnO. Per cent.	CaO. Per cent.	MgO. Per cent.	P_2O_5. Per cent.	CaS. Per cent.
Slag sample 1*	9.20	0.55	8.87	76.10	4.86	2.94	0.63
" " 2	9.50	0.73	9.70	71.40	5.33	6.90	0.34
" " 3	9.30	2.30	8.42	66.17	6.47	7.32	0.31
" " 4	10.28	7.13	9.45	50.71	6.90	15.87	0.27
" " 5	6.99	11.98	5.40	46.84	4.09	24.73	0.52
" " 6	4.79	26.03	4.62	42.05	6.83	16.33	0.65

	CO_2. Per cent.	O. Per cent.	CO. Per cent.	CH_4. Per cent.	N. Per cent.
Gas sample 1*	3.5	8.1	2.0	0.9	85.0
" " 2	3.0	3.0	10.6	1.0	81.4
" " 3	3.0	0.3	28.3	1.6	66.6
" " 4	1.8	0.2	29.8	1.8	65.0
" " 5	1.2	0.8	1.6	0.9	95.6

Important reactions occurring between sample-periods (4) and (5) are not shown in this table; but many special investigations have proved that, during dephosphorization, manganese returns to the

* Samples of metal and slag were taken *after* the corresponding gas samples.

metal-bath, where it effects an elimination of sulphur. In proof of such behavior, the following analyses of samples from the bath, taken March 16th, 1887, may serve:

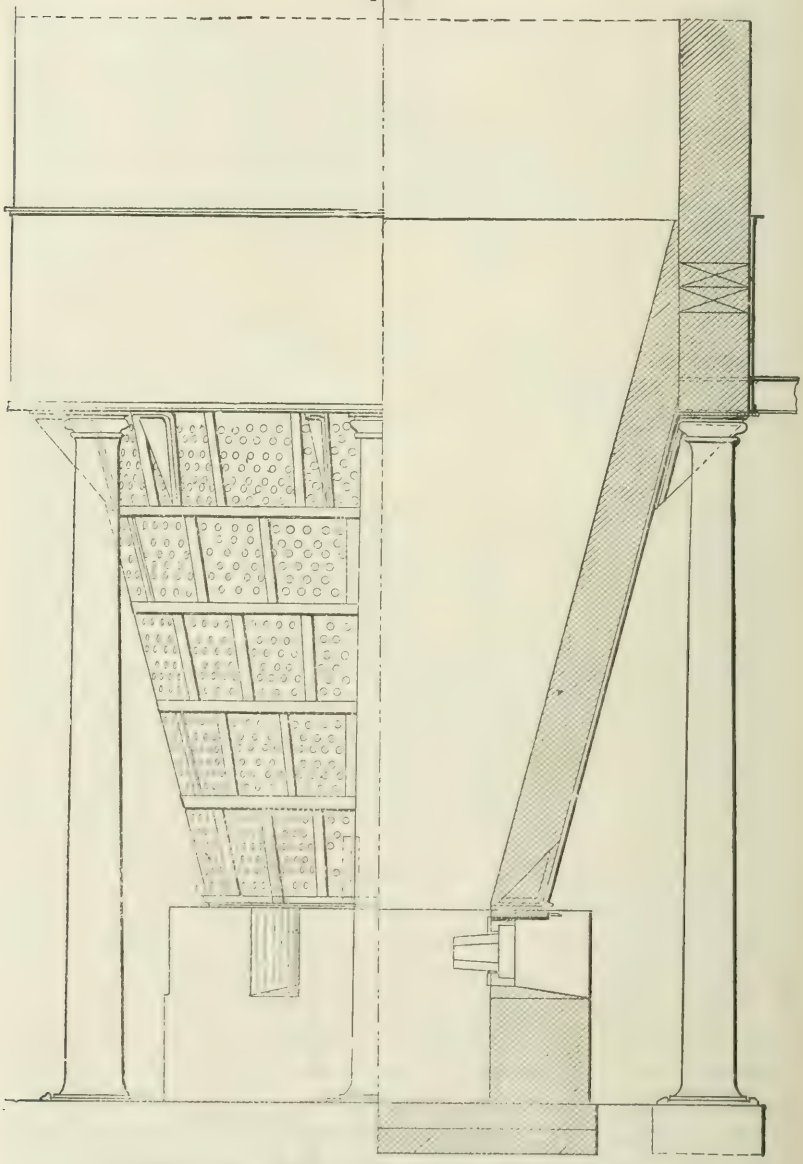
	Mn. Per cent.	P. Per cent.	S. Per cent.
Heat No. 184.—Upon disappearance of spectrum line.....	0.19	2.070	0.133
At second lime-addition.....	0.62	0.463	0.067
Upon addition of FeMn. Steel-test.....	Undetermined.	0.110	0.052
Heat No. 185.—Upon disappearance of spectrum line...	0.24	2.180	0.072
At second lime-addition.....	0.81	0.718	0.042
Steel-test.....	Undetermined.	0.075	0.026
Heat No. 186.—Upon disappearance of spectrum line...	0.24	2.390	0.081
At second lime-addition.....	0.79	0.483	0.047
Steel-test.....	Undetermined.	0.082	0.056

This reaction cannot create surprise, since it is well known that the manganese of the ferromanganese, added for deoxidizing and recarbonizing, likewise induces a removal of sulphur. That sulphur actually volatilizes from the vessel, is indicated by the fact that the slag from highly sulphurous charges contains no proportional increase of calcium sulphide, so that the elimination of sulphur from the metal exceeds what is thus accounted for in the slag. More striking than the volatilization of sulphur is that of phosphorus. It depends on temperature, and at times, with hot heats, reaches 30 to 40 per cent. of the total phosphorus; but with cold heats this elimination ceases almost entirely. Tests have determined that, with like percentages of phosphorus and like quantities of lime, the resulting slag has shown at one time twenty parts of phosphoric acid to fifty of lime, and at another time thirty of phosphoric acid to fifty of lime, according as the heat ran hot or cold.

All attempts at a regular production of high-carbon basic ingot-iron have failed here as elsewhere. Probably, the method of carbonizing the metal by means of carbon, as introduced at the Phœnix works, and previously alluded to, may be a step in the right direction. The ingot-iron of Hörde generally carries 0.5 per cent. manganese and is put on the market with a wide range of carbon, and also with welding-properties, if demanded. For the material used in the current manufacture of rails, ties, etc., a large number of

analyses has shown that, with 0.5 per cent. manganese, 0.07 per cent. sulphur is harmless, while 0.07 to 0.10 per cent. sulphur will

Fig. 25.



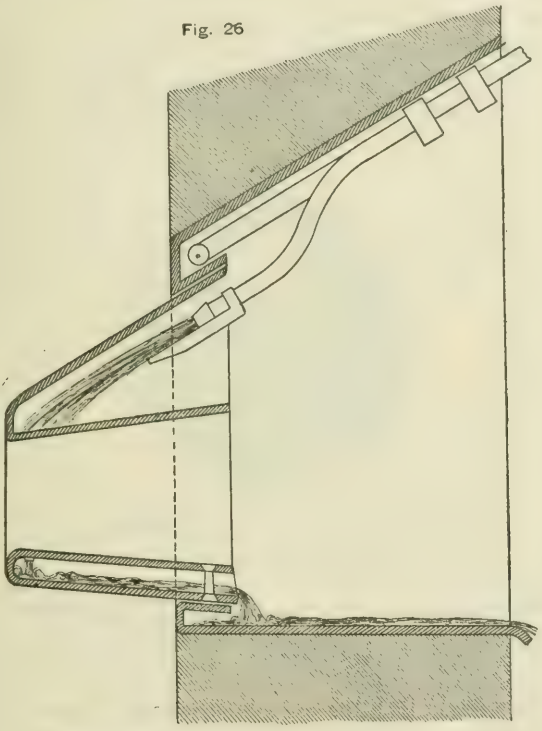
Lürmann's Construction of Blast-Furnace Bosh and Hearth.

betray its presence in the number of rejections in the rolling-mill,

and anything over 0.10 per cent. renders the product useless. These limits have been sharply defined in the course of years by subjecting to accurate chemical analysis every defective or worthless product of the mill.

Phœnix.—The basic Bessemer works at Phœnix, near Ruhrort, were built in 1884, and put in operation toward the end of the same year. The necessity for filling in the ground of the whole plant to a depth of 8 meters increased considerably the difficulty of providing

Fig. 26



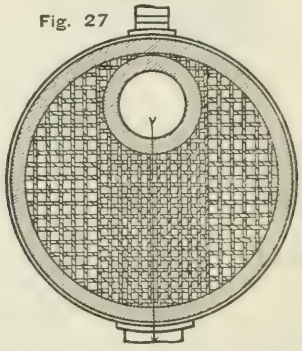
Water-Cooled Tuyere at the Hörde Works.

proper foundations, and led to the design of the lightest possible iron construction for all the buildings. Between the iron beams the walls are panelled with half-brick. Space is limited, and hence the general arrangement which is shown in Fig. 49 (Plate VI.). Direct metal from two blast-furnaces, producing 6000 metric tons a month, can be run to the converter-house by a locomotive. The same track also serves to bring in the remainder of the pig, which is melted in three cupola-furnaces. At Phœnix, the Thomas pig carries 2 to 2.4 per cent.

phosphorus, 1.8 to 2.3 manganese, 0.2 silicon, 2.8 carbon and 0.02 to 0.07 sulphur.

The three converters are served by one central charging- and transfer-crane and by an independent casting-pit. The central crane takes the charging-ladle from its truck and raises and swings it; two men on the upper platform of the crane empty the ladle into the converter; the crane is then lowered, and the blow begins as the vessel is righted. The second arm of the same crane is ready to receive the casting-ladle. Ferro-manganese is thrown into the converter from the platform of the transfer-crane. After a completed blow, when the casting-ladle has been picked up by the transfer-crane, the latter swings into line with a casting-crane, which, in turn, takes the ladle and does the pouring, while the transfer-crane is pre-

Fig. 27



Arrangement of Checkerwork for Hot-Blast Stoves at Friedenshütte, Upper Silesia.

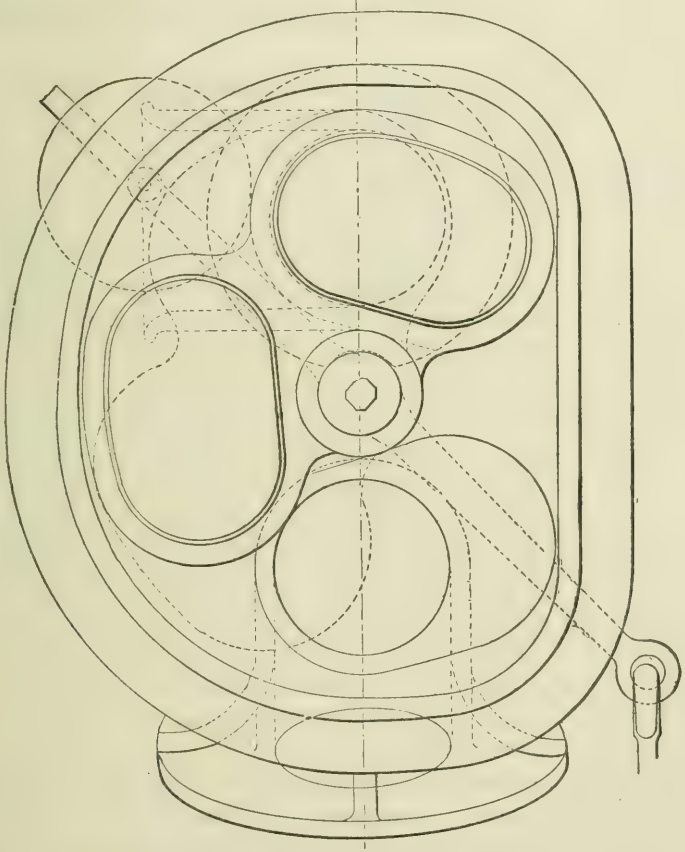
pared to handle a new heat. The 1200-kilo ingots taken out of the moulds are charged into soaking-pits which are not specially fired. Thence they pass to the blooming-train, and, after that, are cut up with heavy shears. The blooms are either sent to the billet or plate-mill, or go to other rolls for working up into a great variety of shapes.

The height of the three cupolas to the charging-door is 5.4 meters, the diameter at tuyeres is 1.4 meter, and that of the shaft 1.8 meter. The level of the hearth is 2.4 meters above the floor. Each cupola melts 35 metric tons per hour, with 700 to 800 millimeters of water-gauge blast-pressure. The two blast-furnaces deliver, in twelve hours, 80 to 90 metric tons of molten pig, which is blown without admixture of cupola-metal. The converter-charge is 11 tons, and the greatest record for twelve hours is thirty-three heats, or 315 tons.

The monthly output amounts to 12,800 tons. The duplex blowing-engine has air-cylinders of 1450 mm. diameter and 1700 mm. stroke; it is fitted with expansion valve-gear, takes steam at 90 pounds, and delivers blast at 27 pounds; its speed is thirty-six revolutions per minute.

For the hydraulic machinery there are ten air-accumulators.

Fig. 28.



Front-View of Gas Cut-off for Blast-Furnace.

Thirty-six non-fired Gjers soaking-pits receive the ingots, which usually measure 410 mm. square. The aggregate yield of the pits has reached 395,000 tons. The rolls of the reversing bloom-train are made of ingot-steel 900 mm. in diameter and 2500 mm. long. The driving-engine has cylinders of 950 mm. diameter and 1250 mm. stroke, and works with 90 pounds of steam. The capacity of

the mill amounts to 325 metric tons in twelve hours; 12,600 tons of blooms per month are rolled, ranging from 200 by 180 mm. to 130 mm. square in cross-section.

The transfer- and the casting-crane have each a central supporting column and two cylinders, one for the live and the other for the dead load. All motions, including the swing, are produced by hydraulic power, controlled by an operator on the crane. The same is true for all the soaking-pit and shear-cranes, an operator being stationed on each. Piston valve-gear is used for all hydraulic valve-motions.

The interval between tapping the pig-metal and filling the converter lasts three minutes for cupolas, and about twelve to fifteen minutes for the blast-furnace, though exceptional delays of two and a half hours may occur without creating difficulties in the blow. From the time of charging the converter till the cast is completed, sixteen to eighteen minutes elapse; of this five to ten minutes is required for casting, stripping and picking up the ingots. The ingots remain in the soaking-pits for thirty to ninety minutes. The interval between lifting them out and cutting them into blooms may cover five to twelve minutes, according to their size.

Peine.—The basic converter-plant at Peine, shown in plan, Fig. 50 (Plate VII.), is served by four cupolas that run with fan-blast. Four ten-ton converters make as high as twenty-eight blows in a twelve-hour turn. The pig carries 3.6 per cent. carbon, 2.7 phosphorus, 2 to 2.5 manganese, 0.06 silicon and 0.04 to 0.06 sulphur. The cupola-coke amounts to 6 to 6.5 per cent. of the weight of pig melted. More skilful mixing of the pig for the converter has permitted a reduction of the lime-addition since 1883 from 19.5 per cent. down to 15.2 per cent. of the weight of ingot-metal. The consumption of chills amounts to 17 kg. per metric ton of ingot-metal, the products being billets measuring 100 to 200 mm. on a side, and slabs 150 to 350 mm. thick, 200 to 800 mm. wide and 700 to 1400 mm. long. The loss by oxidation in the converter has also been reduced from 15.22 per cent. to 13.35 per cent. On the other hand, the output, which was 13,520 metric tons in 1882, rose to 66,450 tons in 1885 and 101,893 tons in 1889, while this year it promises to reach 116,000 tons. The only iron used is Ilsede pig.

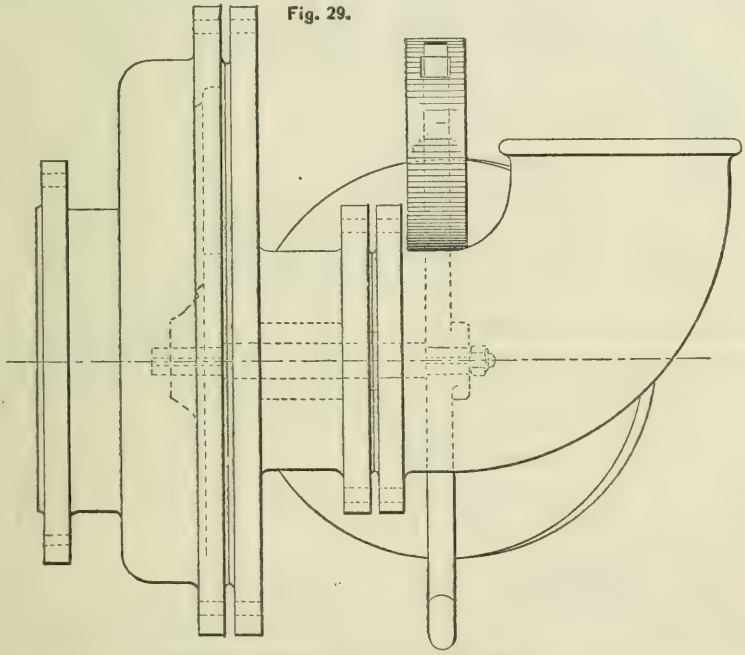
As regards the durability of the vessels, the lining which used to be worn out by 56 heats in 1882 now stands 162 heats, and the life of the bottoms has increased from 11 heats in 1882 to 28 heats at present.

The duration of the blow now averages 10.68 minutes and the

dephosphorizing period (after-blow) 3.55 minutes. In the past year the works produced 12,000 tons of crude ingots. The rest of its product supplied one 640 mm. three-high billet train; one 860 mm. reversing plate-mill, and one rod- and merchant-mill having a 525 mm. roughing-train, a 420 mm. middle-train, a 350 mm. fast train, and a 260 mm. finishing-train. One large rail- and beam-mill designed to have two three-high trains with roll-diameters of 750 and 850 mm. respectively is nearing completion.

In a twelve-hour turn the billet-train, served by a single heating-

Fig. 29.



Side-View of Gas Cut-off for Blast-Furnace.

furnace, makes 100 tons of billets from 200 mm. blooms that weigh 350 kilograms; the ingots are brought to the furnace hot by a locomotive-transfer from the converter-house. The coal consumed for reheating amounts to 4.5 per cent. Last year's production was 47,000 tons of billets, with a waste of 2.7 per cent. The plate-mill turned out last year 6390 metric tons of plate varying from 5 to 30 mm. in thickness. The monthly record for March was 1050 tons. The product is principally used for boiler-plate. The work of the ingot-metal rod- and bar-mill last year amounted to 14,441 tons of rods, 1315 tons of flats, 1242 tons of shafting ranging from 90 to

160 mm. in diameter, 5128 tons of mine-rails, and 1514 tons of merchant bar. Altogether 77,788 tons of pig-iron were used last year for the manufacture of ingot-metal. The ingot-shipments covered 5707 tons; the product amounted to 70,639 tons of ingots, and besides 34,268 tons of billets, 4015 tons of plates and 13,101 tons in the rod- and merchant-mill.

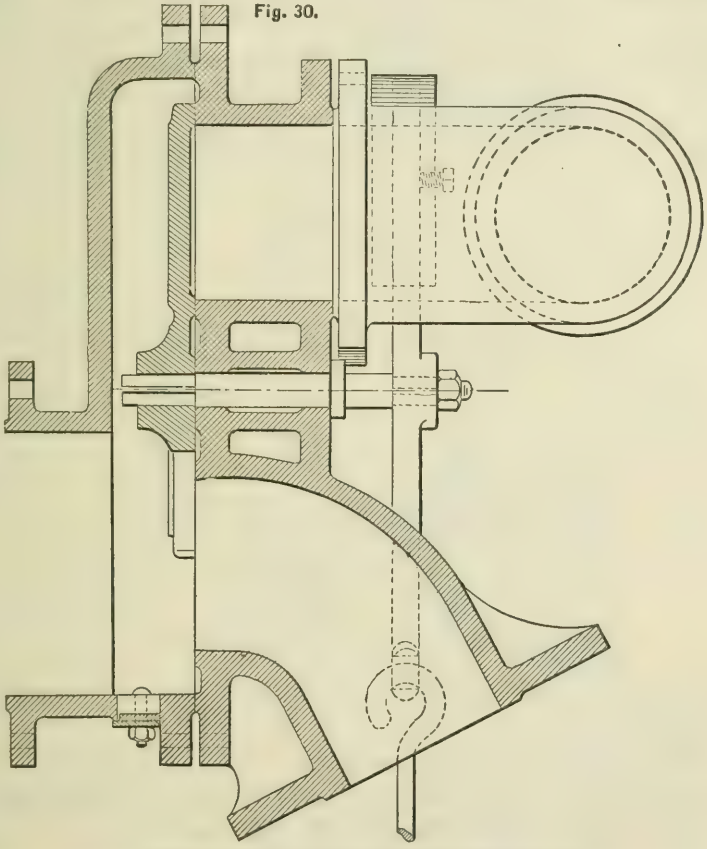
2. *Open-Hearth Metal.*—The manufacture of ingot-iron in the open-hearth furnace is generally conducted in Germany by the Martin process, according to which pig-iron is decarbonized by malleable scrap—commonly the crop-ends and rejections of the mill's own production. Ore or hammer-scale is employed only in small measure, usually as auxiliary to complete the decarbonization. The regular pig-and-ore or Siemens process, if used at all, finds application only in isolated cases and to a very small extent. Equally limited in its introduction has been the practice of blowing air into the metal; the special, and, as it proved, perfectly practicable apparatus shown in Figs. 51 to 53, which was used for some time in the Phœnix works at Ruhrort, has since been abandoned because economic advantages for it could not be demonstrated. The process is as frequently acid as basic; the latter method, however, is continually gaining ground, and it is especially interesting to observe that such works in particular as aim only at small output, or have but small quantities of scrap to work up, prefer the basic process. To such extent, indeed, is this true that even in ordinary machine-shops and foundries 1.5 to 2 ton open-hearths are found in successful operation.

At the present day all furnaces for acid as well as basic working are designed with a high roof—a silent recognition of Frederic Siemens's principle of free-flame development. The regenerators are set below the general floor-level, but with free space between them and the bottom-plates of the hearth; less frequently they are placed horizontally in front of the furnace, and, in a few cases, as at Phœnix, they are of cylindrical design, standing independently. The substitution for checker-work of flues as used in blast-furnace stoves, is steadily gaining favor.

The hearth-bottom is commonly prepared with tar-dolomite, though in recent practice calcined magnesite, dolomite or limestone, with milk of lime or of dolomite as binding-material has also been employed. Repairs are made with a paste of tar-dolomite. As a rule, both bridge-walls are air-cooled; water-cooling is exceptional.

Attempts have been made, and not without encouraging success, to water-cool the zone most subject to wear around the furnace at the level where slag comes in contact with the walls. The roof is still made in all cases of silica-brick, and a mixture of chrome-iron ore and tar generally serves to separate it from the basic hearth. Pure magnesite brick or a layer of rammed magnesia is used more or less

Fig. 30.



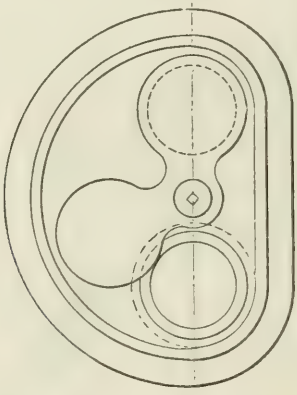
Section of Gas Cut-off for Blast-Furnace.

for the same purpose. Entire hearths of magnesia, on the other hand, are uncommon. The light advantages which they may possibly possess over dolomite hearths do not compensate for their high cost in Germany. The roof as well as the ordinary fire-brick walls which carry it are always specially supported so that no load from the roof is brought upon the hearth or upon the separating layer.

The size of the furnaces has steadily grown, though in Germany fifteen tons capacity has rarely been exceeded.

As a rule, uncalcined limestone serves for the basic addition ; it is charged at the same time with the whole quantity of pig and scrap, the latter being commonly introduced between the two halves of the pig-charge. The earlier practice of charging the scrap in several lots is dropping more and more into disuse ; indeed, not the slightest disadvantage can be traced to the newer method. Every effort is made to shorten the charging-period which, in the case of large charges, occupies much valuable time ; at Phœnix, near Ruhrort, for example, the charging consumes 2 to 3 hours, during which the first part of the charge already melts. The remedy has been sought either in a greater number of charging-doors, with a correspondingly

Fig. 31.

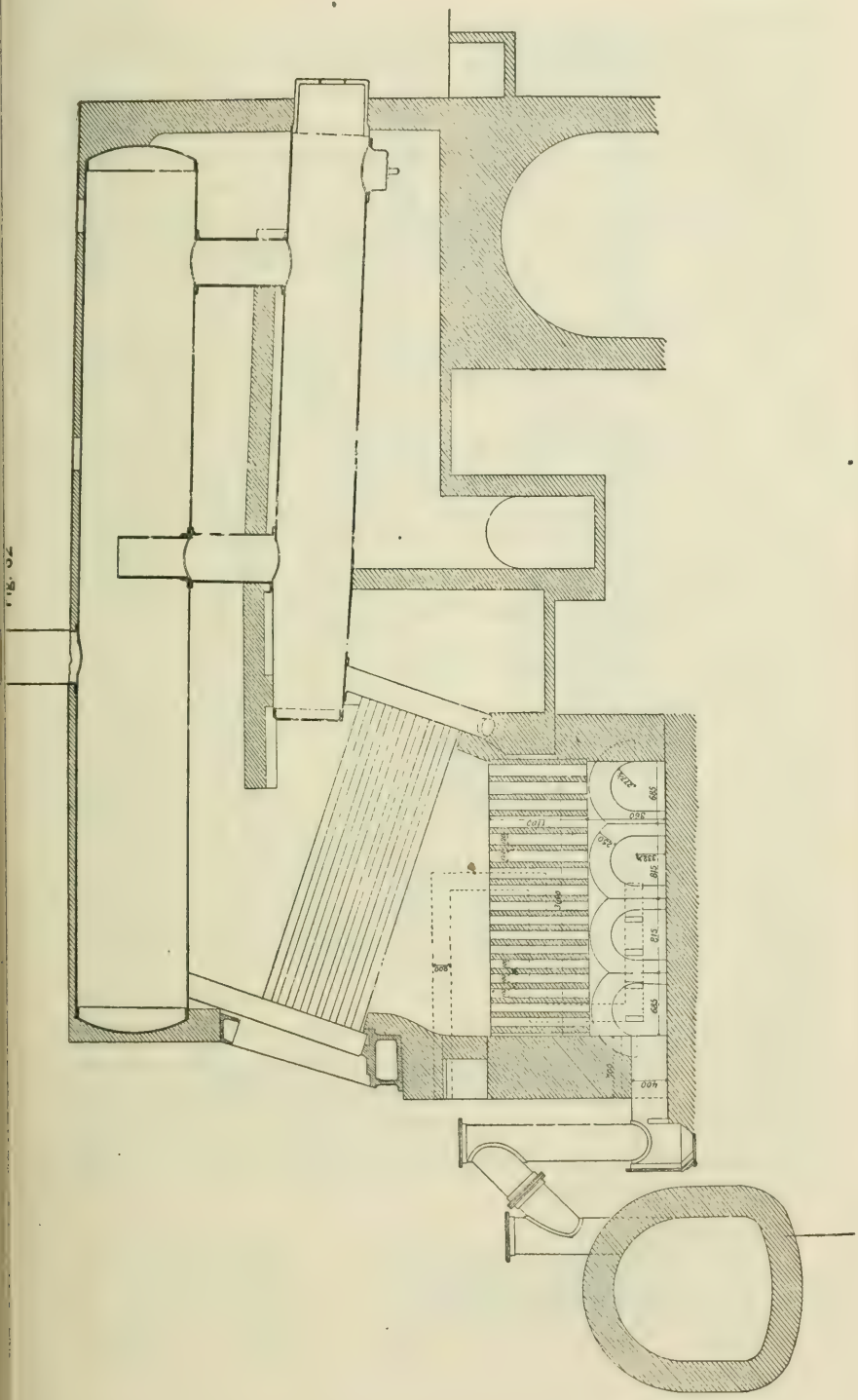


Spectacles in Gas Cut off for Blast-Furnace.

larger force of men, or in the adoption of hydraulic charging-machinery.

Before the addition of ferro-manganese, the slag is drawn only when it carries much phosphorus. If low in phosphorus, the relatively thin and diffused cover of slag is not a disadvantage.

Basic open-hearth metal is generally regarded as an excellent material, for the reason that it is practicable to make accurate tests during its manufacture, and, since ordinarily not more than 3 or 4 heats can be run in twenty-four hours, there is sufficient time for both chemical and mechanical examinations. A by no means unessential cause of the common preference for open-hearth iron over converter-metal lies in the better opportunity offered by the open-hearth for a uniform distribution of the manganese introduced as ferro or spiegel. For the lack of such opportunity much difficulty has heretofore been



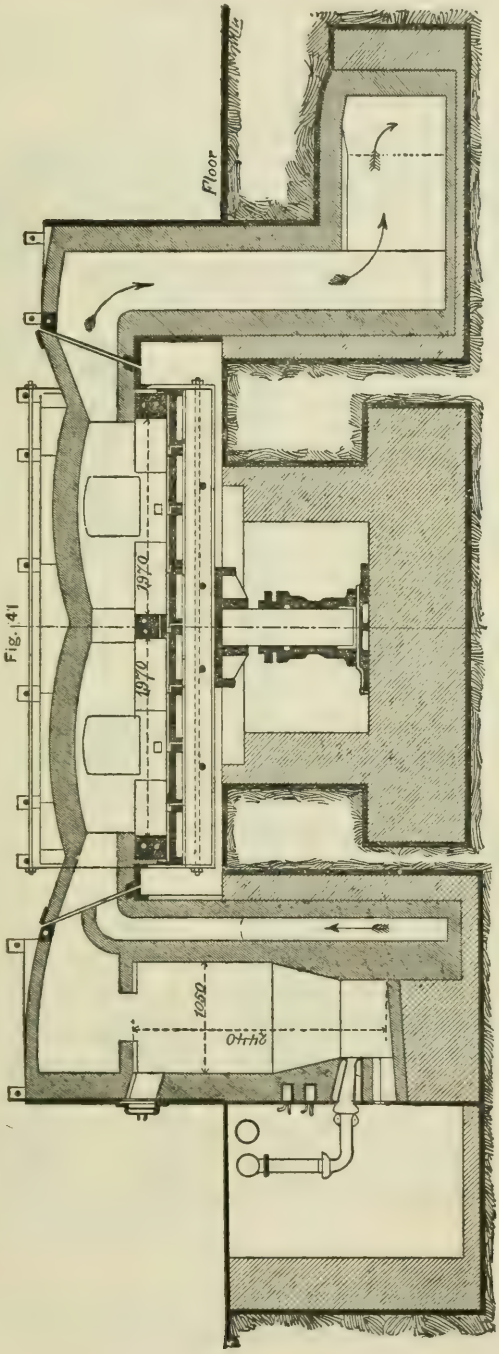
Longitudinal Section of Gas-Fired Boiler at Friedenshütte, Upper Silesia.

encountered in producing high-carbon metal in the basic converter. We may therefore welcome here also as progress the practice introduced and developed at the Phœnix works, in which the bath is deoxidized with a small addition of ferro-manganese, and then further carbonized by introducing a stream of pulverized coal. Not only is economy thus effected in the use of expensive ferro, but at the same time the product runs lower in manganese, and is consequently more homogeneous. Concerning the details of this practice I refer to Mr. Thielen's paper, to be read at this meeting.

The following are illustrations of general practice:

Hörde.—The Hörde works have nine regenerative open-hearths for the manufacture of ingot-iron; six are in constant operation and three are undergoing repairs, warming up, etc. The fuel is water-gas. The newer furnaces measure 8 by 3 meters, and have a capacity of fifteen metric tons, though the charge does not commonly exceed ten tons. The capacity of the older furnaces is seven tons. The hearth is made of dolomite, and the sides and roof of silica-brick manufactured at the works. The newer furnaces stand 300 to 350, and the older ones 180 to 250 heats. They are run with scrap and about twenty to twenty-five per cent. of pig. Ore is used (Siemens process) only in small quantities, so far as it may serve to correct the carbon-contents of the bath. If the scrap is in convenient form, six heats in twenty-four hours can be made in the newer furnaces. These furnaces are also fitted to combine the Thomas and the Martin processes. For this practice Thomas metal is blown down to 0.3 per cent. phosphorus in the converter, to the limit, that is, at which the second lime-addition would have to be made, and no considerable oxidation of iron has yet occurred. The Thomas metal is then conveyed in a ladle to the open-hearth and there finished. One furnace can easily make eight to nine such heats in twenty-four hours, and the Thomas metal is thereby converted into excellent open-hearth iron. The choice of this practice depends on the supplies of scrap and on the excess of converter-output over rolling-mill capacity. The Hörde basic open-hearth iron is always, apparently, of better grade than the converter-metal, even when no analytical difference can be detected. The following comparison is to the point:

Thomas iron (Converter-metal). Heat 265 contained:		Open-hearth Ingot-iron. Heat 202 contained:	
Phosphorus,	0.042	Phosphorus,	0.046
Manganese,	0.595	Manganese,	0.580
Carbon,	0.135	Carbon,	0.137
Sulphur,	0.037	Sulphur,	0.046



Pietzka Furnace.—Longitudinal Section on A B C D, Fig. 42.

It is to be noted that the basic open-hearth process removes little or no sulphur, so that its product averages relatively more sulphur than does converter-metal, as the following figures taken at random from open-hearth heats will show:

Heat No.	48.	51.	75.	84.	88.
Sulphur, per cent.,	0.078	0.062	0 066	0.044	0.049
Phosphorus, "	0.049	0.047	0.051	0.045	0.031

This as yet unexplained action is further illustrated by the following samples taken from one heat:

Heat No. 954, *August* 25*th*, 1888.

Sample.	Phosphorus, Per cent.	Carbon, Per cent.	Manganese, Per cent.	Sulphur, Per cent.
1, . . .	0.084	0.242	0.62	0.067
2, . . .	0.080	0.246	0.72	0.066
3, . . .	0.062	0.227	0.59	0.067
4, . . .	0.079	0.215	0.66	0.072
5, . . .	0.064	0.167	0.39	0.079
6, . . .	0.067	0.137	0.39	0.069
7, . . .	0.056	0.135	0.47	0.070
8, . . .	0.050	0.132	0.41	0.076
Steel sample, . . .	0.078	0.154	0.36	0.087

Apparently, open-hearth ingot-iron is less affected by sulphur in its physical qualities than converter-metal. The effort to explain the difference of the two metals in behavior by a difference in specific gravity has failed, the two kinds of ingot-iron having been found to possess exactly the same specific gravity.

Phœnix.—The open-hearth works for ingot-iron manufacture at Phœnix consist of an old plant of four 10-ton furnaces and a new one of two 20-ton furnaces. All the furnaces are basic-lined, and work up all the scrap made in the works as well as malleable scrap of all kinds. The output amounts to 3200 tons per month. All grades of metal, ranging from 0.6 to 0.9 per cent. carbon, are produced and principally turned into tires, axles and high-grade plate- and merchant-iron. Recently, the furnaces have been supplied with gas from ten fan-blown cupola-producers measuring 1650 millimeters in inside diameter and 2800 millimeters above the grate.

IV. Rolling.

As the description of the improvements in mechanical arrangements has been treated in Mr. Daelen's paper, read at the recent

New York meeting of the Institute, attention will here be called only to a few points.

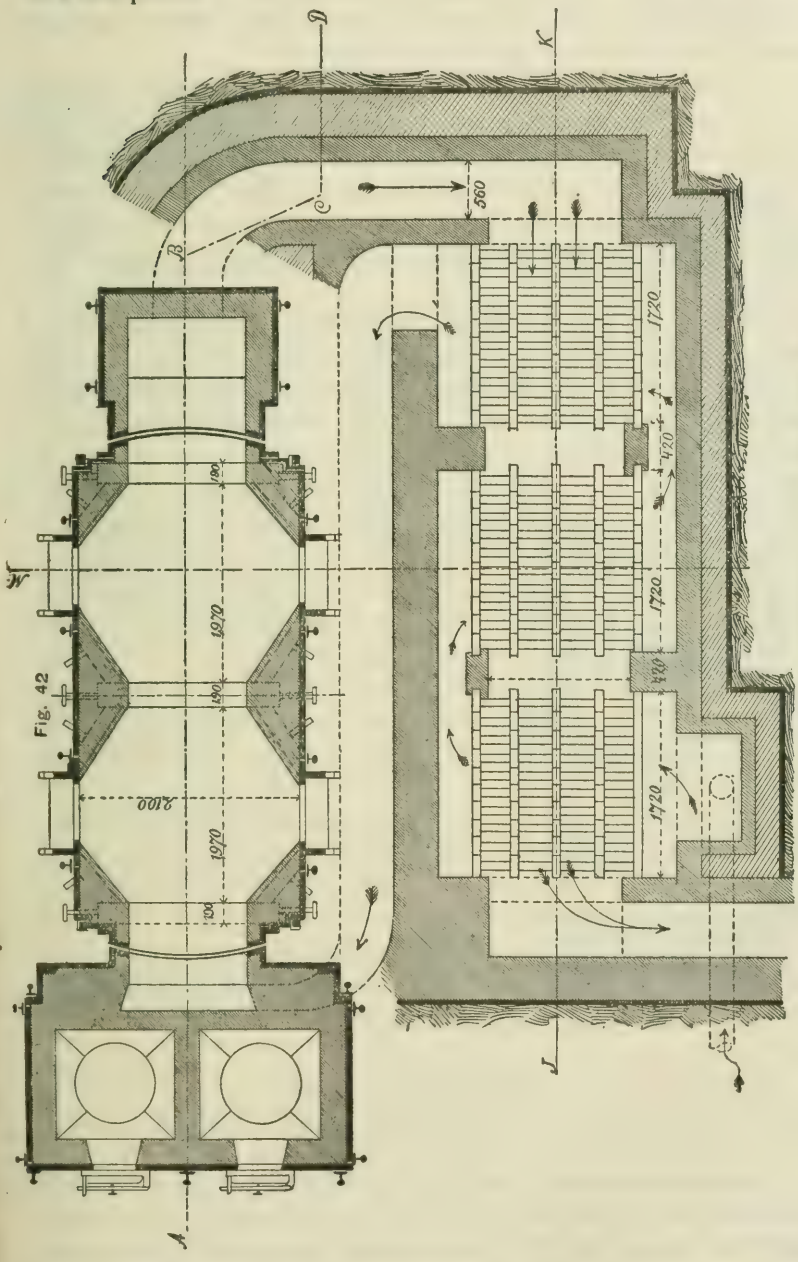


Fig. 42

Pietzka Furnace.—Horizontal Section on E F, G H, Fig. 44.

Ingot-iron is now used in Germany for the greatest variety of pur-

poses. It need not be said that rails are made of it exclusively. For this purpose, however, it is usual to employ only the metal from the converter. Railway-ties are made of ingot-metal, as is also the greater part of the wire manufactured in the country.

For structural shapes, particularly angle- and channel-bars, and U-, T- and I-beams, ingot-iron is more and more used, while for boiler-plate it has long been recognized as superior to wrought-iron. The former prejudice against the basic and in favor of the acid metal has entirely disappeared. No distinction is now made between them, and it is presumed of a low-carbon ingot-iron that it was made by the basic process.

Whether the open-hearth or the converter metal is the better, depends essentially upon the original material. When this is so uniform in composition that it can be blown with great certainty as to the result,—as, for instance, at Peine,—the best commercial irons can be produced from the converter, and even iron for horseshoe-nails not inferior to the Swedish. If the open-hearth furnace is generally preferred for special high grades of product, this is only because it is easier, in the open-hearth furnace, to produce a uniform product from heterogeneous materials, and particularly to avoid in high-carbon metal an irregular distribution of the manganese.

In the casting of ingot-iron there has been no noteworthy change of the old methods. Funnels or sprues are often used, which deliver right and left, etc.; ingots are cast from the bottom or the top, with or without covering, in all of which, so far as I know, there is nothing new.

Attempts to compress the ingot-metal, *i.e.*, to cause a better distribution of blow-holes, whether by mechanical pressure or, according to Krupp's method, by means of carbonic acid, have proved less effective in improving the product than has the increased care which has been taken to secure a uniform chemical composition, and, above all, a precisely adjusted temperature, determined by experience for each class of metal, so that it shall be poured neither too hot nor too cold.

Among the many mechanical devices for giving the desired form to the metal, allusion will here be made to a few machines specially connected with the treatment of ingot-metal.

In the arrangement of rolling-mills, it has been earnestly sought to develop the old-fashioned crude trains into carefully-constructed machines, the gearing of which should not rattle loosely, but transmit power without loss, while the rolls should be placed with fine ac-

curacy, and the pieces to be rolled should be automatically moved back and forward in the precise pre-determined position. In the latter respect, the rolling-mill of the Phœnixhütte, shown in Fig. 49 (Plate

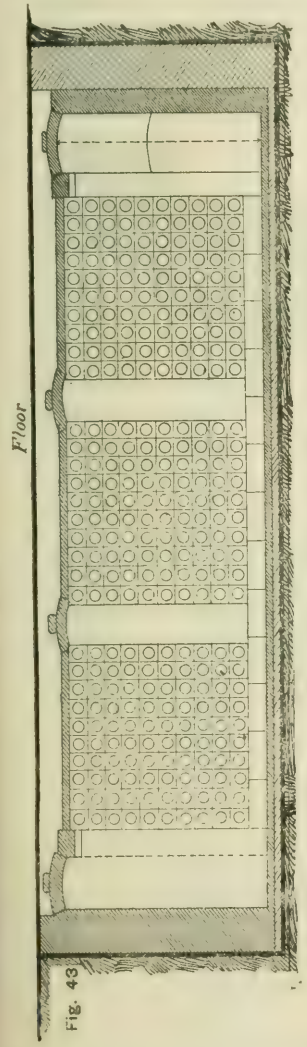


Fig. 43.—Pietzka Furnace.—Longitudinal Section on J K, Fig. 42.

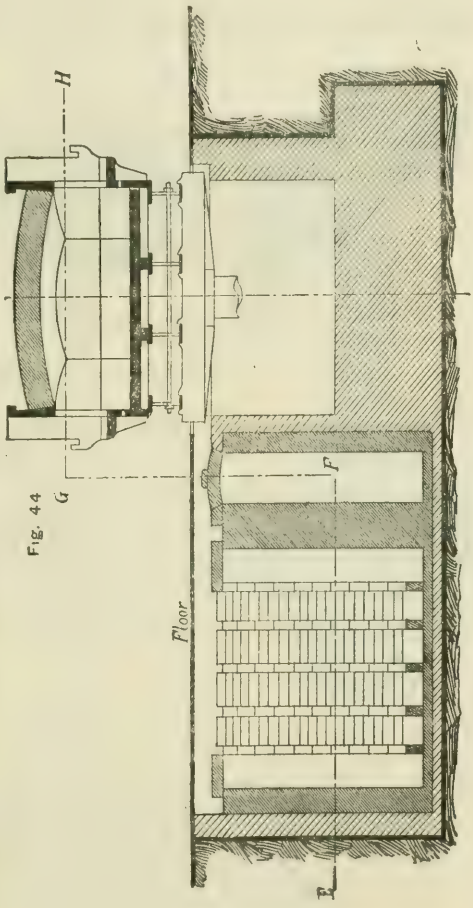


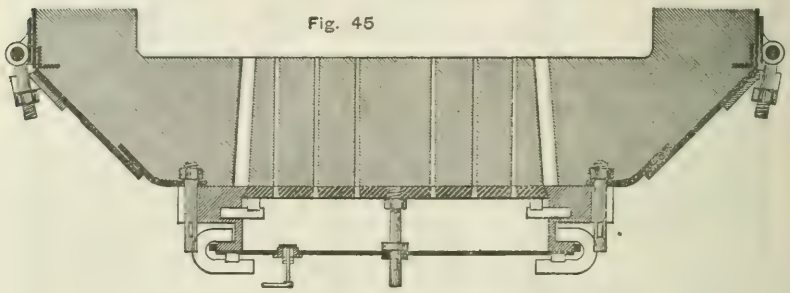
Fig. 44.—Pietzka Furnace.—Cross-Section on L M, Fig. 42.

VI.), may be adduced as a good model. Here, the liquid pig from the blast-furnace is transformed into ingot-iron in the converter, recarbonized and cast into ingots. These are annealed in Gjers soaking-pits, then removed by an automatic lever and delivered to a train of live rollers, which carries them to the blooming-mill, from which they

pass to the bloom-shears and to the rolls without having required, since the metal left the furnace, a single kilogram of coal to be used upon them.

Where room is lacking in the rolling-mill,—which is very often the case in the older establishments, since the requirements as to length of rolled pieces have been steadily advancing,—feed-roller trains are used, which incline upwards and often extend far above the roof (as

Fig. 45



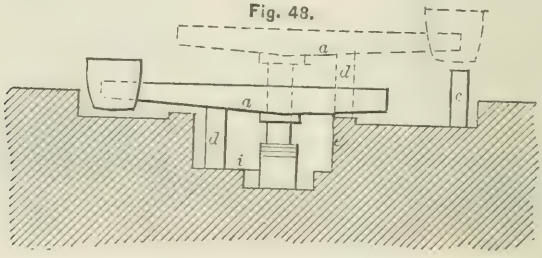
Draw-Bottom for Converter.

at Oberhausen and Phœnixhütte). The space below is thus saved for other uses.

Street-railways have been very widely introduced in Germany. The ingot-metal rails are often rolled solid, receiving only in the last pass the necessary groove, as shown in Figs. 54, 55.

In the manufacture of wire, the most important improvement, doubtless, has been the automatic guides (*Umführungen*) which have

Fig. 48.

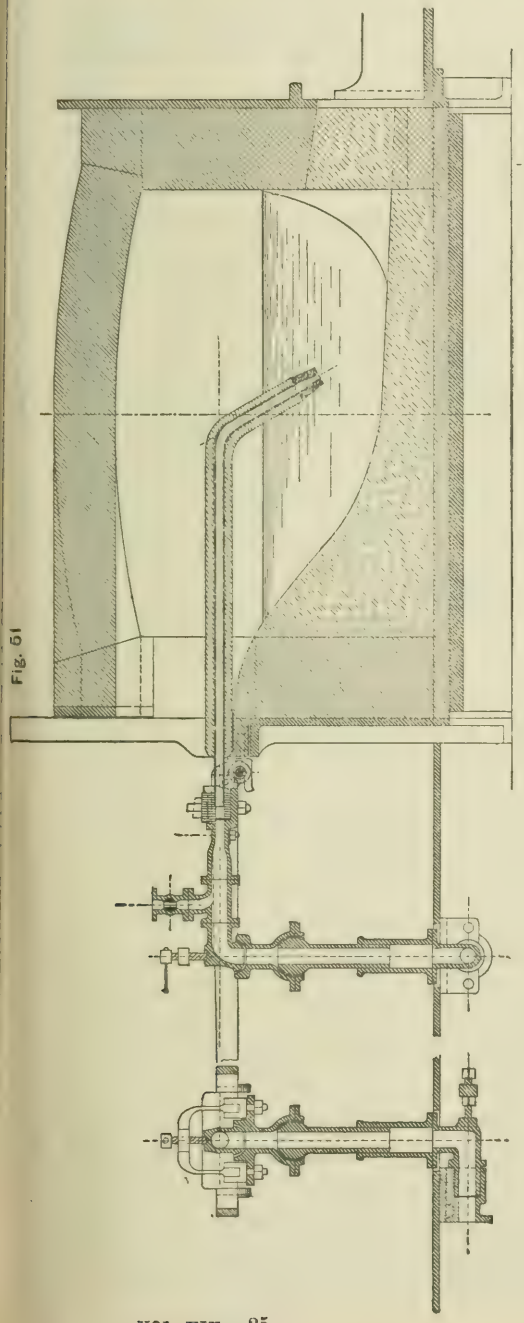


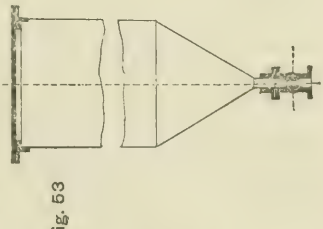
Safety Stop on Steel-Casting Crane.

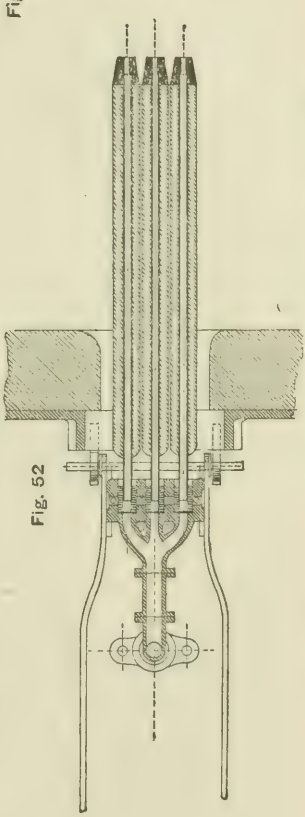
increased not only the product of the works but also the safety of the workmen.

Mannesmann Tube-Rolling.—With good reason may the invention of the brothers Mannesmann, in the manufacture of pipes, be regarded as an important improvement. For pipes destined to meet

severe requirements of resistance to interior pressure, it is indeed







Figs. 51, 52.—Apparatus for Injecting Air into Open-Hearth Bath.
Fig. 53.—Funnel for Use with Coal-Dust.

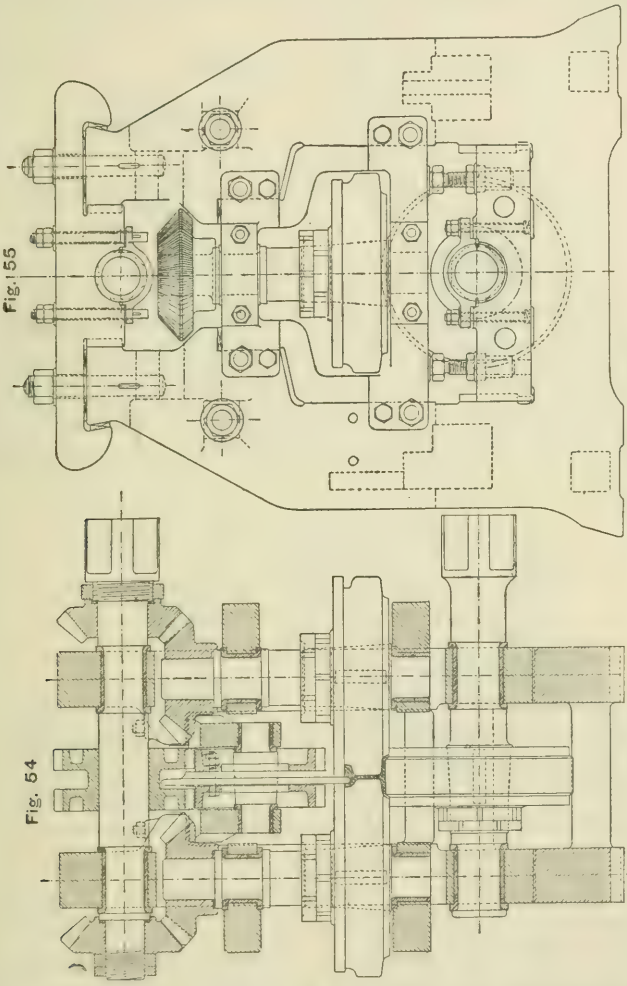
calculated to revolutionize previous practice. I shall therefore attempt to describe it.

The process consists in feeding a solid, highly-heated round bar of ingot-metal between rolls which, while their axes are oblique to the axis of revolution, revolve both *in the same direction*. The metal of the surface of the bar thus acquires an increased motion in a spiral direction, and is drawn over its core, receiving consequently the form of a pipe. It is, indeed, not practicable, without an excessive expenditure of power, to make the interior diameter of such a pipe very large. But it is sufficient that an interior space is created, for there is no difficulty in widening it over a mandril by further operations between rolls in the form of conical disks, revolving in opposite directions. Since, in this operation also, the pipe moves spirally forward and all its parts are spirally pushed and pressed, the metal becomes still denser. It is this spiral arrangement of material which makes the Mannesmann pipes so remarkable, quite apart from the advantage they possess in presenting no lines of welding whatever. Moreover, blow-holes (which are *invariably* present in ingot-iron) are so squeezed out spirally as to make the walls of the pipe completely impermeable. A proof of this is the retention of hydrogen for weeks in a piece of Mannesmann pipe, closed at both ends. Pipes thus made and enlarged have been successfully produced of all diameters up to 400 mm.

The Mannesmann rolls have to be driven at a speed of 300 revolutions and upward, and their relative positions to each other and to the driving-shaft must be made adjustable with nicest precision through a wide angle. These severe requirements in coupling the rolls to the driving-shaft were met by a special design indicated in Fig. 56. Two shafts, a and b, are each provided, at points equally distant from the center S, with two arms that form equal angles with aS and bS, and constitute couples which will remain in constant contact. The touching points s_1 and s_2 slide toward and away from the center S, each describing an ellipse during a complete revolution. In practice this movement is accomplished by setting a bronze semi-cylinder a' into a rounded groove on each driving-arm and making it bear with its flat surface upon a similar semi-cylinder, b', which is fitted into the corresponding driven-arm. The device is ingenious and works well.

Another serious obstacle which had to be overcome by special design lay in the excessive unit-stresses in the teeth of common gearing when transmitting very high powers at high rates of speed.

The form of the new Mannesmann gearing is indicated in Fig. 57. The teeth or pins are shown at a' b'; they are so shaped that one seizes the other like a fork. It is evident that when both wheels have the same number of teeth, and when their axes are parallel to



each other, the forks and the blades must meet accurately, provided the teeth be always kept in a parallel position. In both wheels this parallelism of the teeth throughout their movement is maintained by special appliances not shown. But the position of the axes of the two wheels may be swung at any angle, provided the line A A, Fig. 57, remains vertical. In that manner a right-angle transmis-

sion, sketched on the right in Fig. 57, may be obtained and a large contact-surface still be preserved for the transmission of pressure.

The storage of power required for this process, for which a fly-wheel of ordinary construction was not sufficient, was attained by making fly-wheels with a rim of *wire* to run very rapidly.

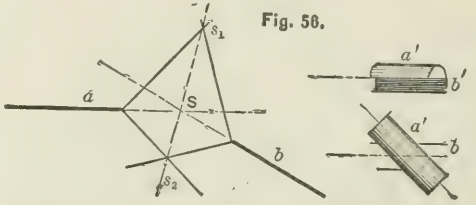


Diagram of Mannesmann's Universal Joint.

The formation of pipes by the Mannesmann rolls is indicated by Figs. 58 to 68. Figs. 58 and 59 show the ordinary rolling process with parallel roll-axes and opposite directions of revolution; Figs. 60 and 61, the same, when the rolls revolve in the same direction. Figs. 62 and 63 show the effect of obliquely-set rolls, operating by

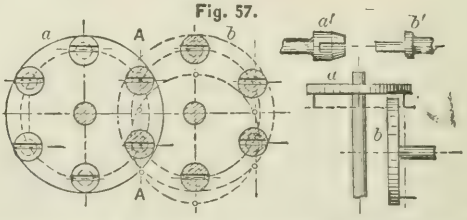


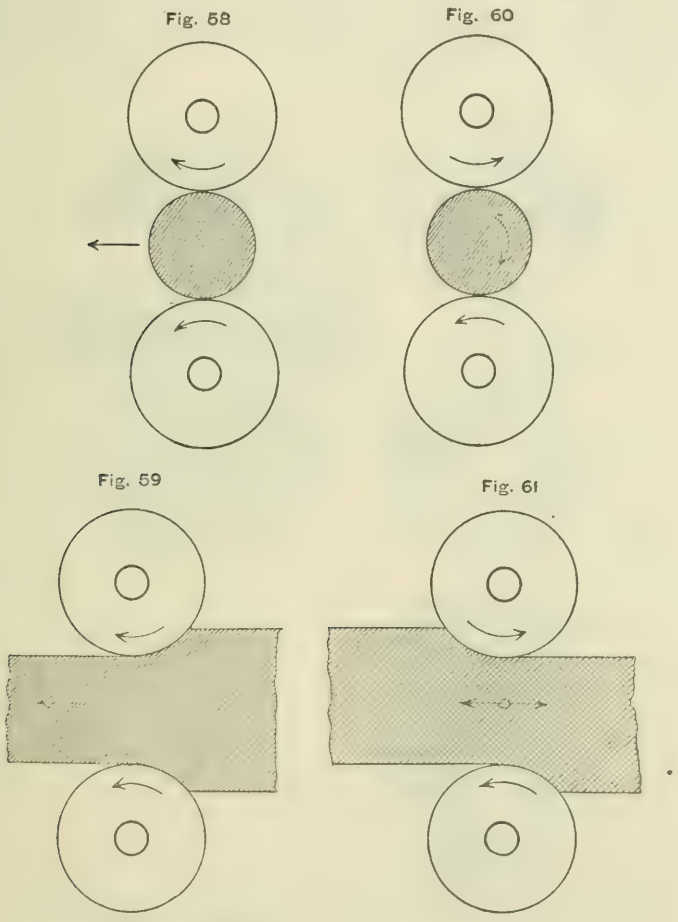
Diagram of Mannesmann's Universal Gearing for Heavy Power-Transmission.

simple friction. Figs. 64 and 65 show the formation of pipes, under similar circumstances, but with the added effect of pressure, *i.e.*, the squeezing forward of the several particles of metal. Figs. 66 to 68 illustrate the enlargement over a mandril by means of conical obliquely-set rolls. Figs. 69, 70 and 71 show the latest arrangement in actual practice. The circular cross-section of the Mannesmann tubes is easily changed in rolling to an I-section, a square, a rail-section or almost any desired form.

V.—Chemical, Mechanical and Microscopic Investigations of Iron.

The progress of the metallurgy of iron in Germany has only been practicable with the help of science. Chemistry, physics and me-

chanics have furnished the foundations for its successive improvements. Sometimes new processes or devices have been directly deduced from the results of scientific inquiry; sometimes it has been the function of science to seek the principles underlying practical improvements, and thus to render their further development pos-



Diagrams Showing the Action of Parallel Rolls.

sible. There are consequently many novelties worthy of mention in the mechanical as well as the chemical investigation of iron. With regard to testing-machines, more perfect apparatus for measuring strains has been a desideratum. To the perfection of such mechanism American example has contributed not a little; and the principles of Mr. Emery's machine, in the Watertown arsenal, have

received full recognition. Above all, the earnest tendency, departing from former rules, is not to draw conclusions from single results, perhaps of a purely accidental character, but to make observations systematically, in series, and upon as many samples as possible, for the elimination of accidental errors; and also, to establish the most suitable method of testing, so that, even if errors be involved, they need not vitiate comparisons where the same methods have been employed.

Fig. 62

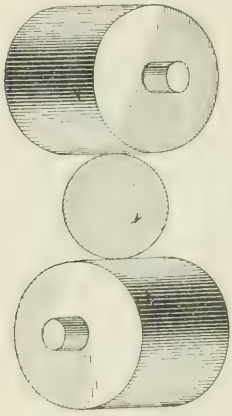
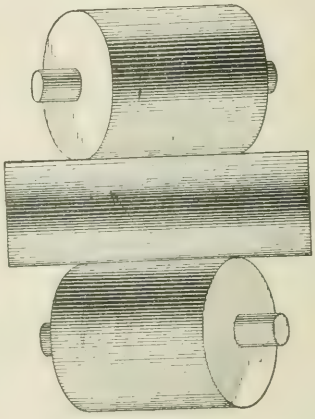
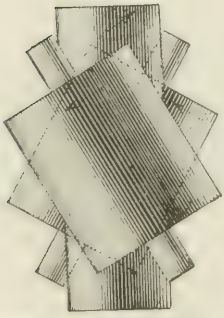


Fig. 63



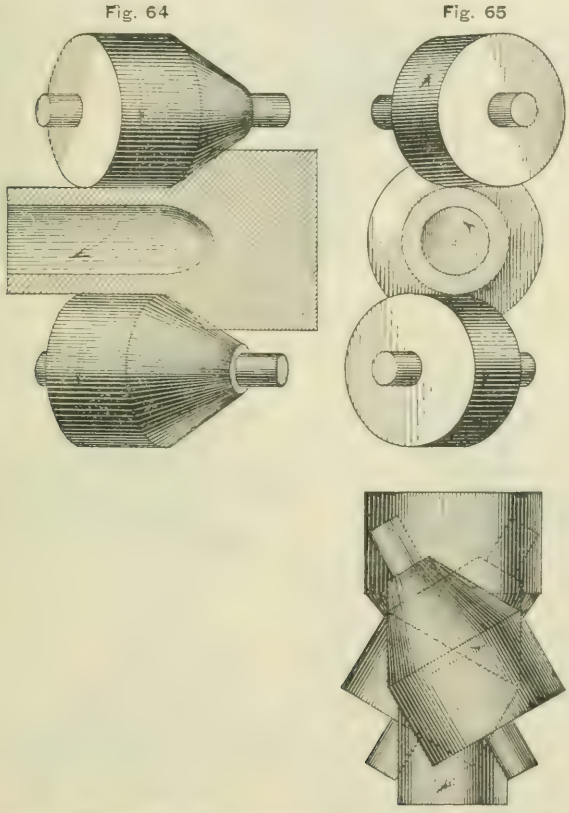


Diagrams Illustrating Mannesmann's Process for Rolling Tubes.

The latter tendency may be asserted likewise for the special branch of analytical chemistry, which deals with the ores and products of iron. Here, also, we have to report not so much the inventions of wholly new methods, as the determination of the sources and limits of error, which are and always will be involved in the different methods employed. It is, undoubtedly, more profitable to have steady results obtained by a known method with a known probable error, than to

deal with data insusceptible of comparison, furnished by unknown methods, with unknown and variable limits of error.

Immense numbers of tests and analyses are now made at our metallurgical works. To take an example: In the laboratory of the Rothe Erde works, near Aachen, there are made in a year, according to the statement of Kintzlé, by two to three chemists and six to seven assistants, 26,500 chemical determinations, and 62,000 bending and tensile tests; so that about 90 chemical determinations

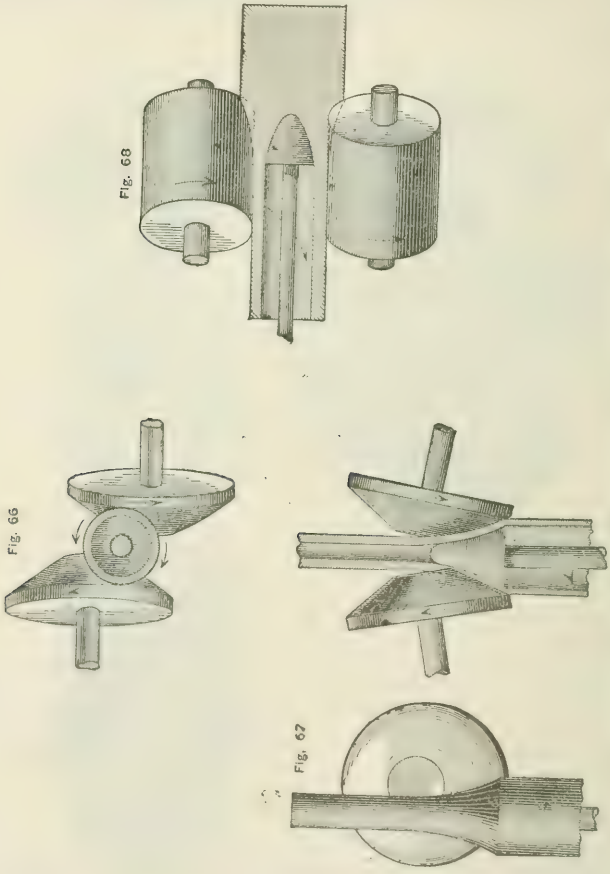


Diagrams Illustrating Mannesmann's Process for Rolling Tubes.

must be made daily. The time required for chemical operations has been proportionally shortened. The different elements can now be determined in the following periods, expressed in minutes: Phosphorus, 30; carbon, 30; manganese, 100; silicon, 60; sulphur (cadmium method), 30. Thus it has been made practicable to maintain a continuous control of the running of works, and even,

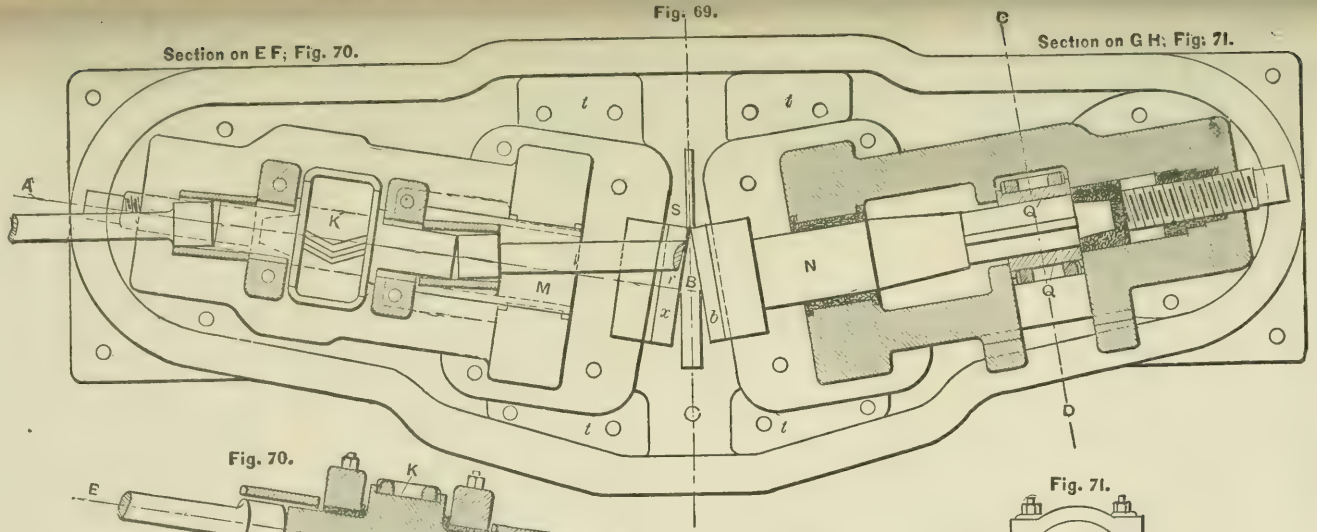
for instance, to wait before tapping a heat in the open-hearth furnace, until its sufficiently complete dephosphorization has been chemically ascertained.

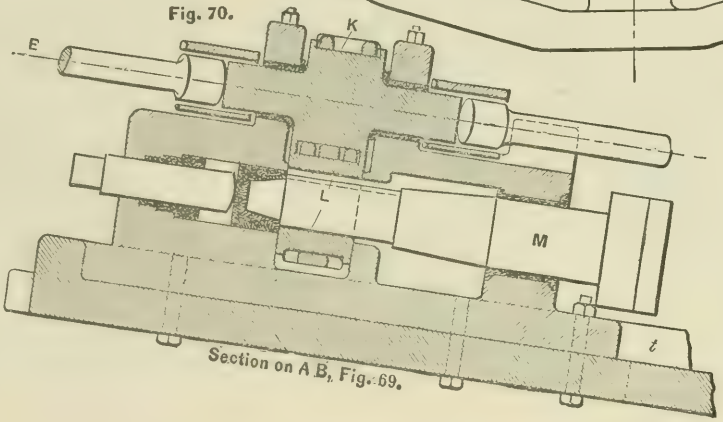
The microscopy of iron is constantly widening its field. The method discovered by Sorby nearly thirty years ago, namely, the examination of carefully-ground sections, instead of the surfaces of

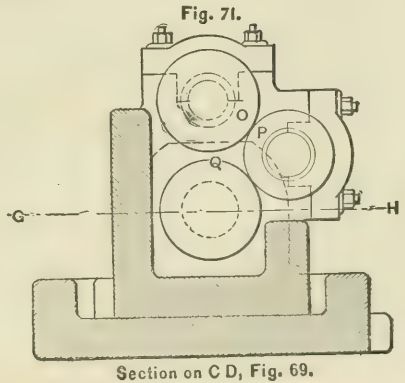


Diagrams Illustrating Mannesmann's Process for Rolling Tubes.

fracture, has been increasingly employed. Samples of metal are ground upon metallic disks, carefully polished, cleaned and etched lightly with very dilute hydrochloric acid, in order to bring out the variations of texture which are always present, even in the finest crucible cast-steel, and which are made distinguishable by the varying depth to which they are attacked. The surface thus prepared is treated to 210° C. (provided this will not, as in the case of hard-





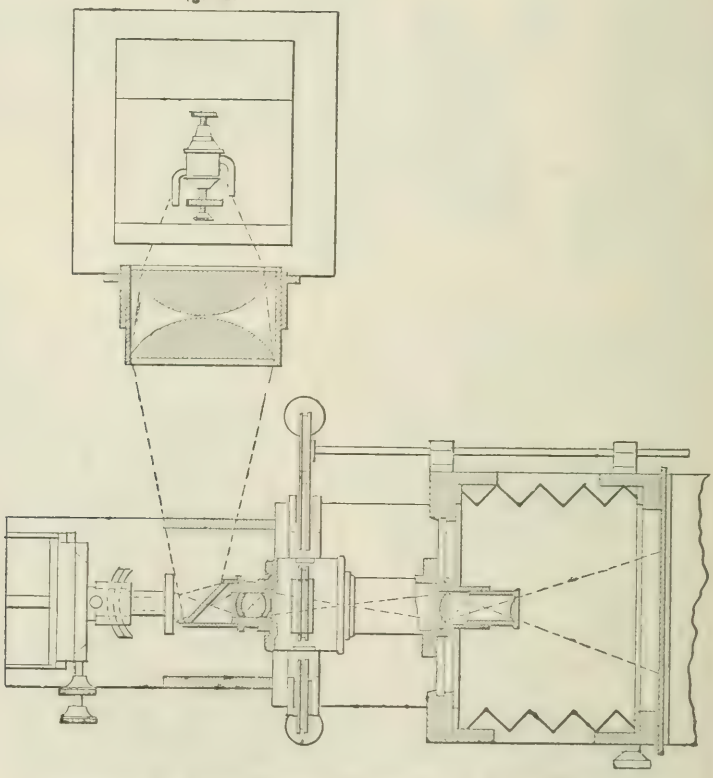


Latest Arrangement of Mannesmann Tube-Rolling Mill.

ened steel, affect its texture), and acquires thereby a variety of coloring which still further emphasizes the various elements in its composition.

How successfully in this way characteristic features of grades of iron can be discovered, comparisons can be made, physical differences coexisting with identity of chemical constitution can be explained, and the true grounds of many disasters can be demonstrated,

Fig. 72.



Apparatus for Perpendicular Illumination in Micro-Photography.

whether in the fault of the material or in the excessive strain to which it was subjected, many series of experiments have already proved.

The aid of photography has been invoked to make the results obtained by single observers the property of all. In photographing sections prepared as above described, only ortho-chromatic plates can be used. A further difficulty, namely, that to get a true photo-

graph, the illumination of the section should be perpendicular to its surface, while at the same time the section must be placed perpendicular to the axis of the microscope, through which it is photographed, has been overcome by the use of a parallel-plane glass plate, which serves also as a mirror. Fig. 72 is an illustration of a convenient apparatus now used for this purpose.

The thirty slides exhibited in connection with the reading of this paper* give a general view of the characteristics detected by the microscope in the different kinds of iron. These views may furnish a good starting-point and guide to those who desire to make similar investigations for the benefit of their own works.

VI. Conclusion.

In conclusion, I may be permitted to say that the outlook for our German iron and steel industry is neither all brightness nor all shadow. On the one hand, the patent-law of 1877 and the tariff-law of 1879 have greatly stimulated industrial activity and technical progress in this business, and the Thomas process, coming just at the right time, has made us independent of other countries for our raw material. On the other hand, the severity of military duty, both withdrawing and estranging from labor for three years every strong and healthy workman, and the compulsory contributions required by law from employers for the insurance (life and accident), medical care and pensioning of employees, constitute patriotic and philanthropic burdens, which we must needs feel in competition with other countries not similarly handicapped. But the ironmasters of Germany make no complaint. On the contrary, spurred rather than daunted by difficulties, they mean, by increased vigilance, energy, economy, technical skill and continuous scientific researches, to maintain the honorable rank of the German Empire among the iron- and steel-producing nations of the world.

Discussion.

Professor Lunge, Zurich: I beg to point out that the Mannesmann-tubes, of which there is such a beautiful exhibit in illustration of Prof. Wedding's paper, are likely to be of the greatest importance for chemical manufacturing, and that in two different directions.

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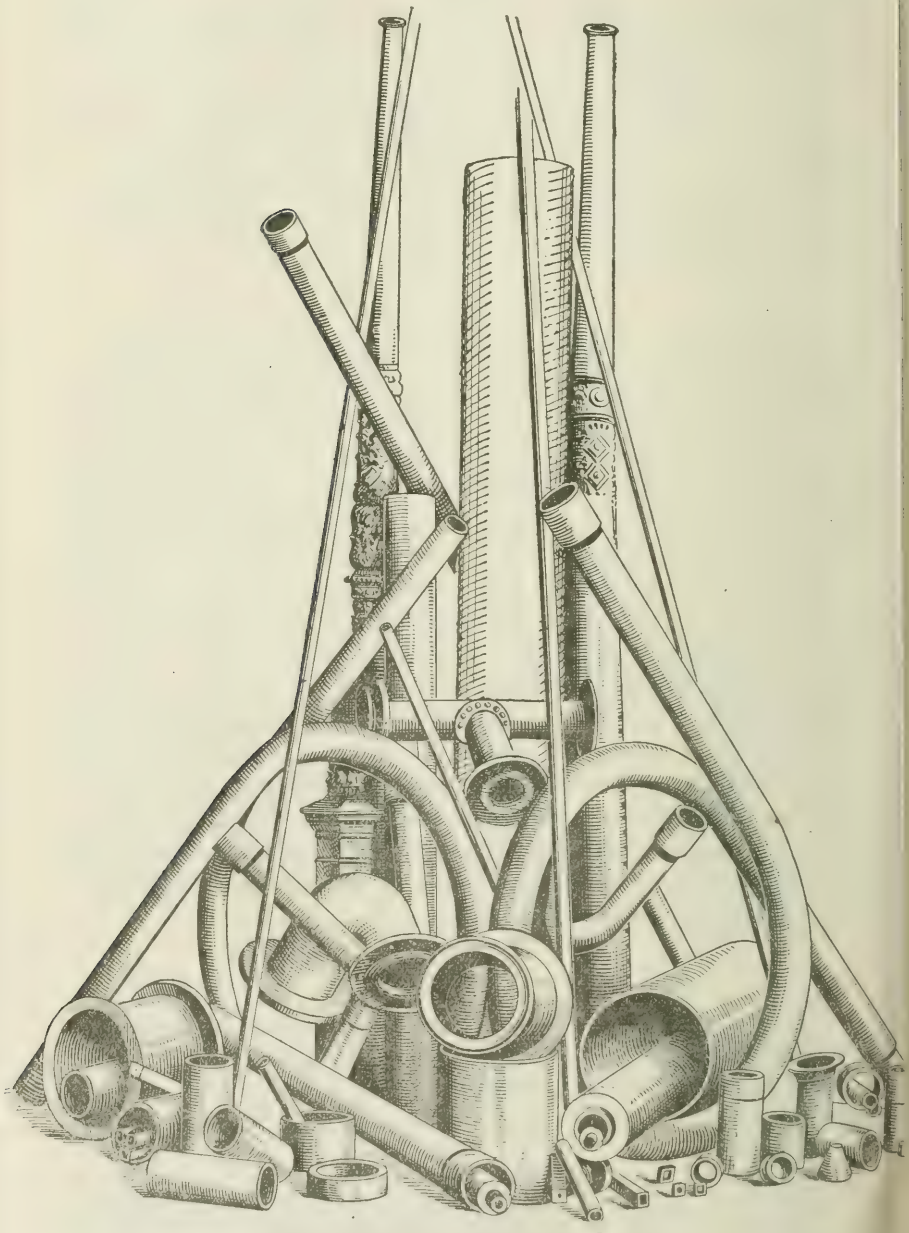
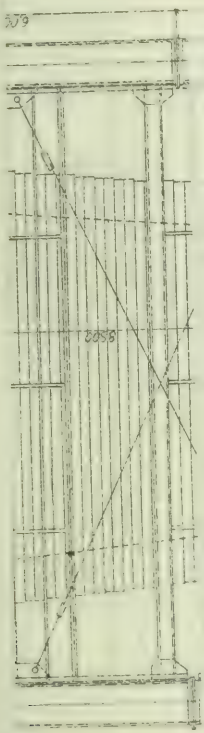
* Photographs of these slides, bound in a handsome album, were presented to the Institute, by Professor Wedding, at the close of the meeting.

In the first place, they seem to supply the best possible material for making high-pressure vessels for storing and carrying liquid carbon-

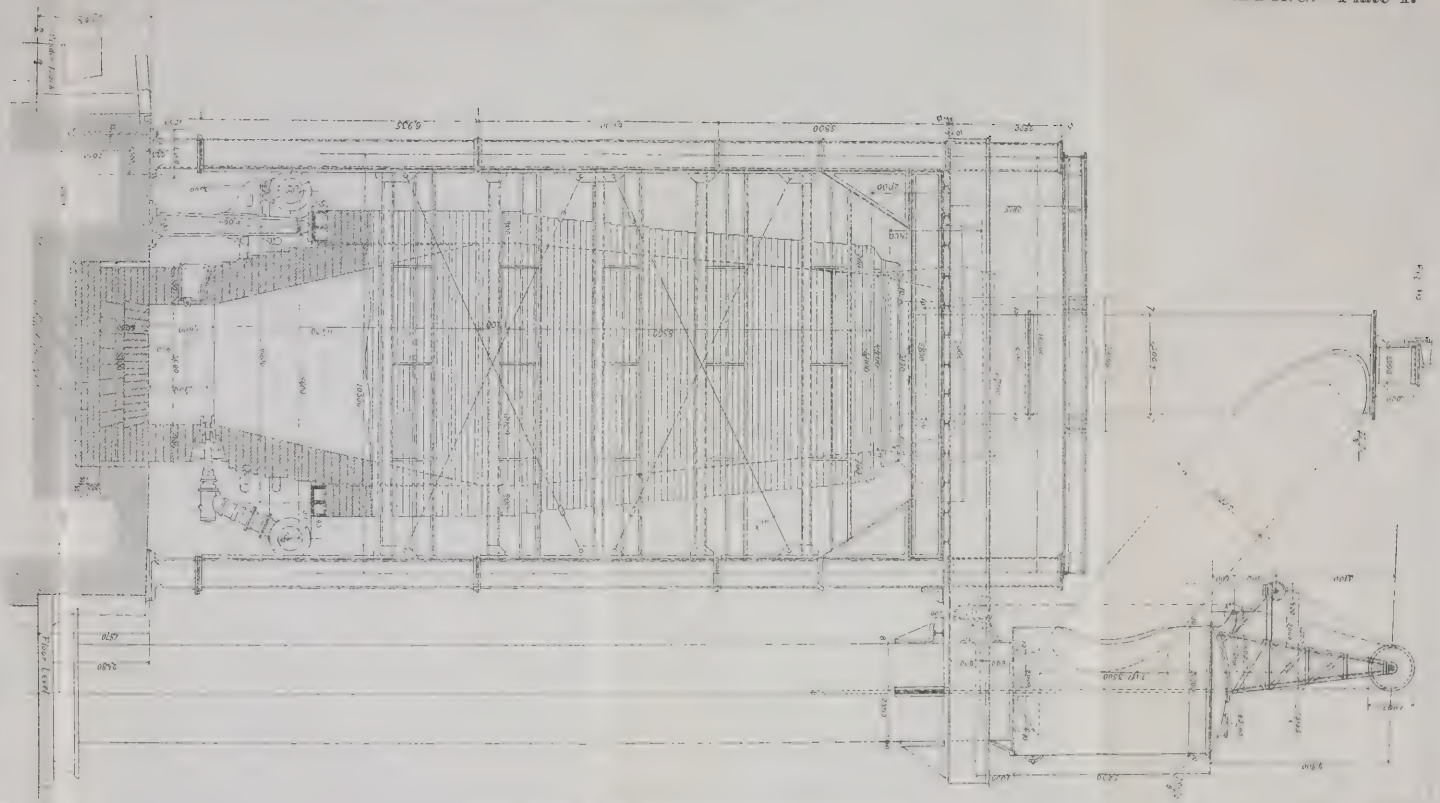
FIG. 73.



Mannesmann-Tubes and Tube-Products.



Blast-Furnace at Fri



No. 1 Blast-Furnace at Friedenshütte, Upper Silesia.

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360
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540
2300

2200

Fig. 33

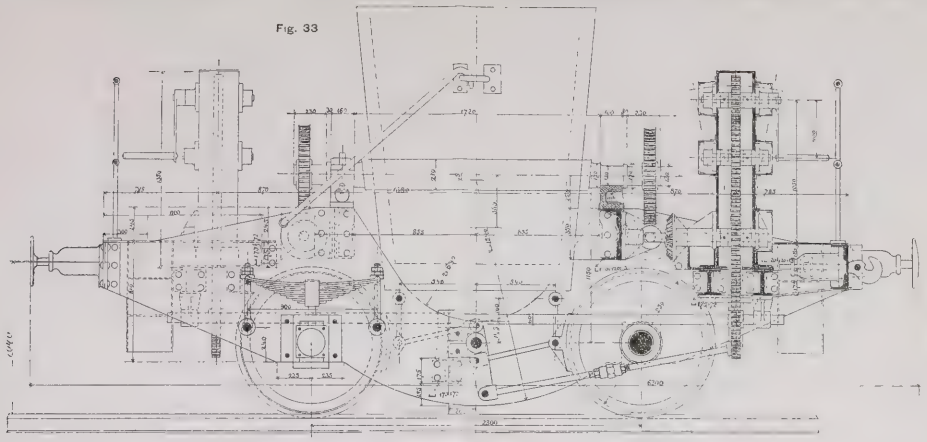


Fig. 35

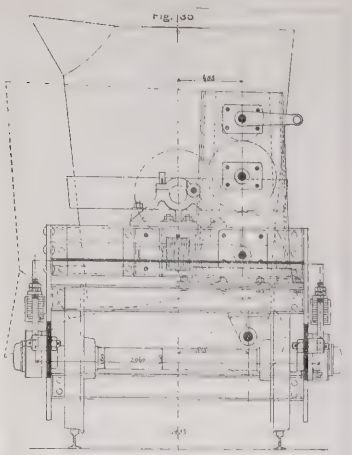
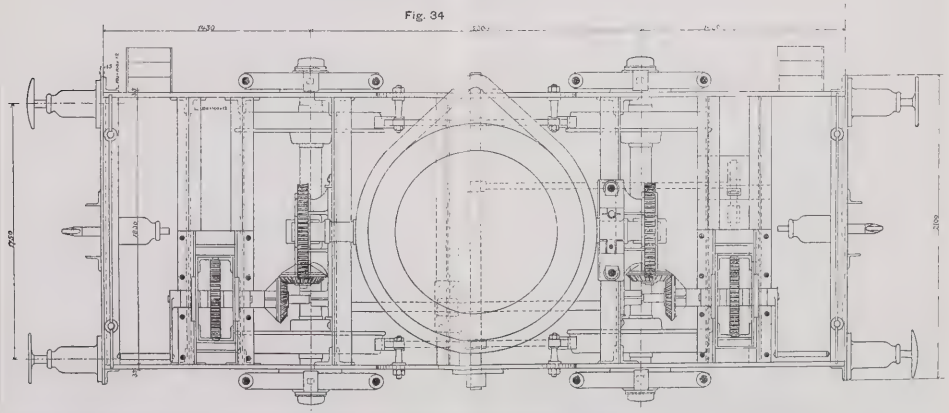
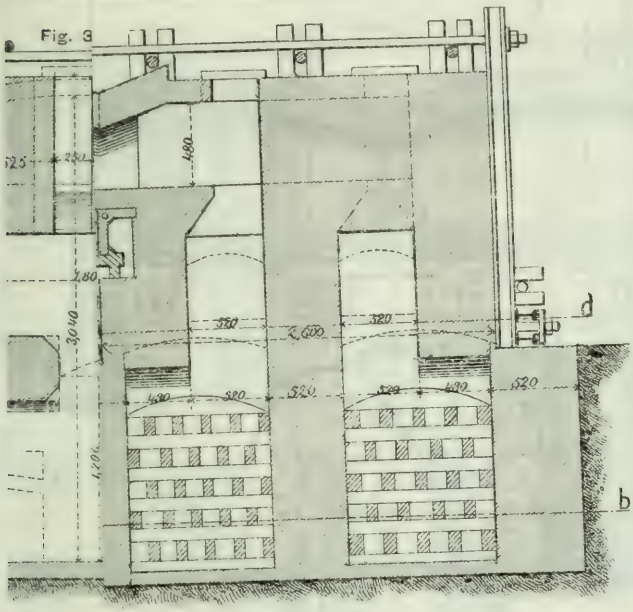


Fig. 34



Ten-ton Ladle-Car at the Hoerde Works, Westphalia.



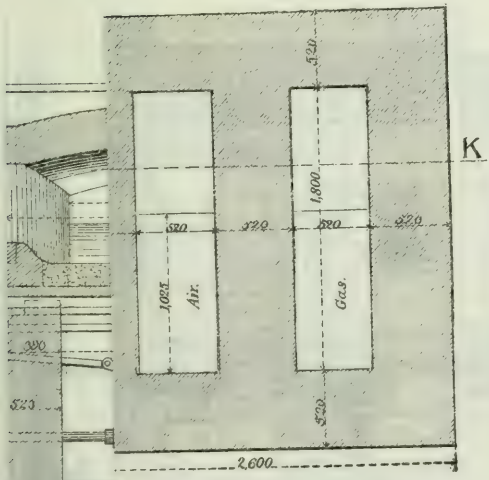
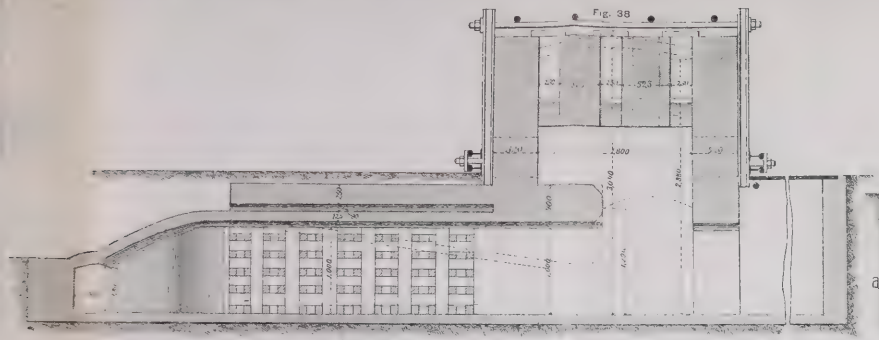
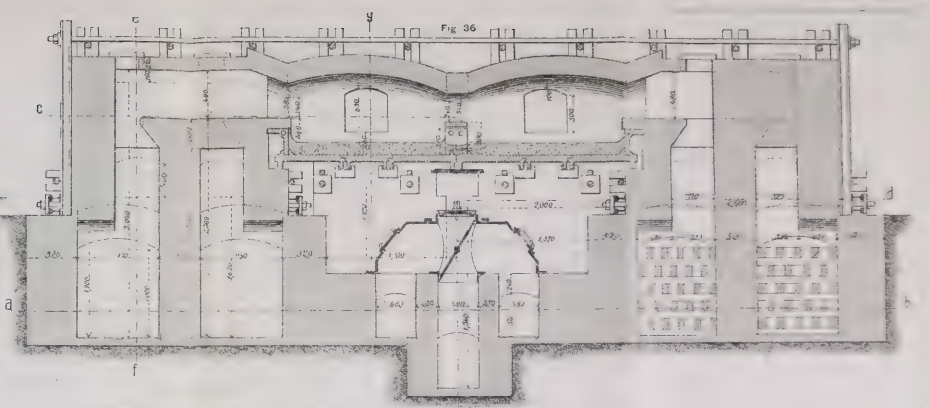
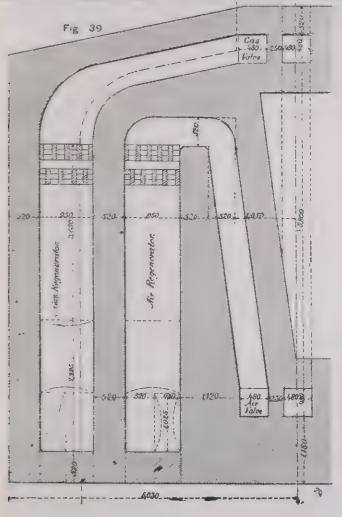
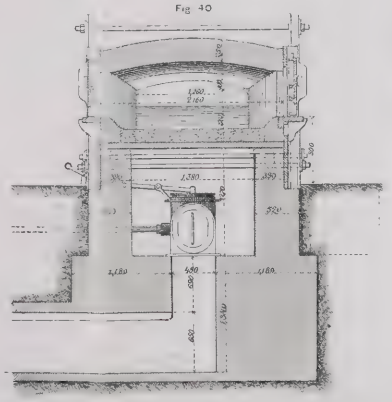


Fig. 37.
Fig. 36.
Fig. 36.
n *ab*, Fig. 36.
Fig. 36.









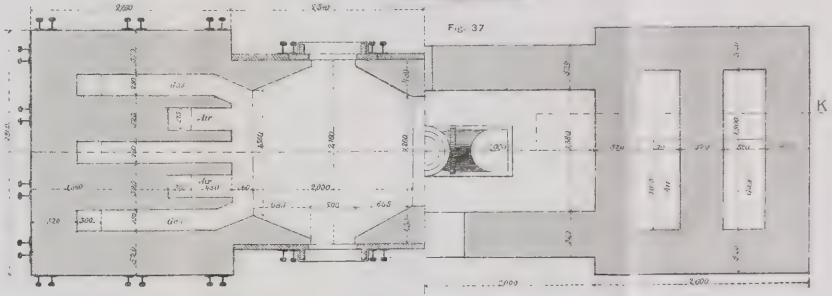
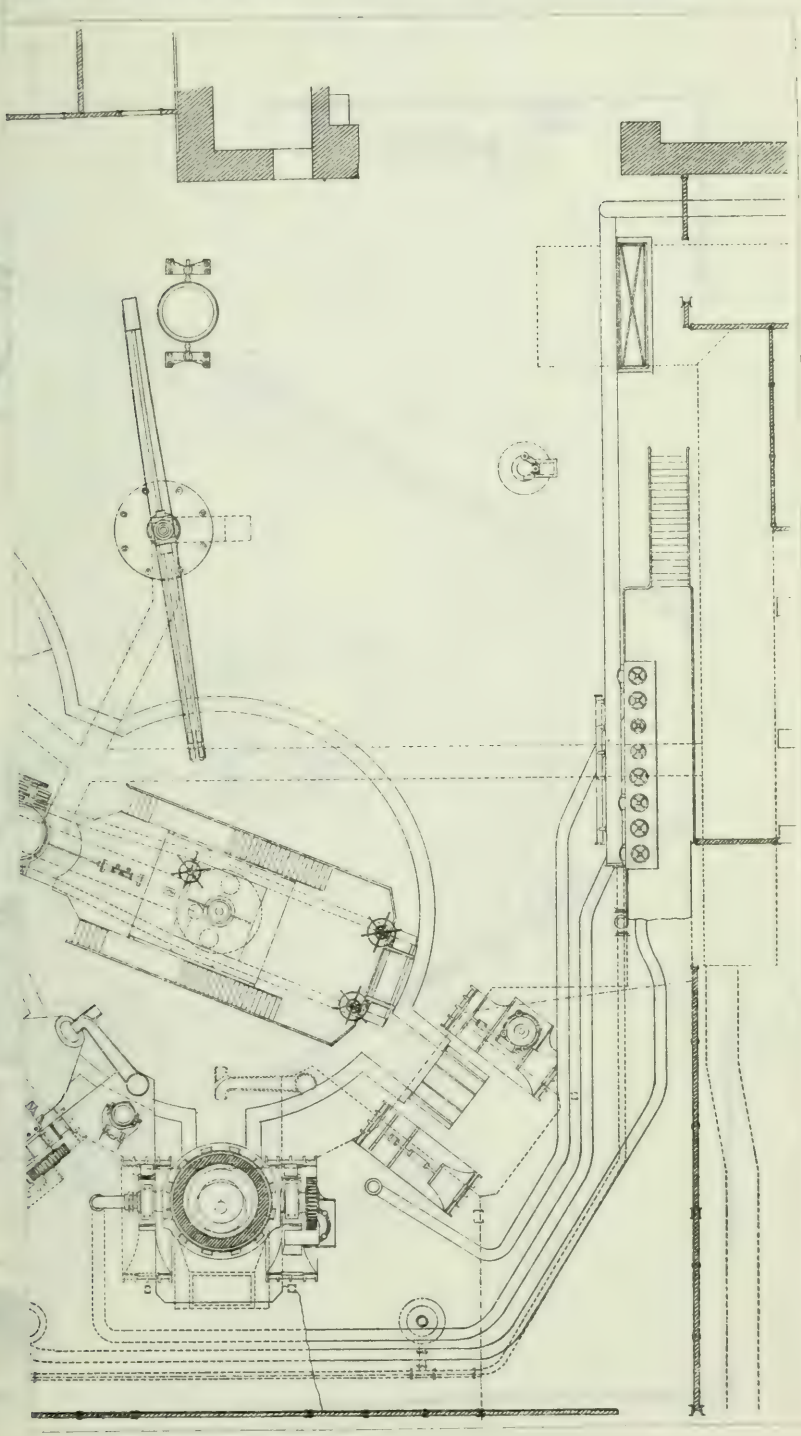
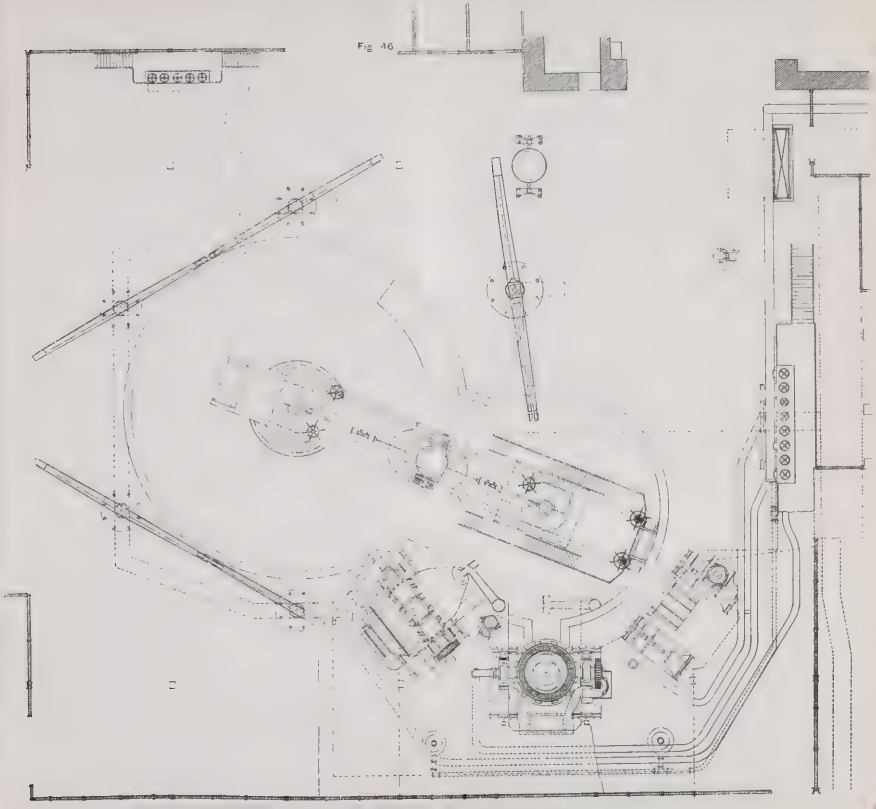


Fig. 36.—Section on *ik*, Fig. 37.
Fig. 37.—Section on *cd*, Fig. 36.
Fig. 38.—Section on *ef*, Fig. 36.
Fig. 39.—Half-Section on Fig. 36.
Fig. 40.—Section on *gh*, Fig. 36.

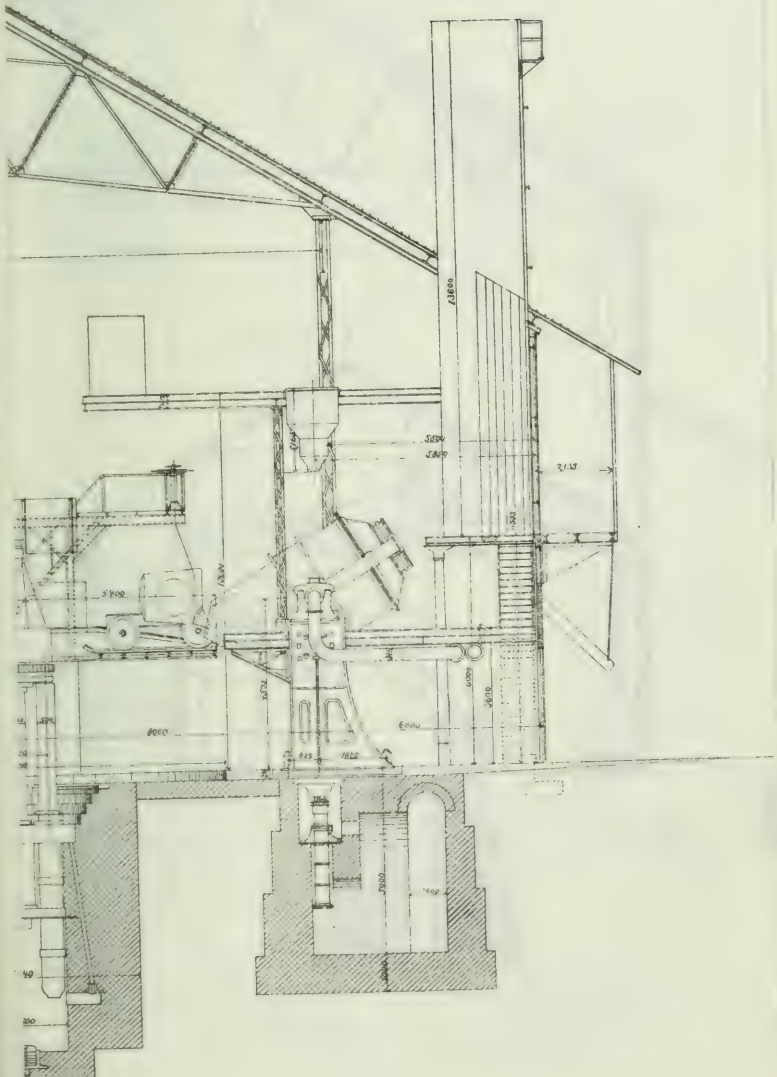
Springer Double Puddling-Furnace with Reversing Flame, at Königin-Marienhütte.



ant at Ruhrort, Rhenish Prussia.

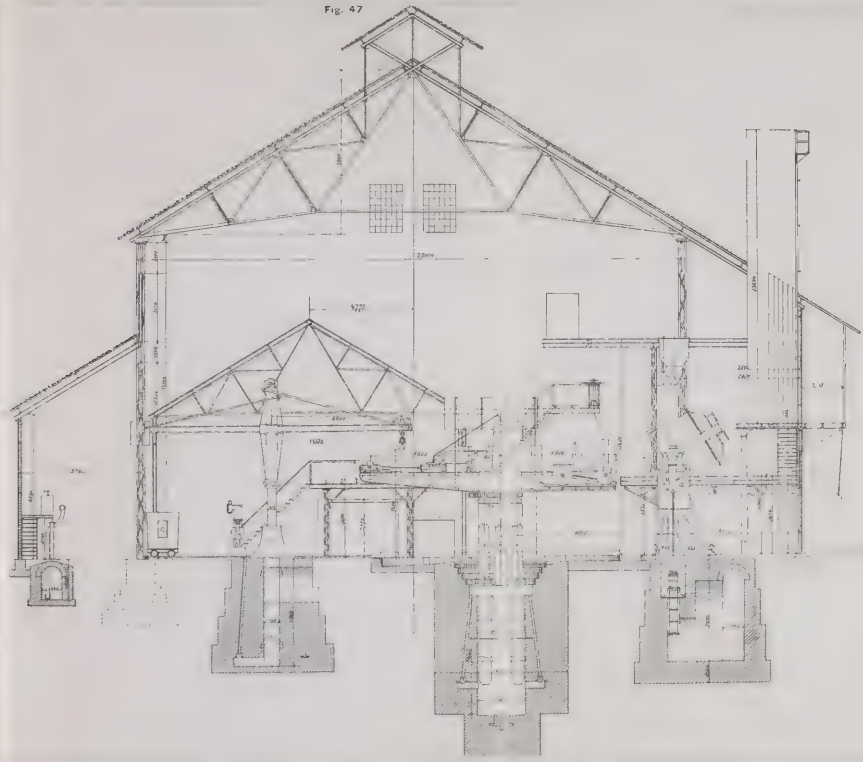


Plan of Phœnix Converter-Plant at Ruhrort, Rhenish Prussia.



ort, Rhenish Prussia.

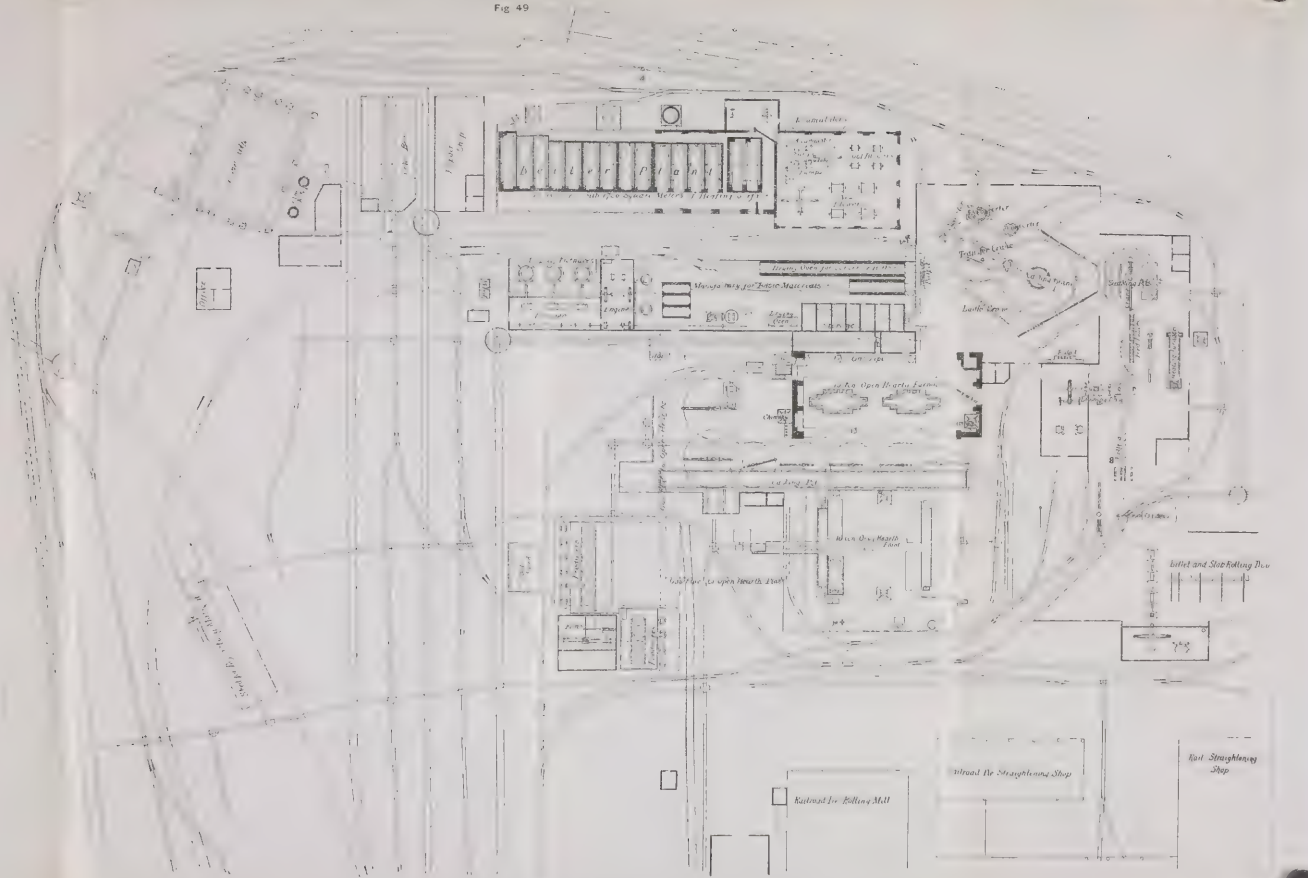
Fig. 47



Elevation of Phœnix Converter-Plant at Ruhrort, Rhenish Prussia.

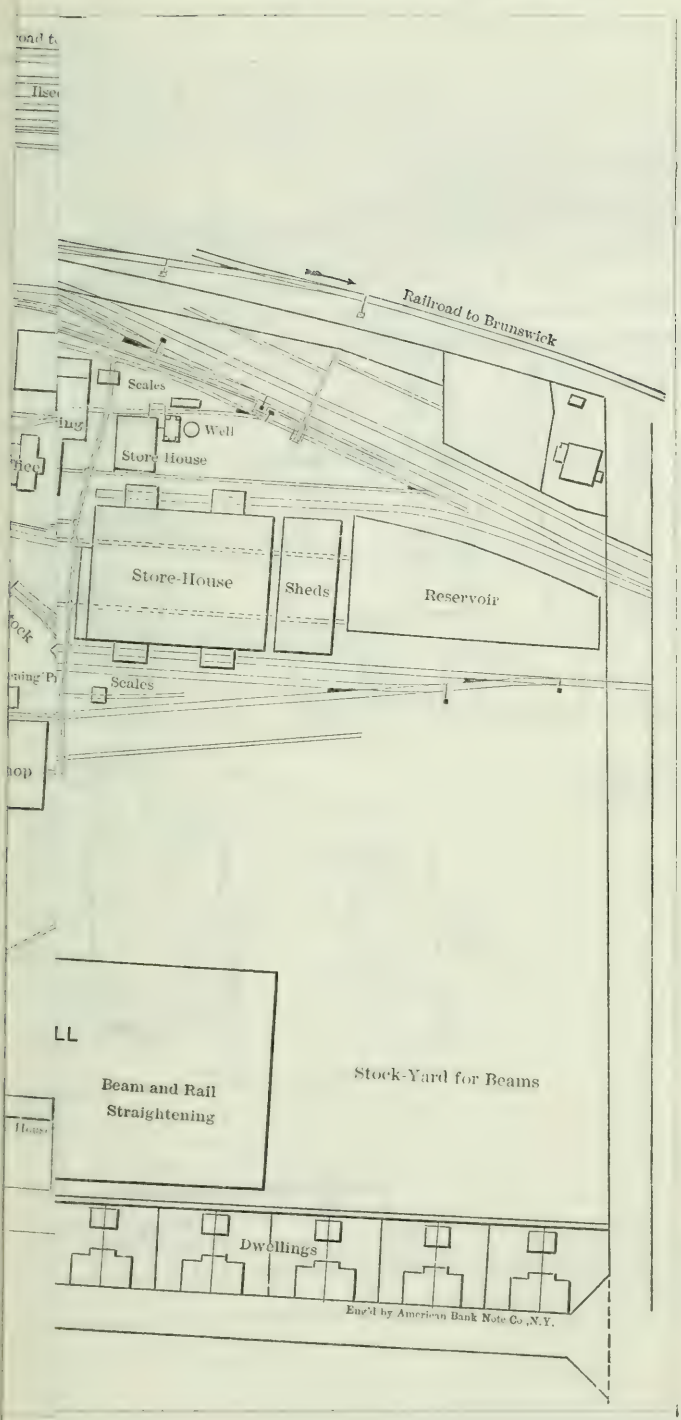
of Phœnix

Fig 49



General Plan of Phœnix Steel-Works at Ruhrort, Rhenish Prussia.

WEDDING.—Plate VII.



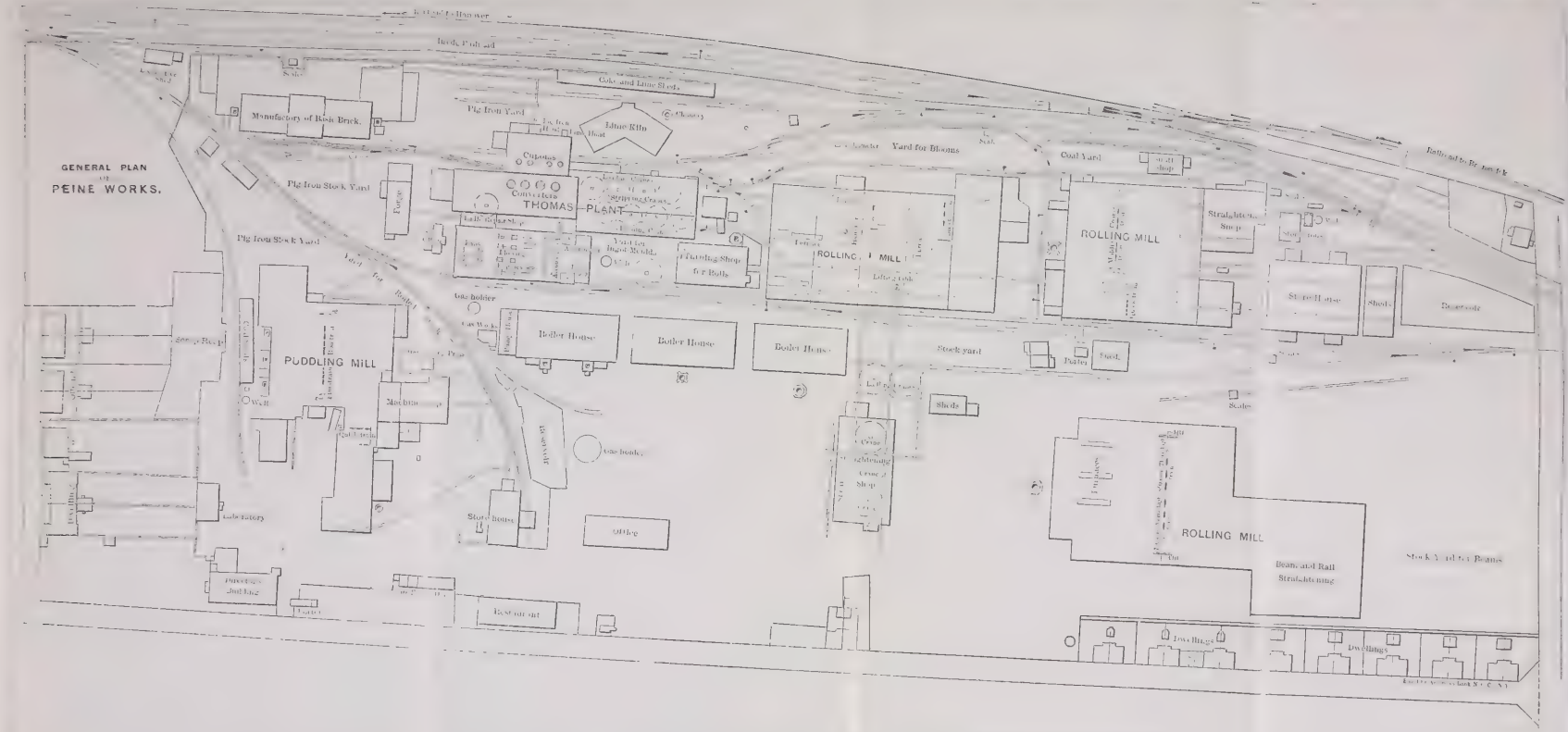


Fig. 50.—General Plan of Steel-Works at Peine, near Hanover.

dioxide, ammonia, sulphur-dioxide, chlorine, and compressed oxygen, and for all similar purposes. Hitherto, the dead weight of the iron flasks required for such purposes has been a serious drawback, since it amounts to several times the weight of the contained liquified or compressed gas—a circumstance which has been absolutely prohibitory in many cases, in which consumers would gladly have availed themselves of the great convenience offered by the commercial supply of such substances in desired quantity on demand. As the Mannesmann-tubes are very much stronger than ordinary wrought-iron or steel tubes of the same weight, they can be made correspondingly lighter; and when it is considered that they are already supplied up to 15 inches in diameter, and capable of resisting at that diameter a pressure of several hundred atmospheres, it is not necessary to dwell on the importance of this improvement.

In the second place, the Mannesmann-tubes offer the possibility of carrying out chemical reactions under high pressures at a fraction of the expense which has now to be incurred for such purposes. The manufacture of dimethyl-aniline among organic compounds, and of sulphur-trioxide among inorganic substances, are instances I could cite in this direction. Altogether I feel bound to welcome this ingenious Mannesmann tube-rolling process as a precious addition to the resources of the chemical manufacturer.

At the close of the reading and discussion of Prof. Wedding's paper a complete set of large drawings exhibited by the author in further illustration of his subject, and a large and remarkably fine exhibit of Mannesmann-tubes and tube-products, furnished by the German-Austrian Mannesmann-Tube Works of Berlin, from its mills at Remscheid, Komotau and Bous, and by the Mannesmann-Tube Co. of Landore, were donated to the museum of the Engineers' Society of Western Pennsylvania, and were accepted with suitable acknowledgments by Prof. J. W. Langley. [Fig. 73 is a view of this exhibit in part.]

THE IRON BREAKER AT DRIFTON, WITH A DESCRIPTION OF SOME OF THE MACHINERY USED FOR HANDLING AND PREPARING COAL AT THE CROSS CREEK COLLIERIES.

BY ECKLEY B. COXE, DRIFTON, PA.

(New York Meeting, September, 1890.)

THE subject of this paper will be treated as briefly as possible under the following heads:

I. The latest designs of some of the machinery used at these collieries in the preparation of coal.

II. The means employed for elevating, conveying and transporting the coal and slate in and about the breaker.

III. The means employed for raising the coal from the mines to the top of the breaker.

IV. The iron breaker as a structure.

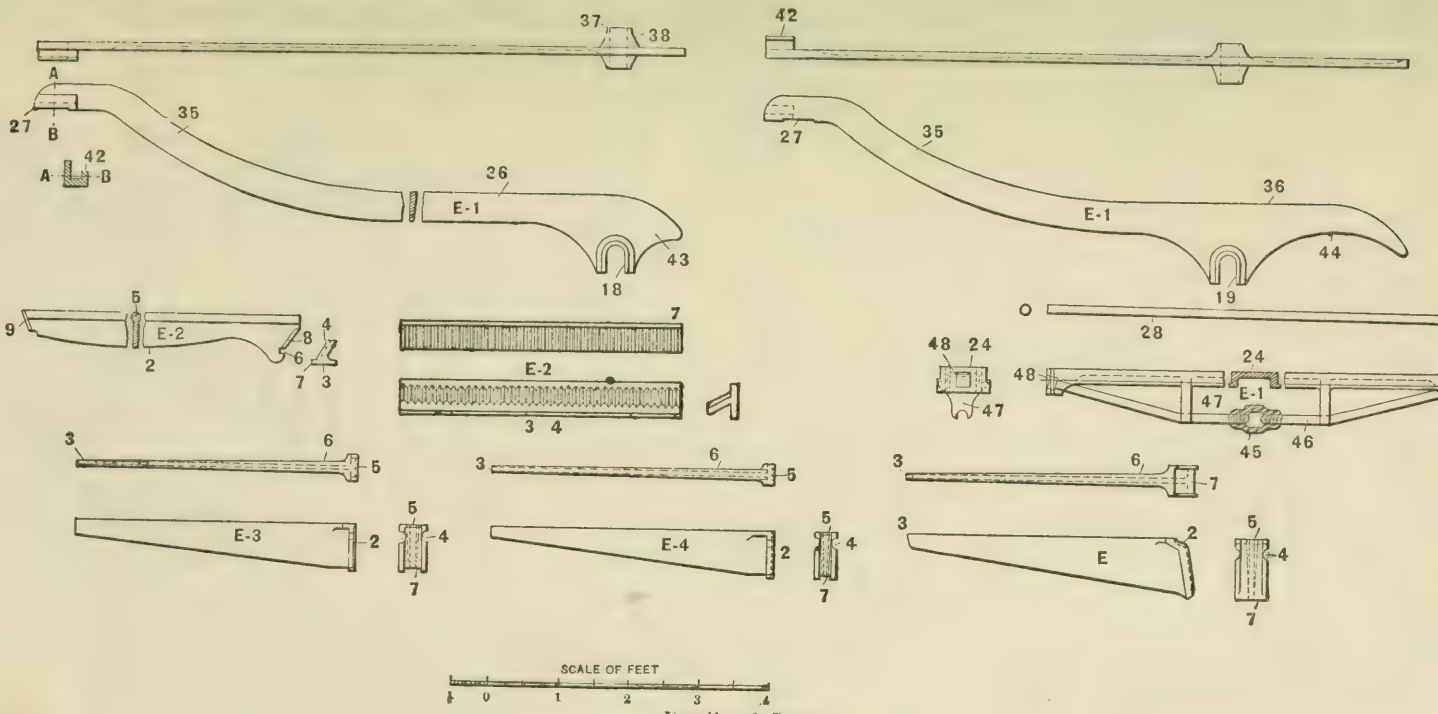
V. The preparation of the coal as effected in the iron breaker.

A key to the plates and the letters and numbers designating different parts, will be found at the end of the paper.

I. PREPARING COAL.

Anthracite coal as it comes from the mines is not marketable. The "run of mine" cannot, as in the case of bituminous coal, be sold. Anthracite, being very compact and practically free from volatile combustible matter, burns only at the surface, and it is, therefore, deemed important to have the lumps as nearly of a uniform size as possible, so that between them a large amount of surface will remain exposed to the action of the air without checking the draught too much, or allowing enough air to pass to cool the coal below the ignition-point. In other words, if the pieces of coal of the size of a chestnut and smaller are mixed with lumps of the size of an egg they fill the air-passages and prevent a free draught. It has long been recognized, therefore, that one of the most important points in preparation is to have a uniform sizing, and also to make as large a number of different sizes as can be produced without too

PLATE I.



Details of Bars.

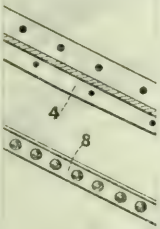
great expense. It is also essential to remove all the dust, which is of little or no use at present, and depreciates the value of coal in the market.

Mixed with the pure coal, large amounts of slate, "slate-coal" and "bony coal" generally occur. In this paper I shall use the term "slate-coal" to designate lumps composed partly of coal and partly of slate, in which the pure coal occurs in such large masses that, by re-breaking, pieces of pure coal of marketable sizes can be obtained economically; and "bony coal" to designate lumps in which the coal and slate are so interstratified that they cannot be separated economically by mechanical preparation, also coal in which the impurities are present in such high percentages as to destroy or greatly diminish its market-value. In other words, slate-coal is coal from which, by breaking and preparation, a certain amount of pure coal can be obtained; bony coal is coal which cannot be economically rendered more pure by mechanical preparation, although it may be used for certain purposes in its crude condition.

The problem is, to remove the impurities as completely as possible. Of course, when the slate occurs in separate pieces, it should be eliminated without further breaking. But the slate-coal must be broken into smaller pieces to separate the slaty portion from the coal. It is generally impossible to sell all the larger lumps which come from the mines, and machinery must be provided for breaking them up into such sizes as the market requires.

The coal coming from the mines should be divided into its various sizes, and the free slate in each size should be removed, before any breaking is done. This can be done either by hand-labor or by mechanical means. In the first case the coal is passed along chutes, on the sides of which men and boys are placed who pick out the slate, and in some cases the bony and slate-coal, and allow the pure coal to pass into the pockets. The mechanical slating of the coal depends upon one or more of three physical characteristics of the coal and slate: The difference in their specific gravity; the difference of the forms in which they break; and the difference of their angle of friction, or, in other words, the difference in the angle of a chute, lined with stone or iron, down which the coal or slate will slide without any increase of velocity. As a rule, slate will not slide down a chute which will carry coal.

Before describing the iron breaker at Drifton, I will give a brief account of some of the newer apparatus used at the Cross Creek collieries for the above purposes.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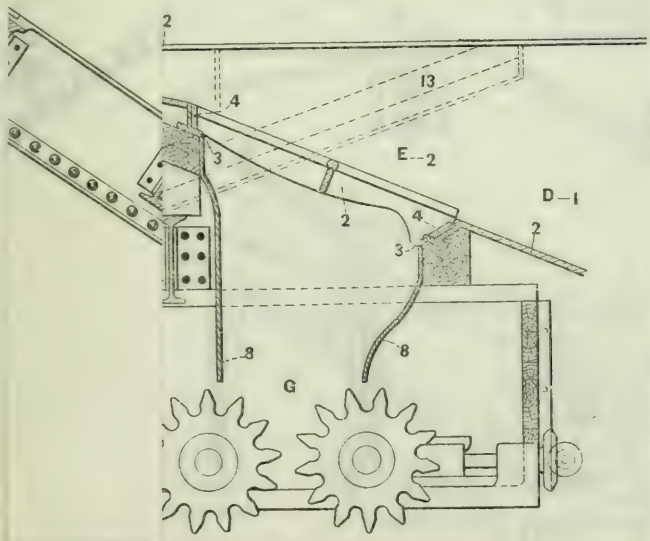
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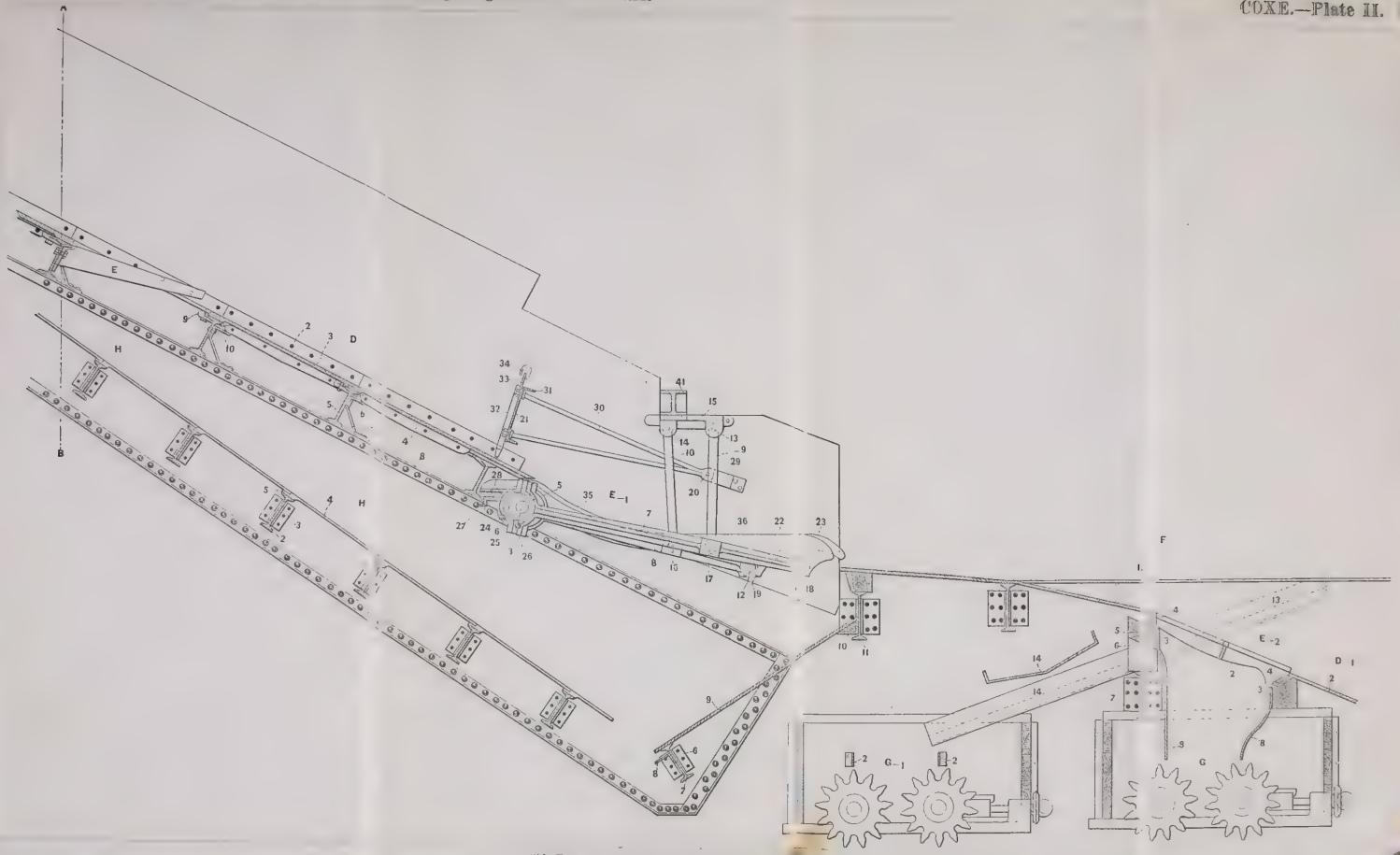
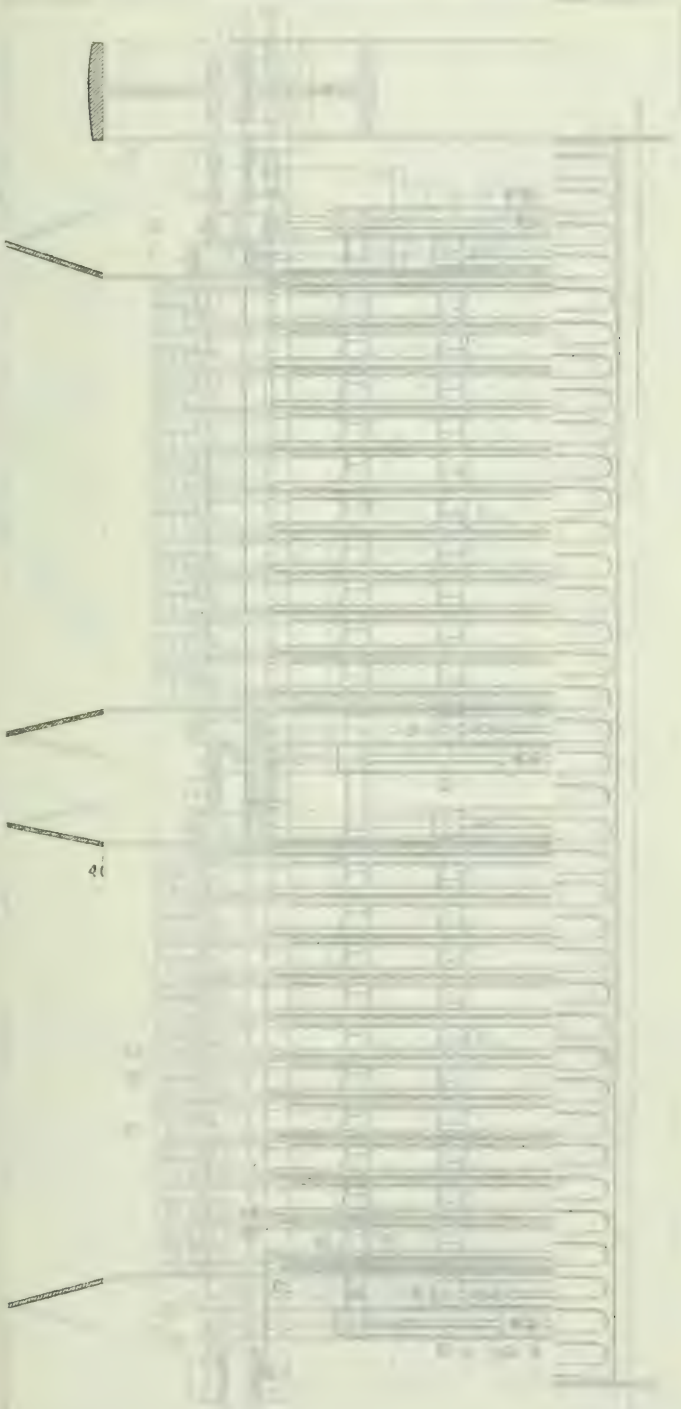
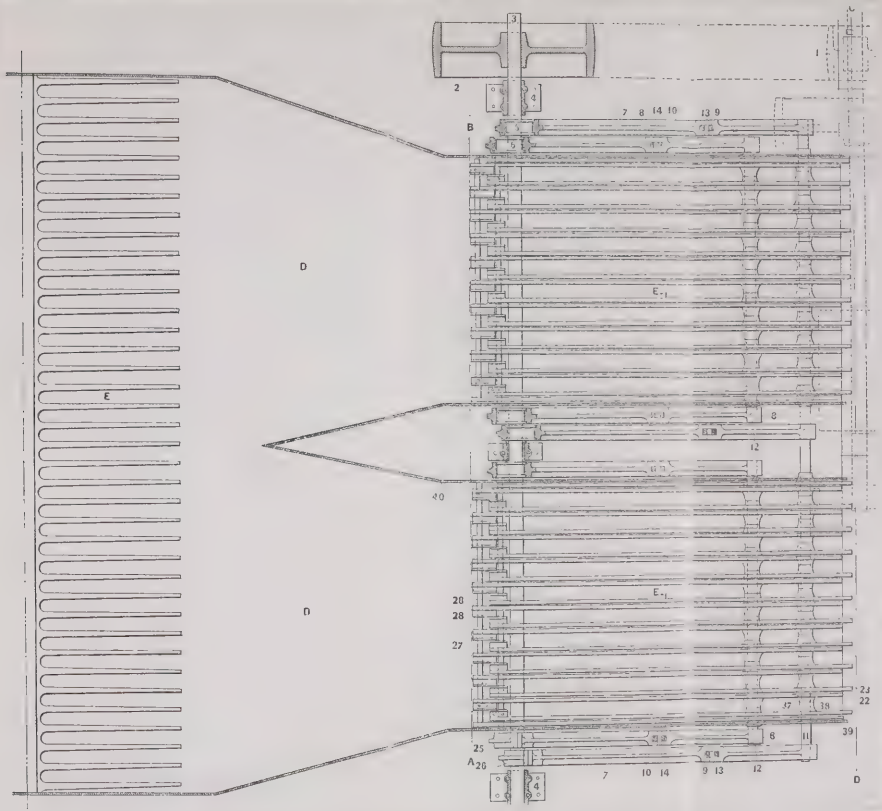


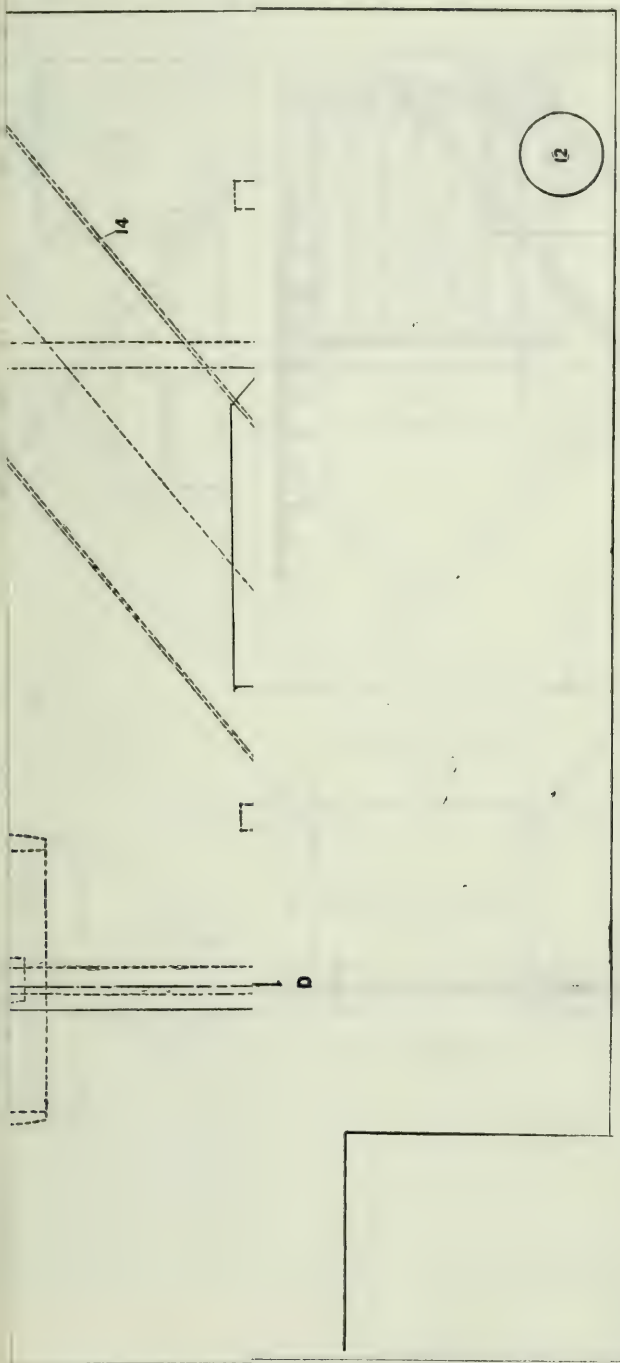
Side-Elevation of the Continuation of Dump, Bars, Platform, etc.



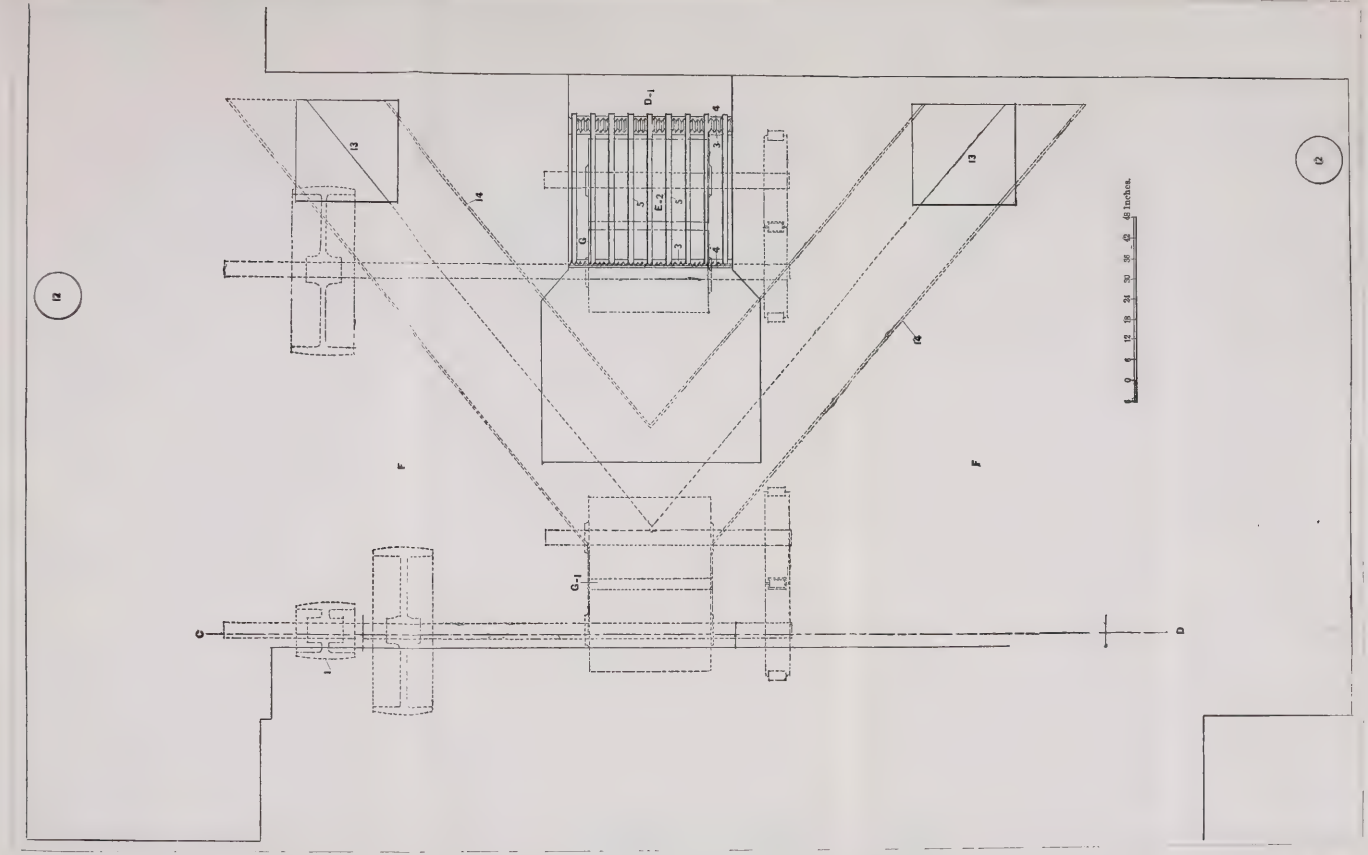
Bars, and



Plan of Dump, Finger- and Oscillating Bars, and Bars shown in Plate II. (For continuation, see Plate IV.)



, shown in PLATE II (T

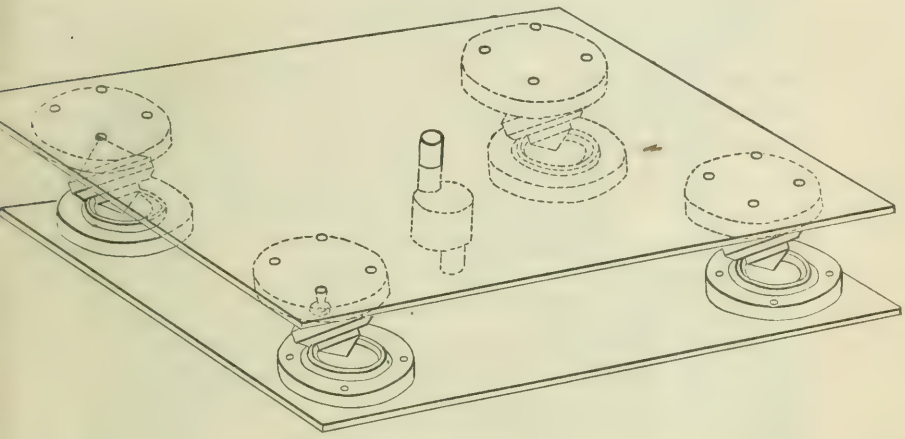


Plan of Platform, etc., shown in PLATE II (The line C D in this Plate coincides with the line C D on Plate III, of which this is a continuation).

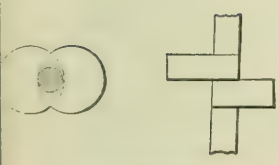
Machinery For Sizing Coal.

This may be divided into two classes: fixed or movable bars, and fixed or movable screens. In the first, the openings through which the coal falls are much longer than they are wide, while in the second the ratio of the length to the width of openings does not generally vary much from unity. In special cases the first class may be used to take out dust or fine coal; otherwise, they are seldom

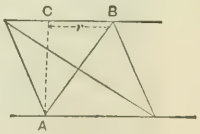
Plate V.



Gyrating Mechanism.



Eccentric Shaft.



Gyrating Cone.

Diagrams of Mechanism for Gyrating Screens.

employed, except for large coal, unless when exact sizing is not important. The reason is, that long flat pieces fall out with the cubical pieces of much smaller dimensions, rendering the coal thus sized unsightly, inconvenient to handle in the furnace, etc.

There are three types of the first class used in the iron breaker:

1. The adjustable bars, supported at both ends.
2. The finger-bars, supported at one end.
3. The oscillating bars.

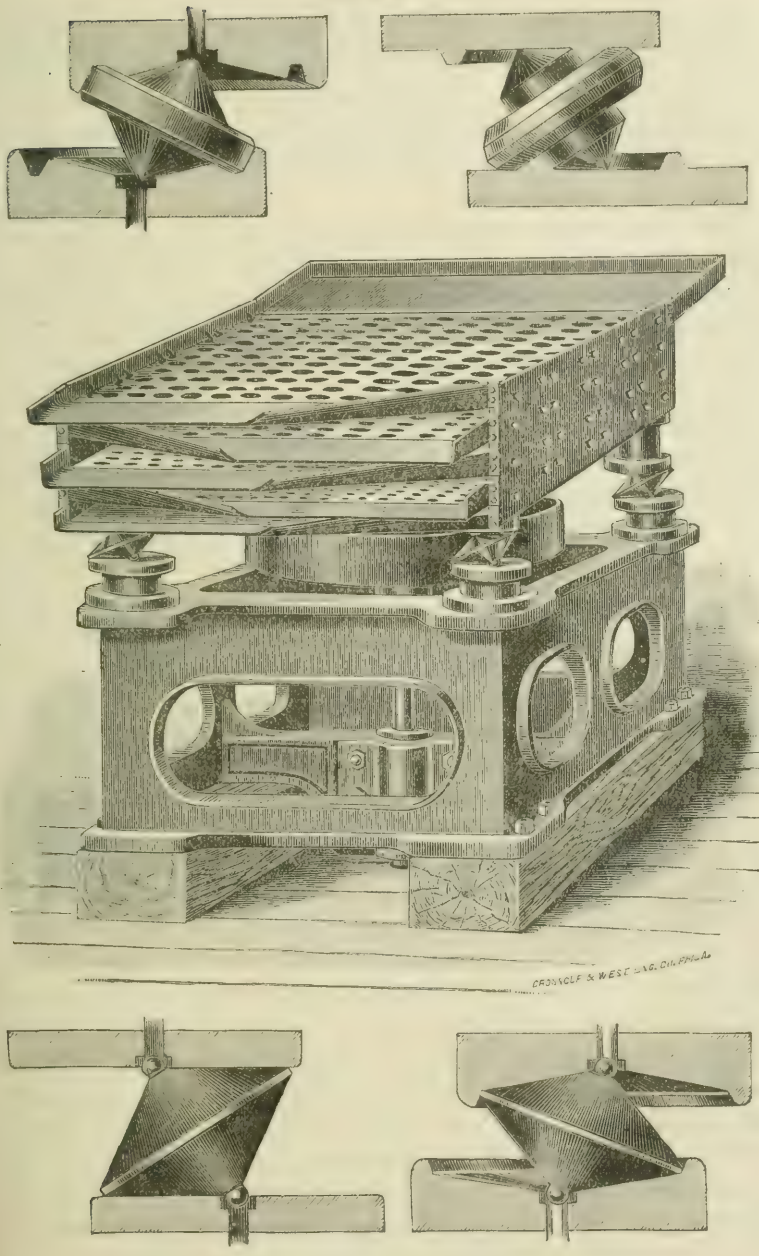
The Adjustable Bars.—E_2.—These bars have been used for many years. Their construction is shown in Plates I., II. and IV. The

top (5) of the bar is cylindrical and projects beyond the web, which supports it, so that any lump which passes through the upper part will fall freely without jamming. The two ends (8) and (9) of the bar are V-shaped, and fit into similarly shaped grooves (4), so that the bars can be set at distances from each other varying with the sum of the width of the bases of the triangles, the usual opening being 4 inches. They are generally four feet long, but can be of any size. E_2, Plate I., left-hand drawing, shows the construction of these bars, and the right-hand drawing that of the bar-bearers.

The Finger-bars.—E, E_3 and E_4.—These have been recently introduced. In using the old-fashioned continuous bars part of the dirt and fine coal is often carried over the bar, and is delivered in the chute at the lower end instead of falling through; and as the spaces between the bars are parallel and closed at the lower end, long pieces often wedge and catch, particularly at the bottom, thus necessitating a frequent cleaning. Of the finger-bars, E, Plate I., the lower end (3) is entirely free, and the bars are narrower there than at the upper end (6); and any lump that may wedge is likely to be loosened by the first lump which strikes it. Three forms of these are shown in E, E_3 and E_4. They are all of the same length, but the distances between the bars differ. Upon the vertical portion at the upper end are two half-holes (4), by which they are bolted to the beam- or bar-bearings. In order to permit the varying of the opening as may be necessary, the bars are cast simply tapering from the top down, or with a greater or less projection near the top, like the old form. The holes are cast or drilled in the bearer for the greatest opening with the ordinary bar; if it is desirable to reduce the opening, the bar with the projection on the head is used. By changing the angle between the top of the bar and the back, a greater or less angle of inclination can be given to the bar, without changing the position of the bar-bearer.

The Movable or Oscillating Bars.—These bars have been introduced within a few years at the Cross Creek collieries for several reasons. They diminish the height of the breaker, because they are shorter and much flatter than the fixed bar of the same cleaning-capacity. They act as a regulator or feeder for the breaker, the amount of coal passing over them per minute being constant, if the supply is sufficient and the number of revolutions remains the same; while, by regulating the speed of the driving-shaft, the quantity can be varied at will, within certain limits. They allow the men upon the platform to get much nearer their work without danger. (With the ordinary fixed bars, the pitch must be sufficient to allow the coal to slide

PLATE VI.



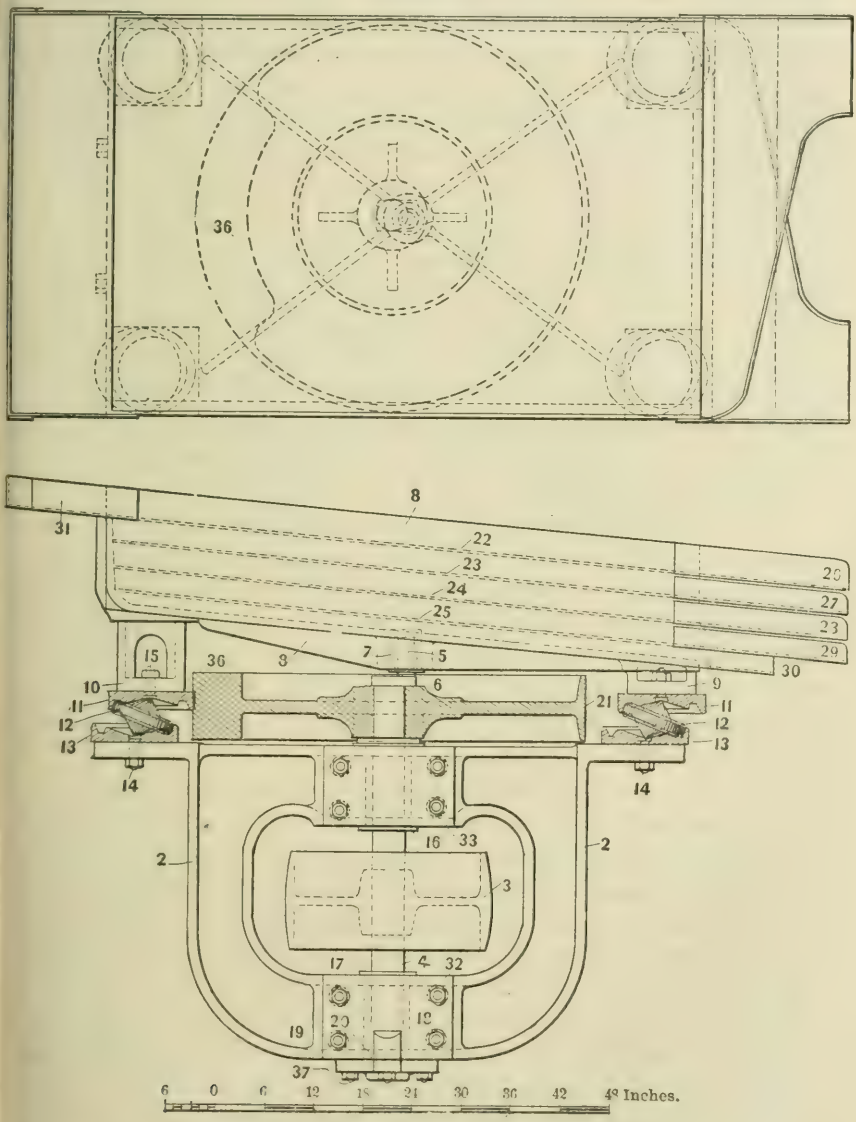
Single Gyrating Screen.

down freely. This it often does, with great velocity, so that the men must remain on the side. When the oscillating bars are used, the coal can be fed upon their upper end, from which it is gently carried to the platform. This allows the men to stand safely in front of the bars instead of on the sides.) They bring the coal cleaner and with less small stuff to the platform, thus permitting a better separation of coal, slate, etc.

The machine, as constructed, is a modification of the Briart movable bars, with some important changes. It consists essentially of a series of double bars, placed sufficiently far apart to allow coal of the required size to pass between the bars of each pair. In Plate I., the upper left-hand drawing shows one half and the upper right-hand drawing the other half of the double bar. When they are set together, the two vertical sides (42) touch, and the two taper sides are turned outwards. The lower ends of the bars carry the bearings (18) and (19), which fit over the shafts (11) and (12), Plate II., while the upper ends (27) are supported upon two round steel rollers (28), which roll upon the plate (24). The details of the bar-roller and its bearer are shown at (24), (28) and (45) to (48). Plates II. and III. show the general arrangement of the bars. On the main or driving-shaft (3) there are two eccentrics (25) and (26), placed 180 degrees apart. At each end each pair moves two eccentric-rods (7) and (8), which are suspended by the pins (16) and (17) to the suspension-bars (9) and (10). The throw of the eccentrics is three inches. When the driving shaft revolves, the points (16) and (17) describe an arc of a circle which is practically a straight horizontal line three inches long, while the center of the eccentric-strap describes a circle three inches in diameter. The centers of the shafts (11) and (12), being half as far from (16) and (17) as the latter are from the centers of the eccentric straps, describe approximately an ellipse, the major axis of which is three inches long and the minor axis one and one-half inches, *i.e.*, the centers of (11) and (12) rise and fall three-quarters of an inch in moving forward three inches, and fall and rise three-quarters of an inch in moving backward; but (12) is always above (11) when it moves forward, and below when it moves backward, and *vice versa*. The bars are flat on top, the extreme lower end (23) being rounded off to allow the coal to roll off easily; then for a certain distance to the left they are horizontal; then they rise in a curve (35), the center of which is upward, to the point where the coal arrives upon the bars. The upper ends of the bars, which are carried by the rollers, are under the coal chute, *D*,

and protected. If the shaft (3) revolves in the opposite direction to that of the hands of a watch, the shaft (12) will be above its medial

PLATE VII.



Single Gyrating Screen—Side Elevation and Plan.

position when moving forward, while (11) will be below its medial position when moving backwards. In consequence, all the half-bars

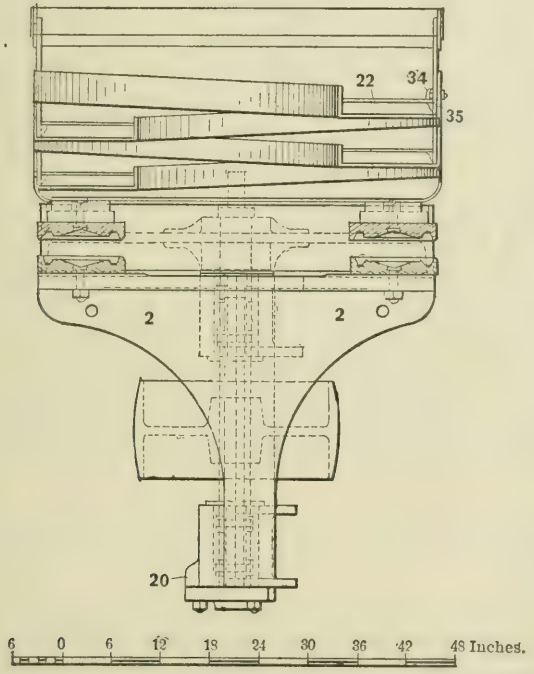
carried by (12) will be above those carried by (11) when moving forward, and below when moving back; while at the beginning and end of each forward motion they will be on a level. The result is, that a lump large enough to straddle two half-bars of the same system will be raised by one set of half-bars, moved forward three inches and deposited on the other set at the end of the stroke, and will then be raised, moved forward and deposited on the first. In this way the coal will be moved forward six inches for each revolution, until it tumbles off at the end. It must be observed, however, that although the upper part (27) does not rise when the half-bar moves forward, the part (35) is carried upwards a little, advances beyond its adjoining half, which is moving backward, and thus pushes the coal forward. All the working-parts, except the portion of the bars used for screening and the supporting shafts (11) and (12) are either covered, as is (28), or are on the outsides of the iron sides (39) of the coal-chute. The bars are immediately over the mud-screen pocket, *H*, into which all the coal falls freely. Shafts (11) and (12) must be far enough apart to allow the large lumps passing through the bars to fall below them.

The essential difference between this construction and that of the Briart bar is, that in the latter the supporting shafts are driven directly by eccentrics, which involves the necessity of having the shafts which carry the bars, rise and fall, the same distance as they move forward, while in this construction they only move up and down half as much as they move forward. It was found that, with the Briart construction, the coal was thrown up and down too much when it was fed forward with any rapidity.

Fixed Screens.—In the sizing-apparatus of the second class, however, as has been already observed, the ratio of the length to the width of the openings in the screen approaches unity. The screen may be either fixed or movable. The former consists simply of an inclined plane, formed either of woven wire screens, or punched or cast plates with round, square, elliptical etc., holes. The coal in this case is allowed to slide or roll by gravity, not too rapidly, down this plane. The larger pieces pass over and the smaller fall through. By placing several screens with openings of decreasing size underneath one another, or a series with openings of increasing size, in the same chute below one another, any desired number of sizes can be made. The objection to these is that their capacity is limited, the sizing is imperfect, and the screens clog more or less. In the iron breaker they are used for the following reason in certain cases.

While the coal is handled or moved in the breaker, a certain proportion of small pieces always breaks off, so that when the coal is loaded from the pockets into the cars it is necessary to take out this finer coal. This is done by allowing the coal to pass over a fixed screen just before entering the railroad car. A number of devices have been used for this purpose; but the most satisfactory at present is the one suggested to us by Mr. Irving A. Stearns, of Wilkes-Barre, which is simply a steel plate punched with holes twice as long as they are wide, and the longer axis of each hole lying in

PLATE VIII.



Single Gyrating Screen—Front Elevation.

the direction in which the coal slides into the car. Although these holes are now generally made with one punch, their size and shape can be understood best by reference to the manner in which they were originally made. Two cylindrical holes of exactly the same diameter were punched so that the circles were tangent to each other, and the two centers of the circles in a line parallel to the axis of the chute. Then two pieces were punched out so as to leave two sides continuous and parallel to the line joining the two centers; the hole

thus formed consisting of a square of the diameter of the circular holes, terminated at the upper and lower ends by a semi-circular space. These punched plates are placed just below the gates to the pockets, in the chute leading to the car. What falls through is taken by a conveyor to the main screening-conveyor, which loads it into a mine-car, to be hoisted to the top of the breaker, as will be explained below.

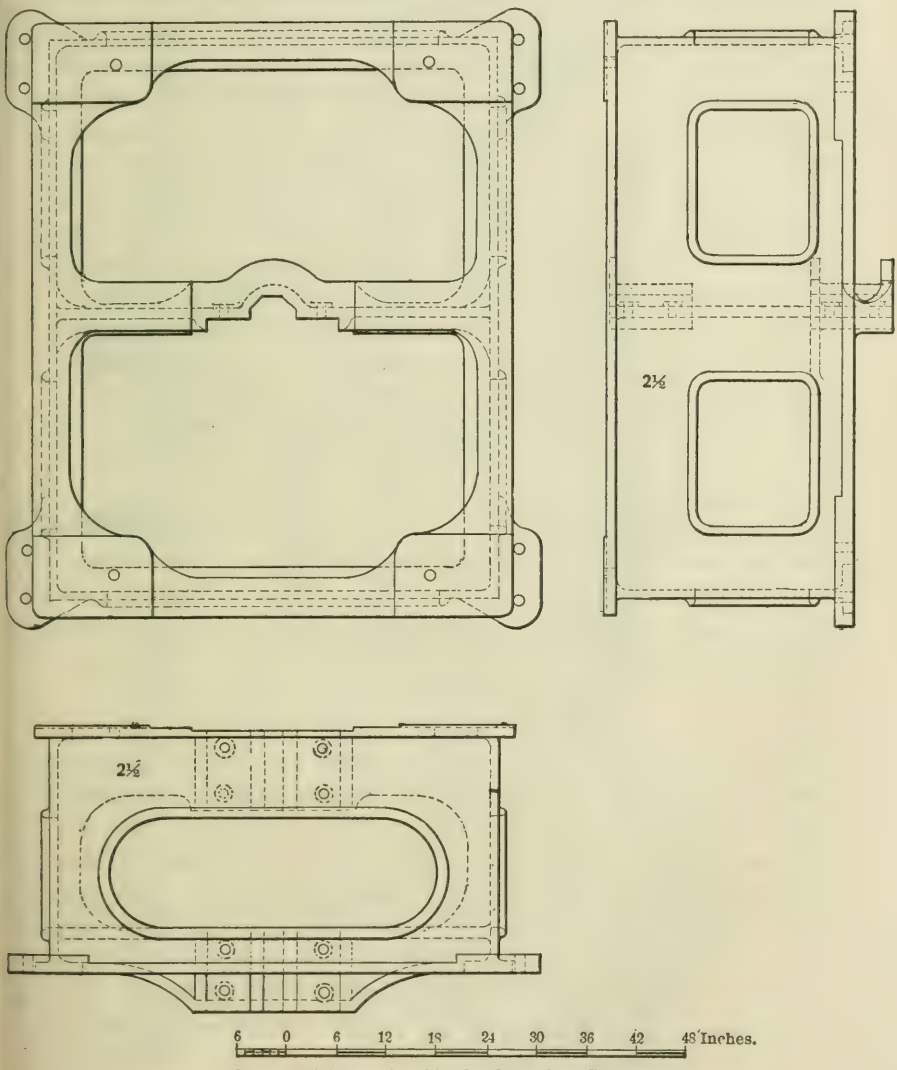
Movable Screens.—The movable screens are among the most important parts of a breaker. They are of two types. In the first type the screening-surface forms a cylinder and revolves about its axis. This form of screen has been often described, and is used in almost all the anthracite collieries; I need barely refer to it here. In the other type the screening-surface is approximately horizontal, and the motion and action is very similar to that of an ordinary hand-sieve. In many cases the screen is moved backward and forward in an approximately horizontal plane. This motion, combined with the inclination of the sieve, causes the coal which is fed on the higher part of the screen to travel gradually across it, allowing the smaller particles to fall through. This form has in many cases been adopted for small coals, and has long been used in metal mining. In other cases the approximately horizontal screen receives a gyratory motion, like the motion a moulder gives to his sieve when screening his sand. This is the type of screen used in the iron breaker. Its great advantage is that the whole surface of the screen is constantly in action, while, in the revolving screen of say 5 feet in diameter, only about 8 inches of the 16 feet circumference is at any one time in action, unless the screen is overcrowded, and the revolving of the screen acts like an elevator and tends to throw the coal back into the screen.

The problem of constructing a gyrating screen, when the screen is to be large and must make a great number of sizes, is to support it in such a manner that it will gyrate easily and safely, and, at the same time, that it will be self-contained, so that the centrifugal force will be counterbalanced and will not shake the building. This has been done successfully.

The method consists essentially in supporting one horizontal plate upon another by means of three or more double cones, while the motion of gyration is given to the upper plate by a crank upon a shaft passing through and journalled in the lower plate, as shown in Plate V. The cones roll freely in a prescribed path on the lower plate, while the upper plate moves upon the other end of the double

cone, its relative motion to that of the cone being the same as that of the bottom plate. The result is that every point on the upper plate describes a circle of the same diameter (in coal-screens generally about 4 inches), but no two circles have the same center.

PLATE IX.



Box Bed-Plate for Single Gyrating Screen.

The cones may be guided in various ways. In Plate VI., the upper right-hand drawing represents on a larger scale the cone shown in Plate V. Upon the upper and lower plate there is an annular,

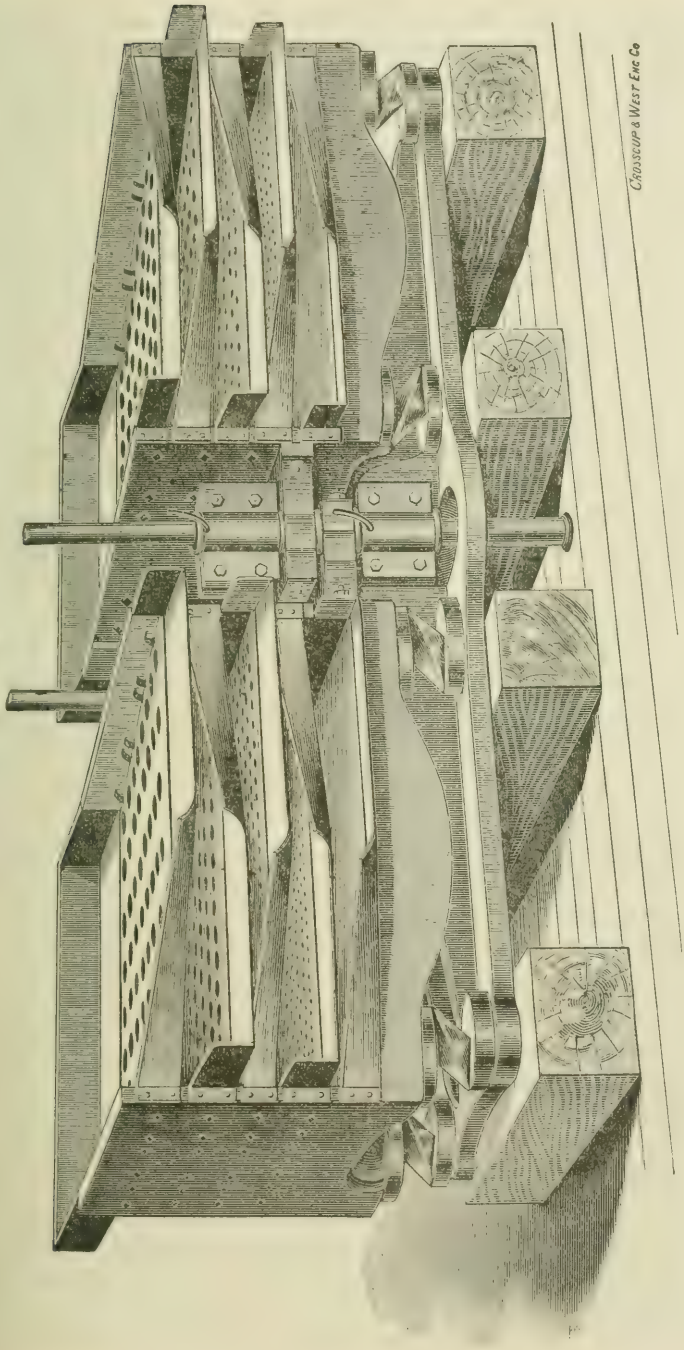
truncated V-shaped track which fits into a corresponding groove in the cone. These cones and tracks are cast in chills, and there is no work of any consequence required to finish them.

In the upper left-hand drawing of Plate VI., the guiding is done by an annular groove in the running-plate and a corresponding annular enlargement of the cone at the outer edge. When, however, the screens are run at high speed, there is a tendency in the double cone to fly from the center; the surface, therefore, on which the cones roll, is sometimes made conical, so that the weight of the screen has a tendency to force the cone towards the center, thus counteracting the centrifugal force to a great extent. In this type the circumferential surface of the enlargement is very broad and has a good bearing against the outer surface of the groove in the running-plates. This form of cone is well suited to resist any tendency of the centrifugal force to throw it out. The two lower drawings, Plate VI., represent other types of cones in which the guiding is done by a ball-and-socket joint at the two points of the cones. Both the running-plates and cones in this type are made in the lathe, and are all fitted to gauge. The same precautions are taken in the lower right cut as in the upper left cut, to counteract the effect of the centrifugal force.

All these forms are working well in practice. The radius of the circle of gyration (which is the same as the length of the driving-crank) is determined by the length of the axis of the double cone and the angle formed by its axis with a vertical line when the machine is running. Thus (see Plate V.), $r = \mathrm{BA} \sin \mathrm{BAC}$. With coal, r is generally 2 inches.

Plates VI., VII., VIII. and IX. show the single gyrating screen, the first type of this screen constructed and used. A large number of them are now running at several of the Cross Creek collieries. The lower plate of the gyrating mechanism, Plate V., has become a box bed-plate (Plate IX.) or a U bed-plate (Plates VII. and VIII.), and the upper plate a screen-box. On the shaft shown in the bed-plate a pulley is placed to drive the screen. The large wheel shown between the screen-box and the bed-plate is counter-weighted to balance the centrifugal force of the screen-box. Above this wheel is a crank to drive the screen-box. The box is 4 feet wide and 6 feet long, inside measurement, giving 24 square feet of screen-surface per shelf. The number of shelves in this type varies from 2 to 6, depending upon the size of material to be screened. The smaller the size of coal, the closer to each other the shelves can be put. The boxes are made from 1 foot

PLATE X.



Front View of Double Gyrating Screen as Originally Built.

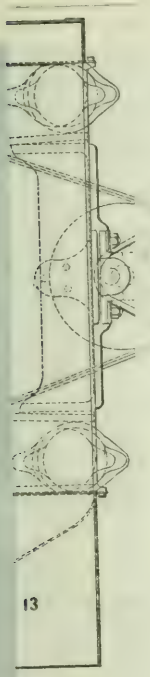
to 2 feet deep. Plate X. shows the double gyrating screen as originally constructed. The counterweight was done away with, the one screen balancing the other, and the whole machine being self-contained. The two screens were driven by two parallel vertical shafts, each shaft having two eccentrics upon it close together, and placed exactly 180° apart, as shown in Plate V.

In this case the tendency of the centrifugal force of the right screen is to pull the two shafts exactly in the opposite direction to that in which they are pulled by the left-hand screen. Each screen, being driven by means of two eccentric boxes placed about 4 feet apart, must describe the prescribed path; and there is little or no work done by the cones in guiding the box, which is not the case when the screen is driven by a single crank.

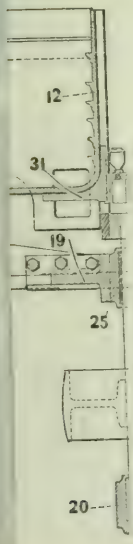
The double screen constructed in this manner* did not give satisfaction, as it was found that when the number of gyrations per minute exceeded 110 to 120 (depending on the weight of the screen-box) a very serious shaking set in. This was due to the fact that, when the two screen-boxes were farthest from the line joining the center of the shafts, they moved in opposite directions, producing a conflict tending to turn the whole machine around a vertical line through its center of gravity, causing great strains upon the eccentric-boxes and the bolts attaching the eccentrics to them. It was also found that the best results in screening were obtained at from 140 to 145 gyrations per minute. Therefore, a comparatively light counterbalance was put on a shaft connected to the outside of each box, which removed the difficulty and allowed the screens to be run smoothly at the required speed. It was also demonstrated that it was necessary for the counterbalance to be driven by belting at the proper speed, instead of being simply carried along by the screen-box. The reason of this is that, as there is always a little play between the bearing on the box and the crank-pin of the counterweight, a jar is occasioned at each change, which does not occur when the counterweight is driven by a belt.

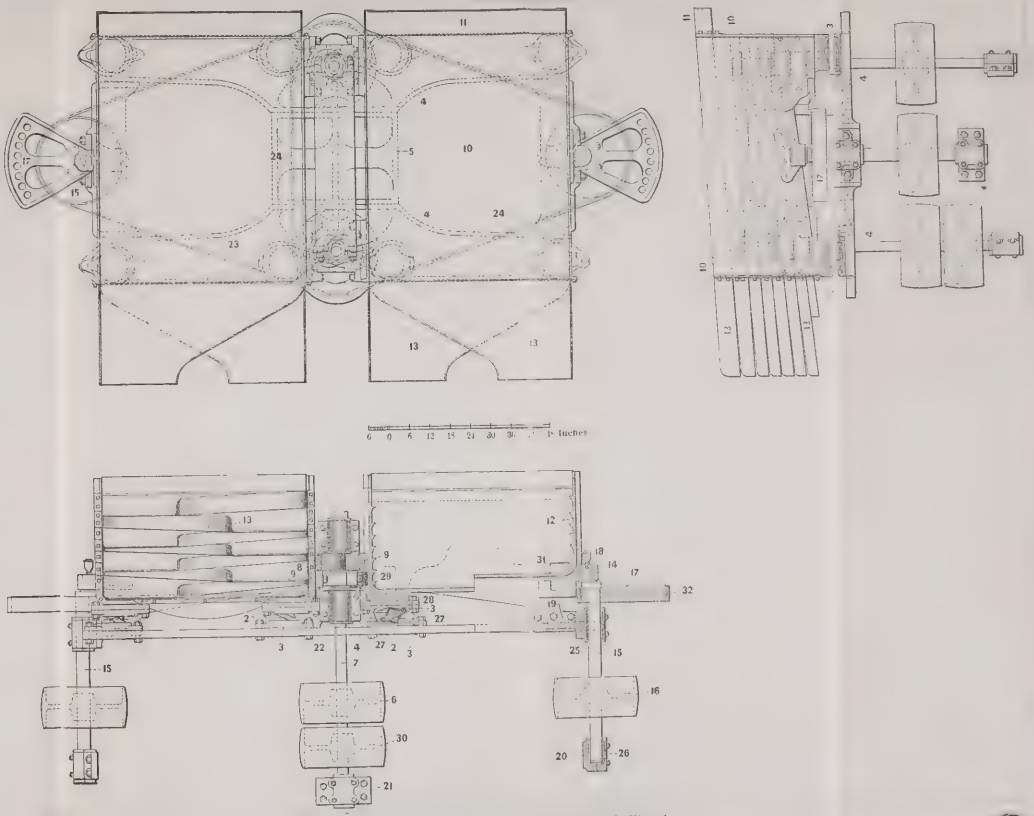
Plate XI. shows the screen as now constructed; (30) being the driving-pulley, (7) the eccentric-shafts, (16) the pulleys on counterweight-shafts, (6) those on the eccentric-shafts. In order to drive all the four shafts exactly in unison two belts are used, one of which passes round the two pulleys (6), and the right-hand pulley (16). The left-hand pulley (16) is covered with one thickness of

* A small portion of this description has already been given in London *Engineering.*

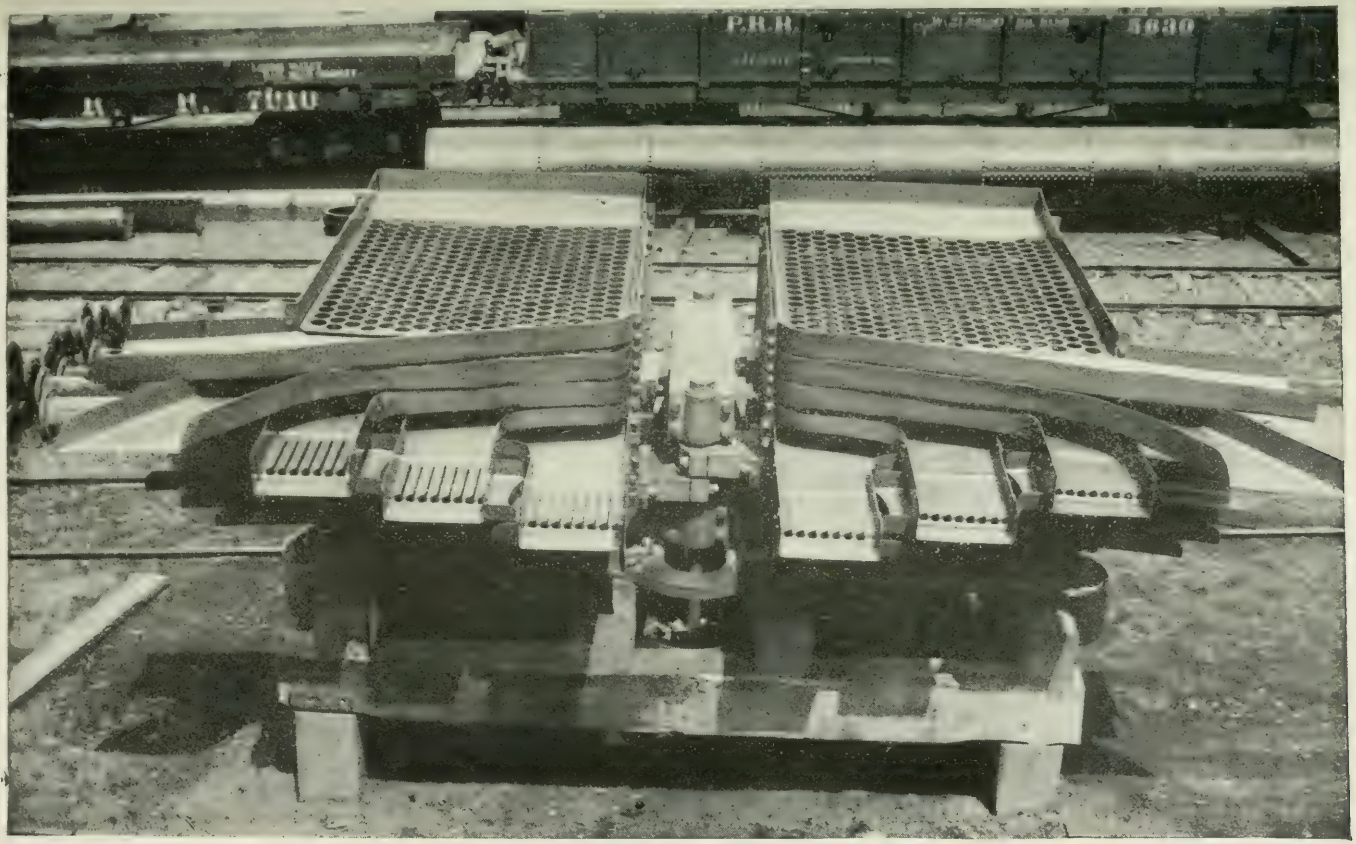


42 48 Inches





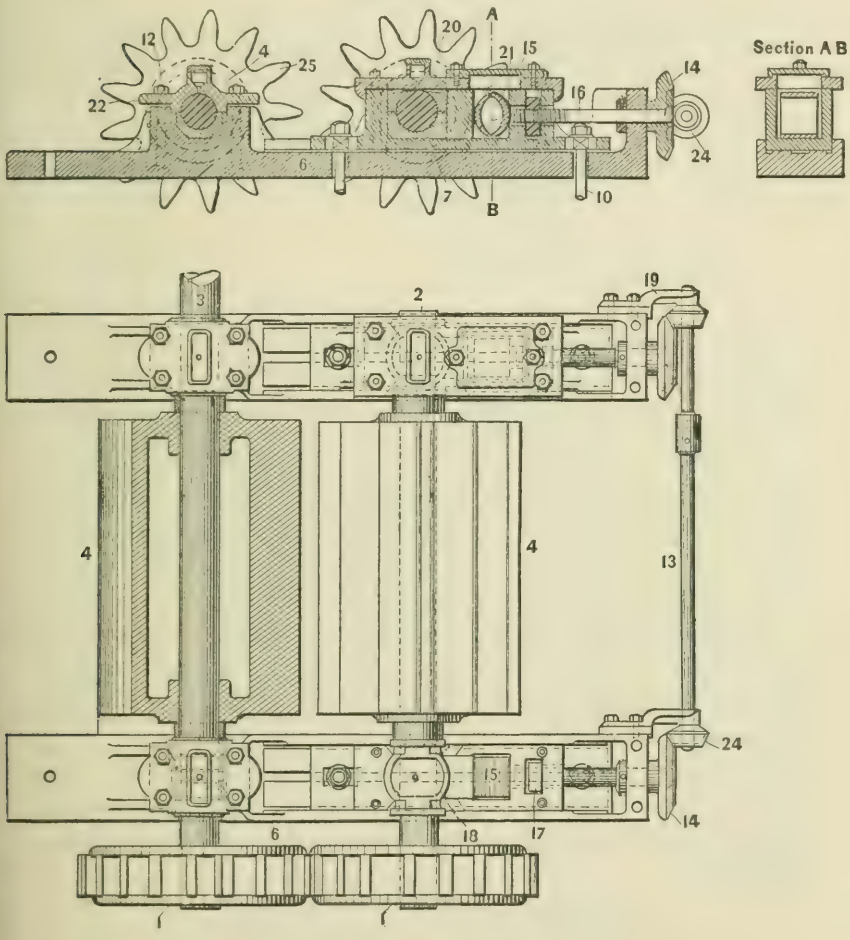
Double Gyrating Screen: Plan, Front and Side Elevation.



Double Gyrating Screen.

belting (23), which passes around all four pulleys. The first belt passes around three pulleys, which are of the same size, but under the second belt.

PLATE XIII.



Adjustable Corrugated Rolls.

Since the adoption of these improvements, the screen runs very smoothly, and there has been no difficulty in building the boxes high enough to make nine sizes of coal in one box.

Plate XII. is from a photograph of the screen, recently constructed,

which makes nine sizes, and has three gyrating slate-pickers, namely, for stove, small stove and chestnut. When the screen runs at a proper speed, the inclination required to pass the coal over the surface of the screen is very slight, namely:

For uppermost or steamboat screen,						$\frac{1}{8}$	inches to the foot.	
" next lower or broken	"					$\frac{1}{8}$	"	"
" " or egg	"					$\frac{1}{8}$	"	"
" " or stove	"					$\frac{1}{8}$	"	"
" " or chestnut	"					$\frac{5}{16}$	"	"
" " or pea	"					$\frac{7}{16}$	"	"
" " or No. 1 buckwheat,						$\frac{9}{16}$	"	"
" " or No. 2	"					$\frac{3}{4}$	"	"
" " or No. 3	"					1	"	"
Bottom of box, dirt,						$1\frac{1}{2}$	"	"

When first constructed, much larger inclinations were given to the screening-plates. The boxes in this case are 5 feet wide, 6 feet long, and from 2 feet to 4 feet high (inside measurements.) From 4 to 8 shelves are put in, making from 5 to 9 sizes. The screen shown in Plates X. and XII. weighs, with bed-plate, 10 tons. The screening-surfaces have always circular holes, varying from $5\frac{1}{4}$ inches to $\frac{1}{16}$ inch in diameter. Cast-iron is sometimes used when the holes are large, but punched steel is now preferred, being much lighter. Copper is used for small sizes when the water is very acid.

About twenty different sizes of holes have been used with coal to suit different circumstances. The sheet-iron lips at the lower end of the boxes are constructed to suit each particular case, so as to discharge the coal on either side of box or at any required point in front. There is no clogging of the holes, as would at first seem likely. The circular form of the holes, and the tendency of the pieces of coal to move in a small circle, causes the holes to clear themselves without difficulty.

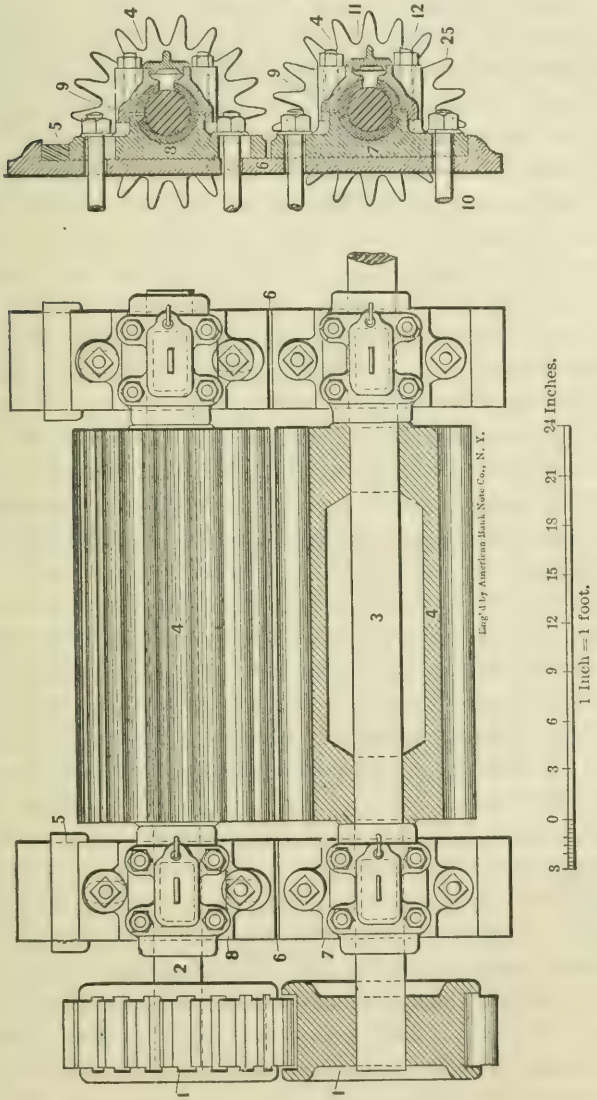
Machinery For Breaking Coal.

For breaking up the coal two methods are used. When the lumps are large and the pieces of slate attached to them are of such a character as to render it economical, the larger lumps are broken by hand, the men using picks made for that purpose. In this way large pieces of pure coal or pure slate can often be obtained; but by far the larger portion of the breaking is done by rolls, the usual construction of which is well known.

I shall, therefore, only describe the rolls used at the Cross Creek

collieries, which differ in one point from those generally adopted. The difference is in the form of the teeth. The rolls used are known as corrugated rolls, and the teeth are continuous from one end to the

PLATE XIV.



Fixed-Distance Corrugated Rolls.

other. There are no points. The end of the tooth is slightly rounded and the part doing the work is cast in chills, so as to give

greater endurance. The principle upon which these rolls act may be explained as follows:

In the operation of a roll as ordinarily constructed, *i.e.*, with pointed teeth, the point of one of the teeth inserts itself into a lump of coal which is passing through the rolls, and breaks it very much as the stroke of a pick would do; that is, the lines of fracture radiate approximately from the point where the tooth strikes the lump of coal. If two pieces of round iron are placed parallel to one another, and at such a distance apart that a piece of coal will just be supported by them, and if a third piece of round iron, placed midway between and in a direction parallel to and above the other two, is then brought down upon the coal, the piece of coal will break near the middle like a piece of wood subjected to a load in the middle too great for it to bear. The result of this action is generally to break the lump into two pieces of nearly the same size. This is the result sought to be attained with the corrugated rolls, and it is for this reason that the plan referred to below, of breaking from one size as far as possible into the next size below, has been adopted.

We use one size of rolls for breaking lump into steamer, another for breaking steamer into broken, another for breaking broken into egg, another for breaking egg into stove, a fifth for breaking stove into chestnut, and a sixth for breaking chestnut into pea-coal. We also make special sizes for special purposes. Experiment has taught us that, although all sizes below the size which is being broken are always made, yet the most economical method is to break any size as nearly as possible into the size immediately below it; in other words, that it is more economical to break your lump as far as possible into steamer, then break your steamer as far as possible into broken, your broken into egg, and so on; of course at each time eliminating all the coal below the size that you wish to break, before passing that size through the rolls. If a piece of any size is simply broken as nearly as possible in two, for the next size, the amount of small coal made is very much less than if the same piece were struck near the center with a pick and broken into a number of fragments. Our old practice, which has not entirely disappeared, was to arrange the rolls in such a manner that by putting them further apart or closer together, we could increase or decrease the quantity of the larger sizes of coal. But where arrangements are made to break the sizes successively, it is not necessary to change the distance between the centers of the shafts of the rolls after the proper distance for most economical breaking has once been determined. Plate XIII.

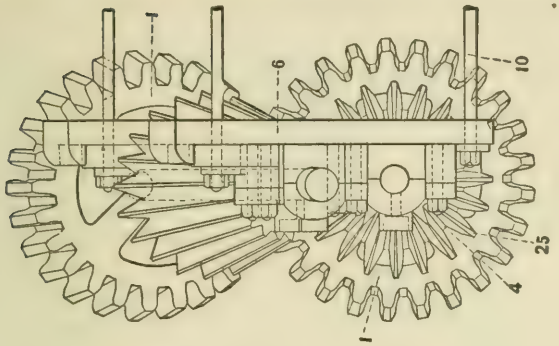
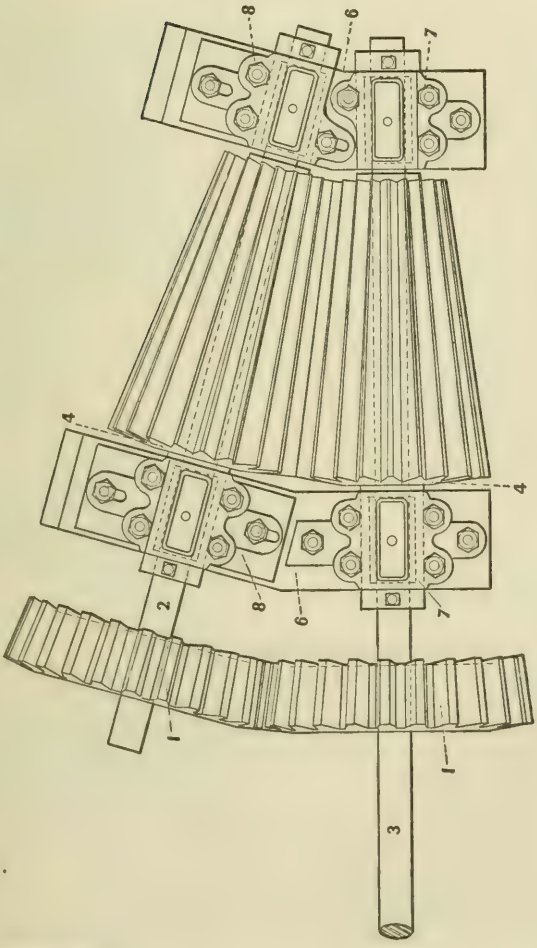
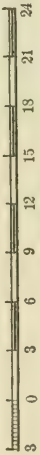


PLATE XV.





Taper Corrugated Rolls.

is a drawing of our large rolls, with the bed-plates, which we have used for our lump and steamer. In these, as will be observed, there is an arrangement for changing the distance between the rolls while they are running (when this change is great, it is necessary to use gear-wheels of greater or smaller diameter), and also a breaking-piece (15) which would be crushed in case a piece of rock or iron should go through the rolls. The bending of the shaft is thus avoided. This disposition of the bed-plates is well known, and has been used for many years. It was originally designed by Mr. David Clark, of Hazleton, and is described by Mr. H. M. Chance in volume A C, *Second Geological Survey of Pennsylvania*. Plate XIV. shows the style of bed-plate used for all rolls smaller than our steamboat-rolls. In this case the pedestals are fixed, and when it has once been determined what is the proper distance between the shafts, no arrangement is made to change their relative position, except by changing the keys which regulate the distance between the pedestals. Plate XV. shows a set of taper-rolls, which are used where a small quantity of a number of different sizes is to be broken up at once. At the upper or larger end the rolls will take steamboat; a little further from the end they will take broken; a little further, they will take egg; and a little further, stove. When the coal to be broken up is of different sizes, and the quantity not large, these rolls may be economical; but the tendency of our practice is to increase the number of rolls, having a different roll for each size to be broken. Plate XVI. shows the cross-section of the teeth and a portion of the body of the roll. The largest is the lump-roll, the second is the steamboat, the third the broken, the fourth the egg, the fifth the stove, and the sixth the chestnut. Of course it is not usual to break up either stove or chestnut-coal except when it is very flat, and therefore not marketable, or slaty or bony. In this case it is broken up into smaller sizes and prepared by jigging, etc.

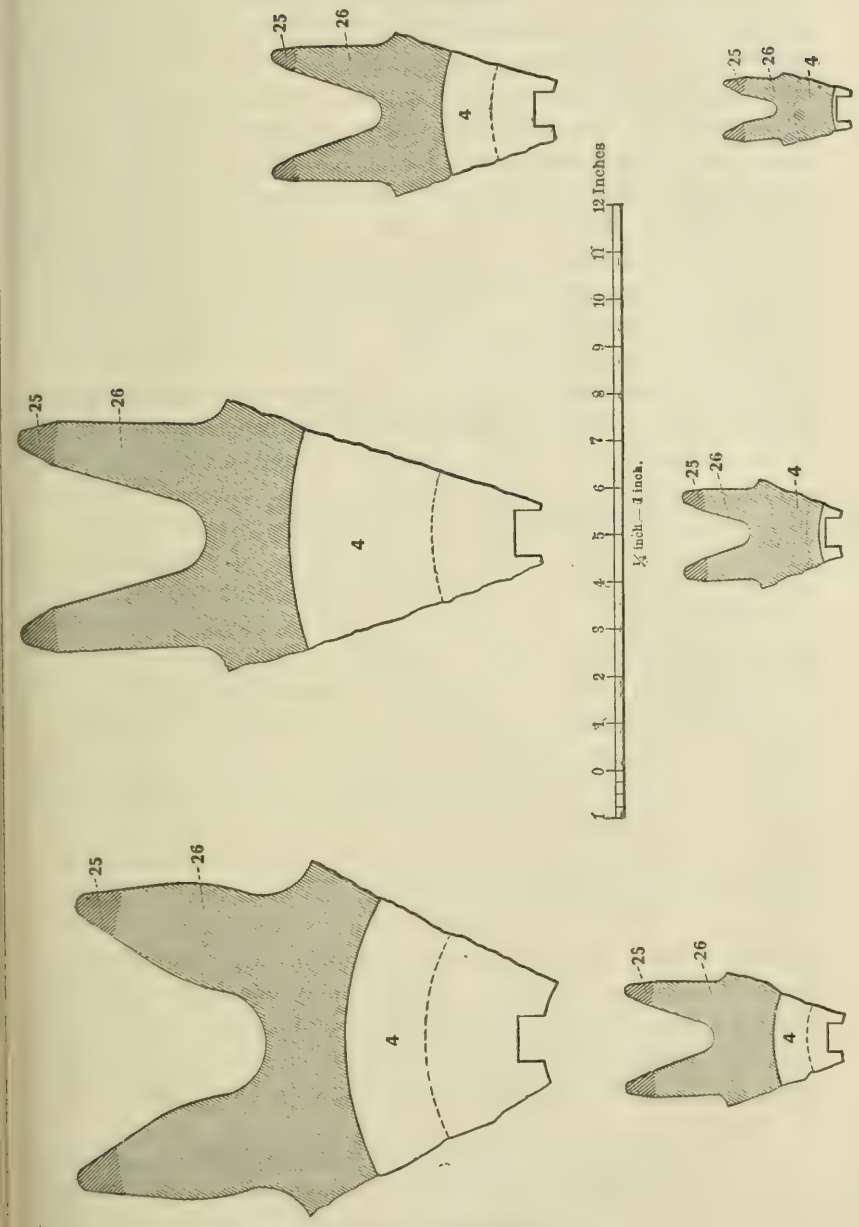
The Slate-Picking Chutes.

Originally, the coal coming from the screens passed down a simple chute, or trough, and the men or boys were placed either above or alongside of it, picking out the slate as the coal passed by. There are three objections to this:

1. When a large quantity of coal is passing, the men can really only pick out the slate on top, much slate being hidden.

2. One and the same piece of coal, having a slaty appearance may be picked up by each slate-picker, in succession, and returned to the chute.

PLATE XVI.



Sections of Roll-Teeth.

3. It is very difficult to judge of the amount of work done by each slate-picker.

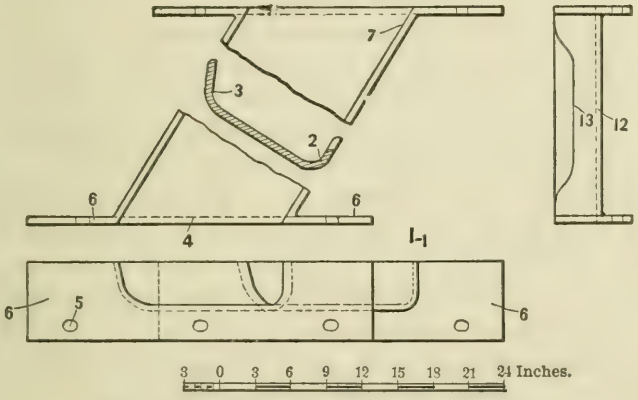
For these reasons, a different type of picking-chute has been adopted, as shown in Plates XVII., XVIII. and XIX. It consists essentially, 1st, of a supply-chute (8), Plates XVIII. and XIX., which receives the slaty coal; 2d, of the intermediate chutes, I and I_1, Plates XVII. and XVIII., where the picking is done; and 3d, of the delivery-chutes (10), Plates XVIII. and XIX., which carry off the coal picked over. The coal from the screen or jig slides, as shown by the arrows, down the supply-chute (8), on each side of which the intermediates, I and I_1, are placed, as close to each other as possible, there being room (14) enough between each two picking-chutes for a man or boy. At the other end of the intermediate is the delivery-chute (10). The supply- and delivery-chutes have the same inclination, but the former is a little the higher, so as to give a slight inclination to the intermediate, the axis of which is placed at an angle of about 8 to 10 degrees with the horizontal, and 25 to 28 degrees with the supply-chute. The slate-picker, who sits with his face towards the upper end of the chute, causes a thin stream of coal to pass in front of him, cleaning it thoroughly as it passes. The same coal is handled by one man only, with this exception, that one or two men are placed at the end of the delivery-chute to inspect the coal, and take out any pieces of slate which may have escaped the regular pickers. Immediately over the supply-chute, and supported on iron rods, is the half-round slate-chute, into which the pickers throw their slate, slate-coal, etc. This continues to the bottom, where it is examined, and the slate-coal is picked out and taken to the rolls to be broken up and prepared. This style of picking-chute is suited for broken, egg, stove, and chestnut-coal; for broken coal, I, Plate XVIII., and for egg and stove-coal, I_1, Plate XVII., are used. For steamboat coal an analogous form is employed, the supply-chute being larger, and the picking-chute longer, steeper, and narrower, only one lump at a time coming down the chute. The steamboat-chutes are generally single, that is, the supply-chute has intermediates on one side only. The other chutes are also sometimes made single for local reasons.

The Jigs Used for the Larger Coals.

These jigs, of which there are twelve in the breaker, are a modification of the Luhrig jig. They may be divided into two parts, the upper being the machinery for moving the piston, and the lower the jig proper. In constructing them a tank is built long enough for the number of jigs desired. This is made water-tight on the

outside, and divided by partitions which need only be approximately water-tight, as there is water on both sides of them. The jigs are built in pairs, being about 5 feet wide and 6 feet long, so as to give a piston 5 feet by 3 feet. Pulley (43), Plate XX., is a heavy fly-wheel pulley, and is driven by a pulley on the line-shaft of the jigs. The jig-pulleys are all alike, and the pulley on the line-shaft is arranged so as to give the proper speed to the plunger (40). Through the pulley (43) passes the counter- or driving-shaft (49), at each end of which there is a crank-disk (45), the crank-pin (48) being so arranged as to slide from the center outwards. By thus sliding the crank-pin (48), the stroke can be increased from nothing to 6 inches. Around the crank-pin there is a square block (47), which slides in the slotted arm (44). As the crank-disk revolves it gives the slotted arm (44), which is on the

PLATE XVII.



Egg- and Stove-Coal Intermediate Picking-Chutes.

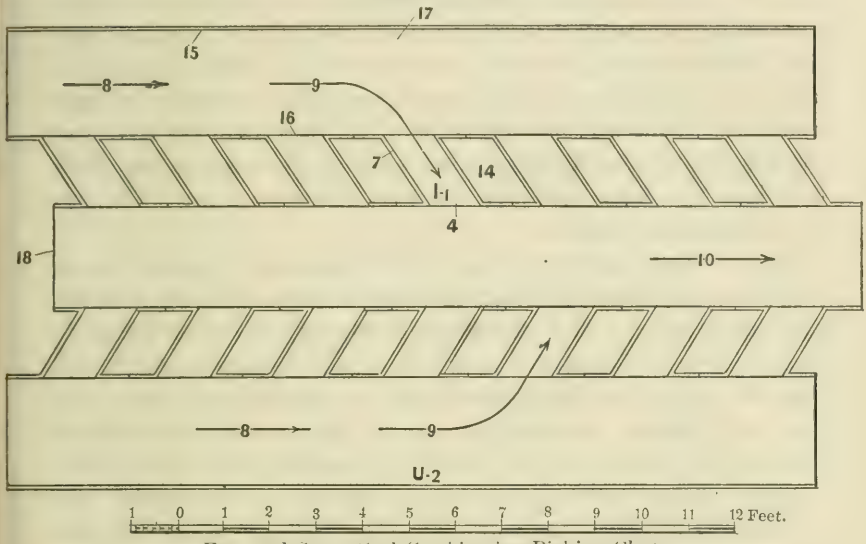
shaft (50), a motion like that of a pump-handle, which is communicated through the shaft to the two plunger arms (46), connected with the plungers (40). In consequence of the sliding of the crank-pin in the slotted arm, the motion upward of the plunger-rod is at a different rate of speed from that downwards. The object of this is to force the plunger rapidly down, lifting the coal quickly, and allowing it to settle slowly. By running the fly-wheel pulley in the opposite direction to the hands of a watch, this effect is produced, the piston descending more rapidly than it ascends. The jigs are arranged in such a way that, while the piston of one jig is rising, that of the other of the same pair is falling. This diminishes the shock upon the machinery, since, to a certain extent, they counterbalance

each other. As the principal work is done in forcing the plunger downwards, the plunger and plunger-rods are made very heavy, thus equalizing, more or less nearly, the work during a revolution. The lower part of the plunger-rod (41) is made of cast-iron, the upper part of two wrought-iron straps (42), which are welded together on top and are kept apart by a piece of wood which is bolted between them.

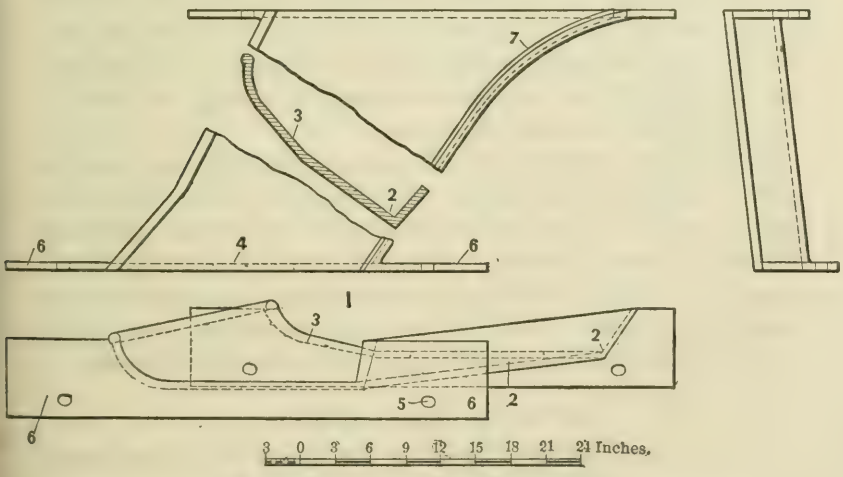
The jig consists essentially of a wooden box, lined with iron plates at (2), where the plunger works, and also at (5) and (7), where the coal is put. In order to guide the water from the plunger to the jig a semi-circular row of planks is put in. These need not be absolutely water-tight, as they are simply intended to direct the current. The coal enters at (5). The adjustable plate (6) is put with its lower end as near the bottom of the jig as possible, consistent with a free discharge of the coal to be jigged. The coal passes out under this, over the perforated plates (37), which are supported by the bar (4) in the center, and by supports bolted to the sides. The coal, rising to the top, is skimmed off by a series of flat strips of iron, carried on two rows of Ewart link-belt chain running over (34). As the coal is scraped up the inclined plane (33) the water drains back. At the top of this inclined plane is a small flat, covered with iron, which is nearly horizontal, but inclines slightly towards the jig. The coal forms here a pile, and the water drains from it back to the jig. As each successive quantity of coal is brought up by the flights on the chain it pushes a corresponding quantity, which has been drained, off the other side down the chute, where it goes either to the picking-chutes to be picked, or directly to the pocket, if it is (as in the case of the small sizes) already clean enough. The opening (10), through which the slate passes, is regulated by elevating or depressing the plate (31), so arranged as to allow the largest piece of slate to pass under it. To the left of this plate (31) the bottom slopes in all directions to the gate (11), which generally remains open, but which can be closed by the lever attached to the bell-crank (17). Outside of this gate is a cast-iron flat pipe or slate-hopper (9), which is closed by the wedge-shaped slide (15), upon the upper surface of which is a piece of oak (14). Slide (15) moves in a casting (13), which has on each side a taper groove. When the slide (15) is pushed through this opening in (13) the wedging action of the taper grooves forces the wood against the face of (13), which is planed and bolted to (9). This makes an excellent gate for closing the hopper, allowing neither water nor slate to escape. If the closing is not perfect, on account of something catching in the slide, it

is simply pulled back and pushed in again. A similar arrangement at (19) and (20) lets out the slime and fine coal which accumulate in

PLATE XVIII.



Egg- and Stove-Coal Combination Picking-Chute.



Broken-Coal Intermediate Picking-Chute.

the bottom of the jig. The gate (11) is closed by forcing it up in the frame (10). This cuts off both the water and slate from the

hopper (9). When this has been done, (15) is drawn out, and the water and slate fall upon the inclined plane (22) and slide down to a point over (27). Here the water drains into the drag-trough (29), and the men who stand at the right of (23) pull the slate over the edge of (23), allowing it to drop onto (24), and thence into the drag; while whatever coal may have come out with the slate is picked out and run, either directly to the pocket, or placed in boxes and carried there. At about the water-line of the jigs there are holes cut through the wooden partition, so as to allow the water to flow from one jig to the other, and thus maintain a uniform level. The jigs are managed as follows:

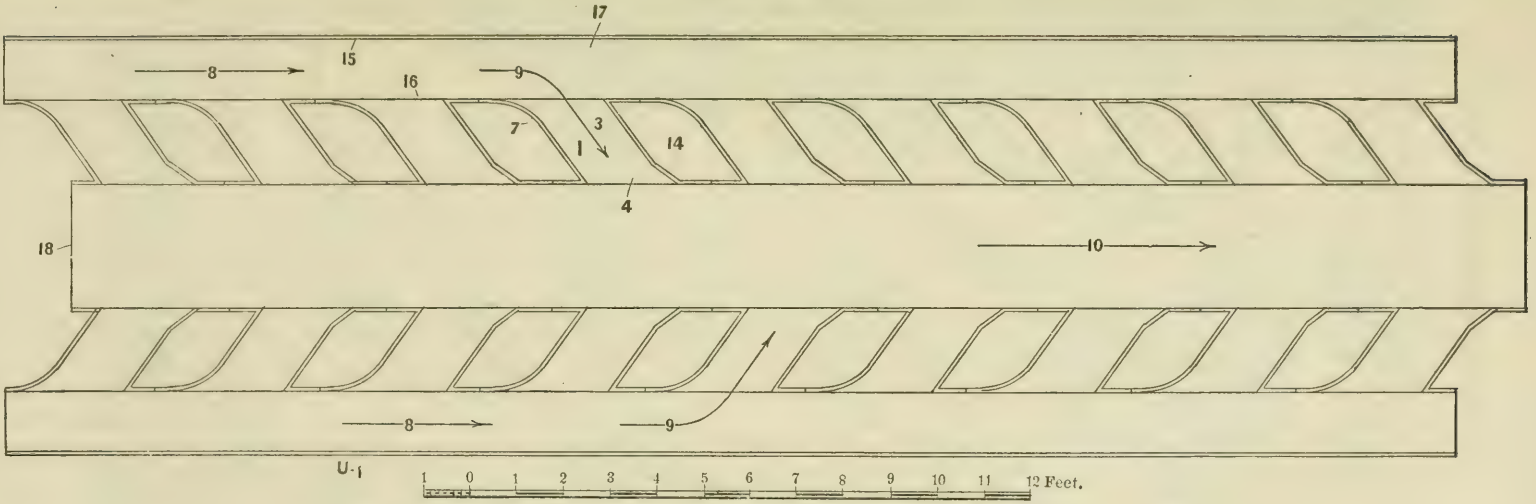
One or more men, according to the number of jigs, are placed between the timbers overhead carrying the running mechanism and the top of the jig. It is their duty to see that the water is kept at the proper level, that the jigs and draws are working properly, and that the coal is uniformly distributed to the different jigs. The men who manage the other part of the jigs stand at the right of (24), and, by means of the lever (21) discharge the slime at the bottom. When too great a quantity accumulates in the tank they close the gate (11) by means of the lever (19) and allow the slate to come out by means of the lever (16). As the slate is taken out it is inspected; the coal is taken out, and if the jig is not working properly, as shown by the slate discharge, they notify the foreman in charge of the jigs. The man on top is also to see that the coal coming out of the jig is as clean as can be expected. If too much slate comes with the coal, there is something wrong with the jig, and it is his duty to notify the foreman.

The shaft (53) which drives the elevators is continuous along the whole length of the jig-house. The arms (35) which support the sprocket-wheels at the lower end, are so arranged at the upper end (36) that they can be lifted up and turned around the shaft, so that if necessary (as if the coal for any reason should accumulate at the bottom) the sprocket-wheels (34) can raise themselves, or be raised without causing the stoppage of any of the rest of the machinery.

Automatic Slate-Picker.

This depends for its action upon the fact that while the coal generally breaks into cubical masses, the pieces of slate of the same length and width are of very much less thickness. Hence, if a quantity of slate and coal which has been passed through a screen and properly sized, the slate, if placed edgewise, would drop through a slit over

PLATE XIX.

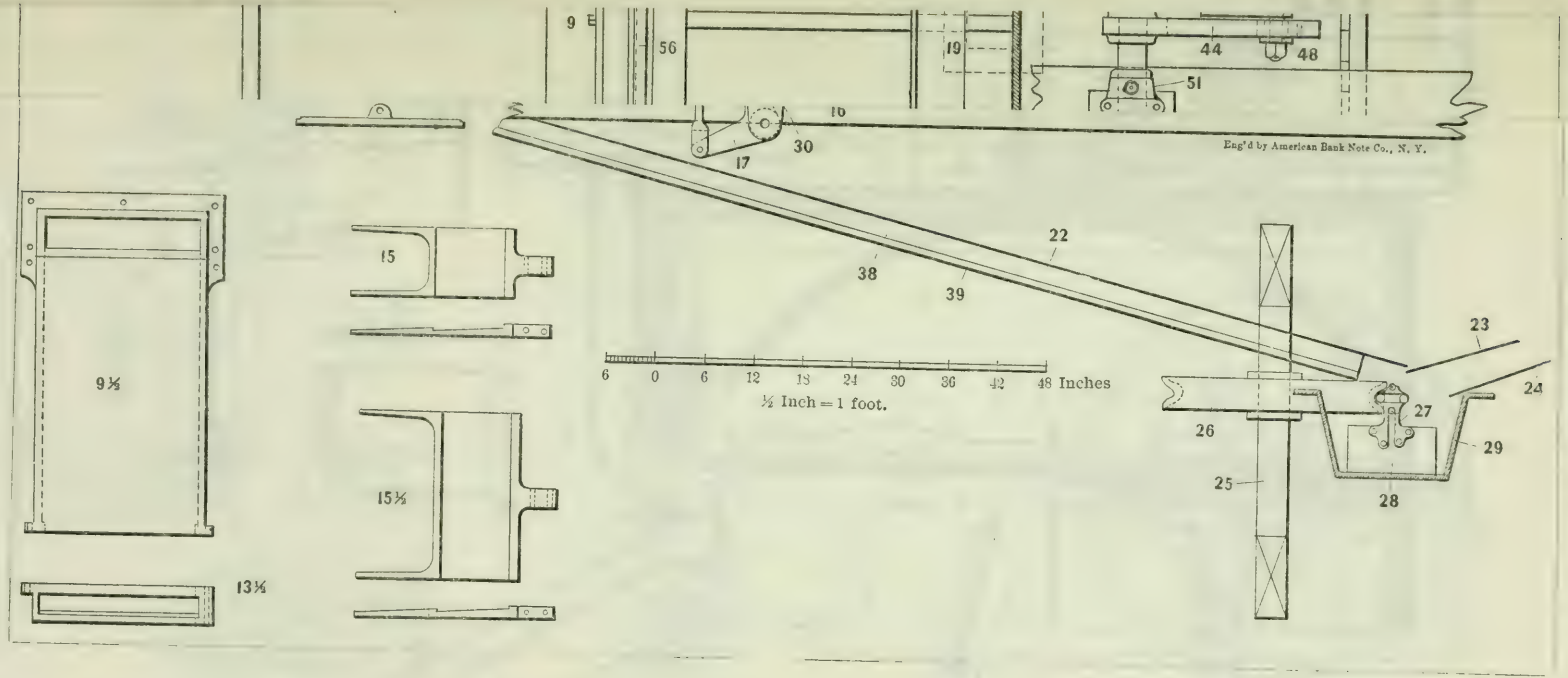


Combination of Egg and Broken Intermediates, U_1.

which the coal would pass. There are two types of automatic slate-pickers, one intended to be placed in a chute and to be fixed, and the other to be placed in the discharge-lip of a gyrating-screen and gyrated.

The fixed slate-picker, Plate XXI., which is of iron cast in one piece, consists essentially of a series of **V**-troughs, one side of the **V** being shorter and at right angles to the other. The lower half of the casting has a taper slit (4) in the short side. The slit is so arranged that anything lying on the long side of the trough and of not too great height can slide out through it. Any lump which is thicker than the height of the slit will of course be retained in the trough. The slits widen as they approach the lower end, and the part of the casting below the cross-bar (5) hangs freely, so that there is nothing to stop a piece from sliding through the slit. This slate-picker is placed in an ordinary trough or chute down which the coal slides. It receives pitch enough to allow the coal to slide over freely, but with not too great velocity. As the coal and slate come down the chutes, each lump places itself in one or other of the grooves or troughs, which are made a little wider than the largest lump of the size for which the slate-picker is to be employed. As the lumps slide down, all the flatter pieces tend to pass out through the slit on the side, while the cubical lumps go over. Should a piece catch in the slit in consequence of the increase in height towards the end, some one of the pieces which follow will generally knock it loose, so that it does not remain and block the slits. This is an important point. The slits if made parallel would soon clog. The flat pieces, which are mostly slate, and which fall through the taper slit, pass over a chute or picking-table or any convenient place, where they are examined by a boy, who takes out any flat coal that may come through with the slate. With the larger coal, such as broken and egg, a great deal of what goes out is often coal and slate-coal, which is carefully picked out, the slate going in a chute to the slate-drag, which deposits it on the dirt-bank. But the flat coal, slate-coal and bony (if valuable) are rebroken, and are prepared with the other small coal that comes from the mines.

When at any colliery there is a great amount of flat coal which, while unsightly and unmarketable in its flat shape, is, nevertheless, pure coal, it is well to provide a separate set of rolls to break this up, if the small coal which is obtained can be prepared separately; for it is not wise to mix the coal already sufficiently pure with coal


9
56
19
44
48
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10
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17
Eng'd by American Bank Note Co., N. Y.
22
38
39
23
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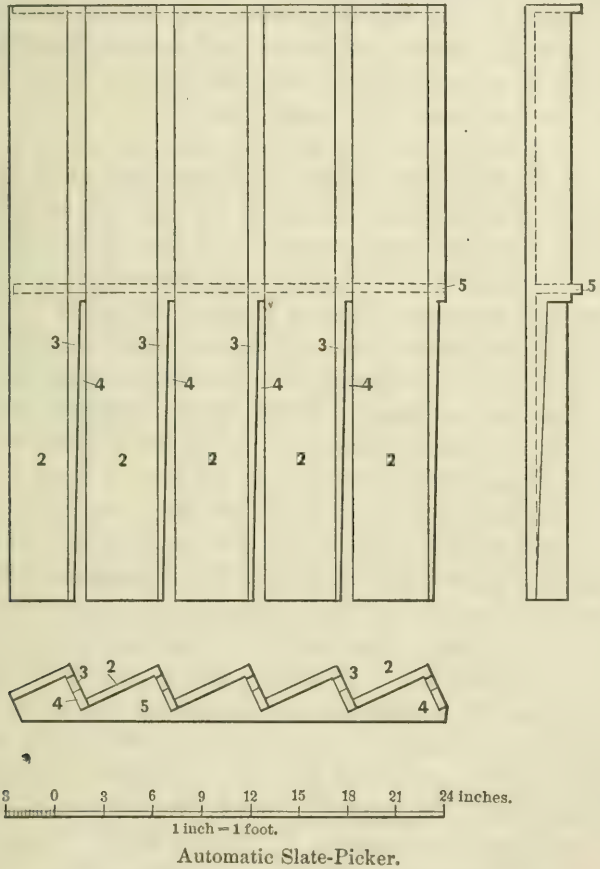
27
29
26
25
28
15
9½
15½
13½
6 0 6 12 18 24 30 36 42 48 Inches
½ Inch = 1 foot.

Drawings of Coal-Jig.

containing slate, as, of course, has to be done where slate-coal, bony-coal and flat-coal are broken up together.

In placing this slate-picker in the chute, the sheet-iron which covers the bottom of the chute is made to pass an inch or so over the upper end of the slate-picker, and the free or unsupported end (2) does not quite reach, and is a little above, the sheet-iron that

PLATE XXI.



Automatic Slate-Picker.

forms the bottom of the chute which carries the coal away from the slate-picker, so that the coal is not checked in any way in passing from the slate-picker to the chute. The size and taper of the slit, the pitch of the picker, the width of the troughs, the length of the upper and the lower portion of the casting, vary with the size of the coal, nature of slate, etc.

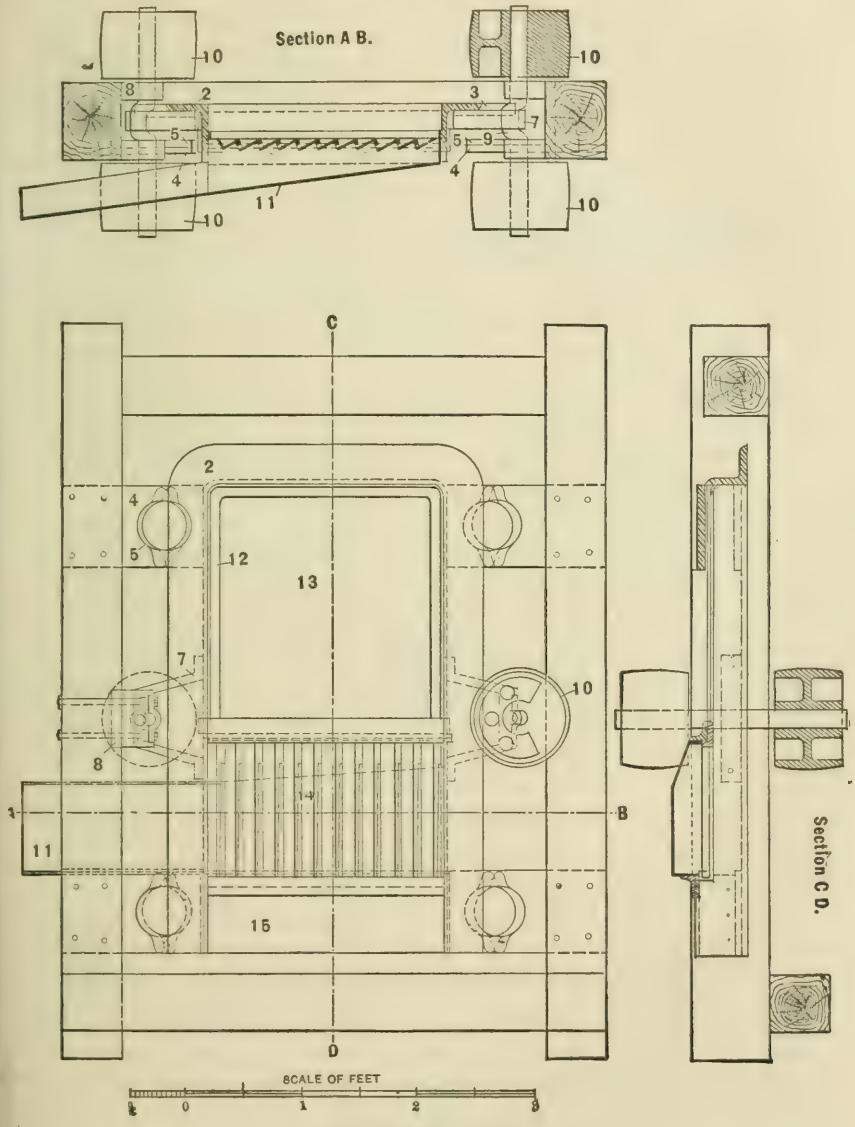
The gyrating automatic slate-picker is made in the same way with this exception, that the upper part above (5) is done away with, and only the part with the slit is used. This is placed on the discharge-chute attached to the gyrating screen, as in Plate XII. The pickers are made in two patterns, to be used according as the screen gyrates in one direction or the other. They must be so arranged that the gyrating motion of the screen has a tendency to throw the coal and slate against the short high side. In this way the latter is thrown out and passes to a jig or picking-table. As will be shown in the description of the breaker, the flat-coal and slate are jigged together. This is only done for small sizes, where the quantity of flat pure coal justifies it, as is the case with chestnut-coal.

A third method of removing slate mechanically is used in several breakers in the Wyoming region. It consists essentially of an inclined plane, down which the lumps of coal and slate are allowed to slide freely. The plane may be covered with iron, stone or slate. The angle is such that the slate will slide down uniformly while the velocity of the coal increases. There is a gap at the end of the inclined plane, over which the coal jumps by virtue of the greater velocity acquired in sliding down the plane, while the slate, moving slowly, drops into it. There are a number of devices for changing the pitch of the chute, the form of the opening, etc.; but, as these machines are not used at the Cross Creek collieries, I do not describe them.

In the iron breaker we have carried the use of the gyrating slate-picker one step farther. One of the great troubles in chestnut-coal is the flat slate and flat coal which it is difficult to remove. We have, therefore, placed upon the main gyrating screen a chestnut slate-picker with a slit larger than would be economical, if it were to be used in the ordinary way. This slit is so large that it is almost certain to take out all the flat slate and a certain portion of coal which, though flat, is marketable. The whole mass that comes from this slate-picker goes to one of the jigs, and by jigging all the heavy slate is taken out. The material coming from the jig then passes over a small gyrating separator, K_4, Plate XXII. This is a single gyrating table, the upper part (13) being a perforated plate with holes punched in it so as to remove any pea-coal that may have got into the chestnut or may have been formed in it when it is passing through the jig, while (14) is a slate-picker so arranged that the slits will only let out coal or slate too flat to go into the chestnut coal. This table which is made of angle-iron, is supported on four conical plates, the throw of which is 2 inches. It is counterbalanced by the four pul-

leys (10). When the stuff comes from the jig it passes into (13), the pea-coal goes out; the coal which is large enough for chestnut coal,

PLATE XXII.



Gyrating Separator.

passes over (14), and the finer stuff passes out through (11). The stuff passing over (15) is examined, and if there is still any slate in

it, it is taken out, while the part which passes out through (11) goes to the small rolls, G_7, Plate XL., the slate, if in considerable quantity, being removed by hand before it is broken.

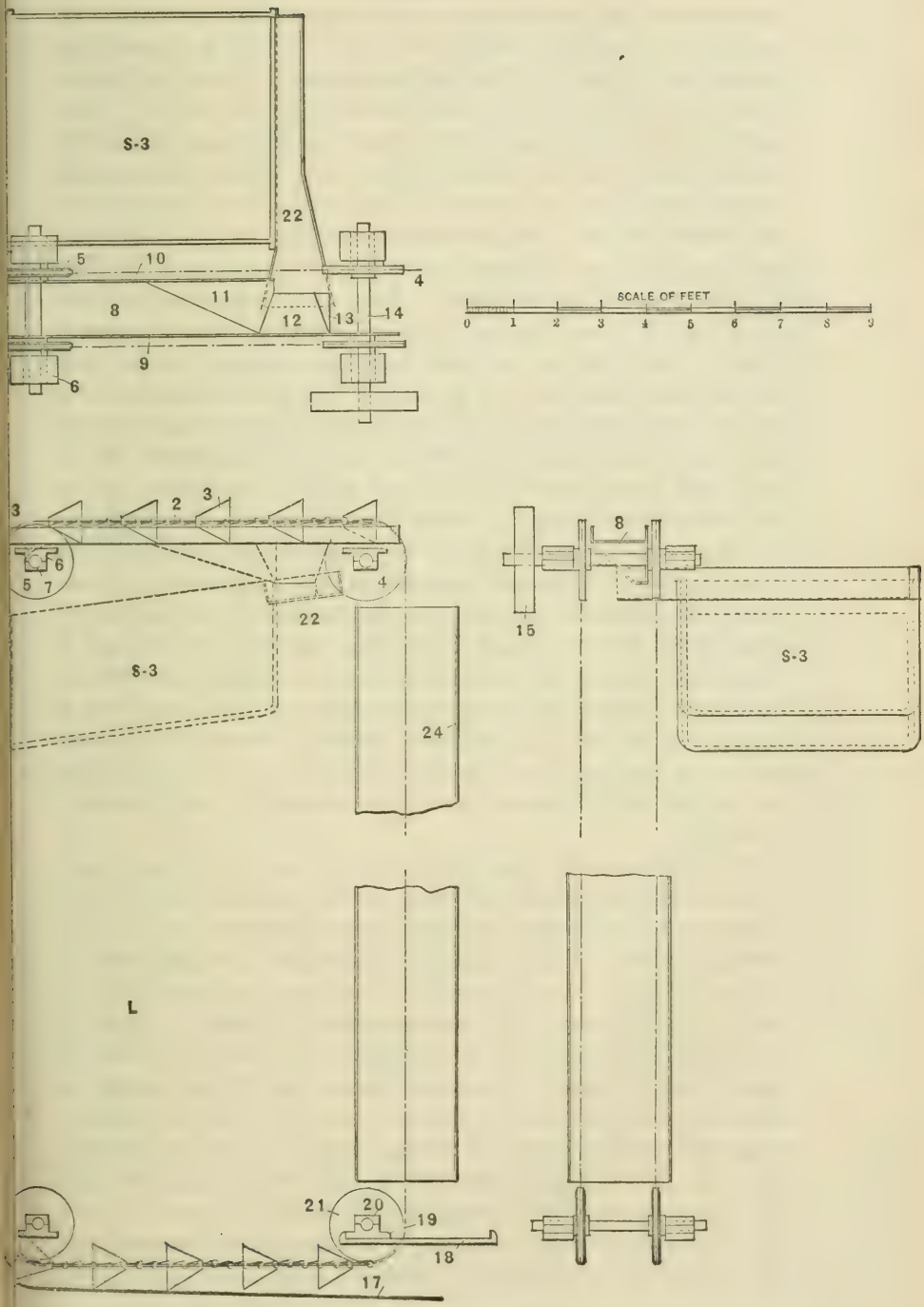
II. Conveying Coal in the Breaker.

The next class of apparatus to which I shall refer is used simply for changing the position of the coal, without involving any question of its preparation. It consists of chutes, elevators, drags and loading lips. The chutes are used for conveying the coal or slate from one portion of the breaker to another, the stuff sliding down by gravity. In this breaker they are constructed, as a rule, of sheet-iron and are generally half-round troughs; the size, inclination and thickness of the iron depending upon the work to be performed; the larger the coal the heavier the iron, and the smaller the coal the lighter the iron and the greater the pitch of the chute. Some of the chutes are flat instead of half-round, and are formed of sheets of steel plate, with steel sides. These are generally used where it is important to inspect the coal. There are a number of local causes that may cause a round or square chute to be preferred in special cases.

Where an abrupt change in the direction of the chute becomes necessary, we often use cast-iron turns, which are spiral, half-round troughs of greater or less length. One large one of this kind, D_1 (Plates XXXIV. and XL.), is used for carrying the lump-coal from the platform, F, to the lump-coal chute. It is a spiral, the center of which is always 16 feet from a vertical axis. The chute is 4 feet wide inside and the section is half an ellipse, the depth to the center being $1\frac{1}{4}$ feet. The pitch of the center-line is about $2\frac{5}{8}$ inches to the foot. It was formerly greater, but we have been obliged to reduce it. It is constructed of 12 sections about $4\frac{1}{4}$ feet long in the center-line, the sections being provided with flanges at each end, so that the pieces when bolted together form a continuous chute.

There is in the breaker but one elevator, L, which is used for elevating the slate-coal when broken by the group of rolls G_3, G_4, G_5, G_6 and G_7 (Plates XXXIII., XXXIV. and XL.), so as to discharge it into the wet screen, S_8. This avoids rehoisting all this coal, which must be done at many of our breakers, where it is picked out at a point too low in the breaker to permit it to be rebroken and passed directly to the screens. This elevator, which is shown in Plate XXIII., has been so constructed as to reduce to a minimum the fall of the coal on entering and leaving the elevator. It consists essentially of two Ewart chains (2), which pass over three pair of traction wheels (5) and one pair of sprocket-

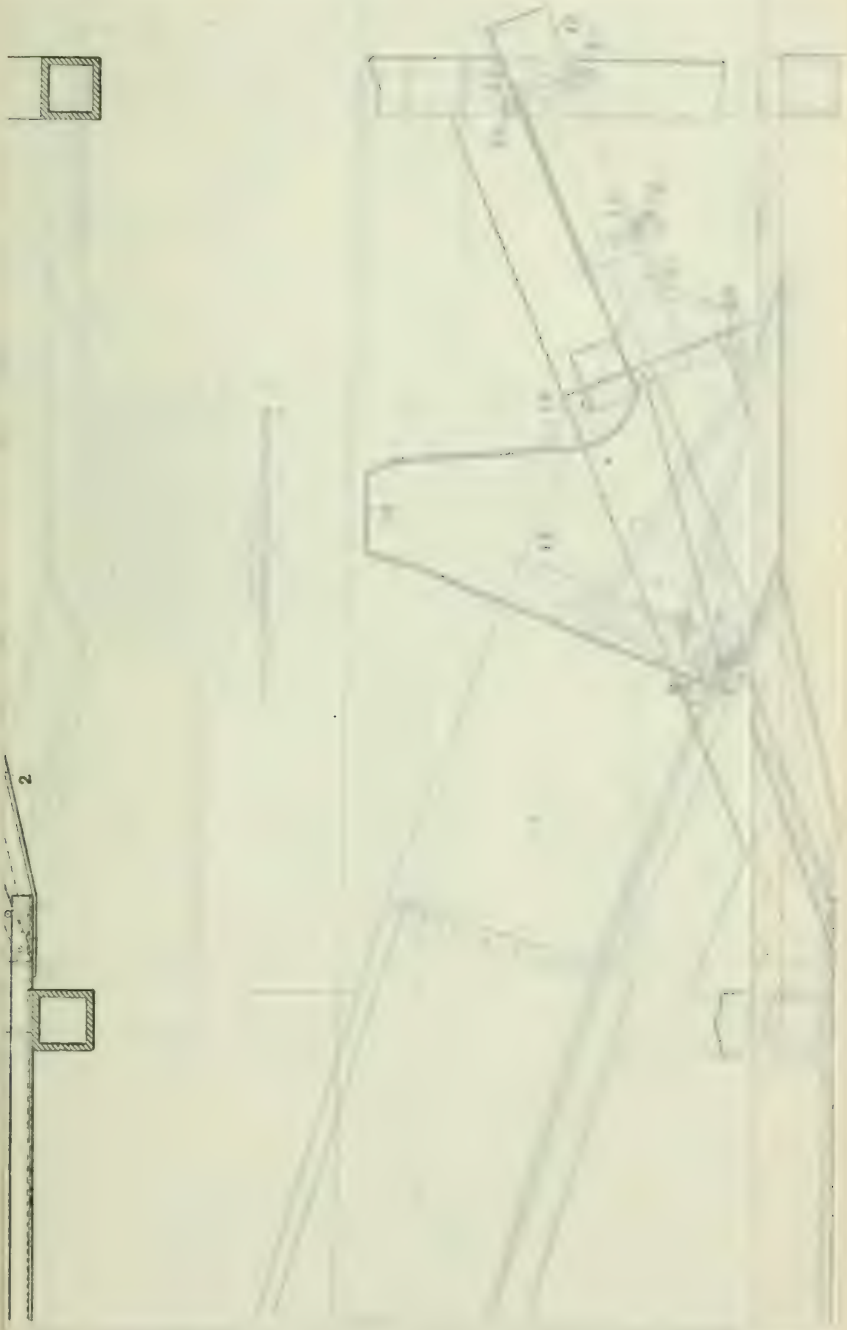
PLATE XXIII.



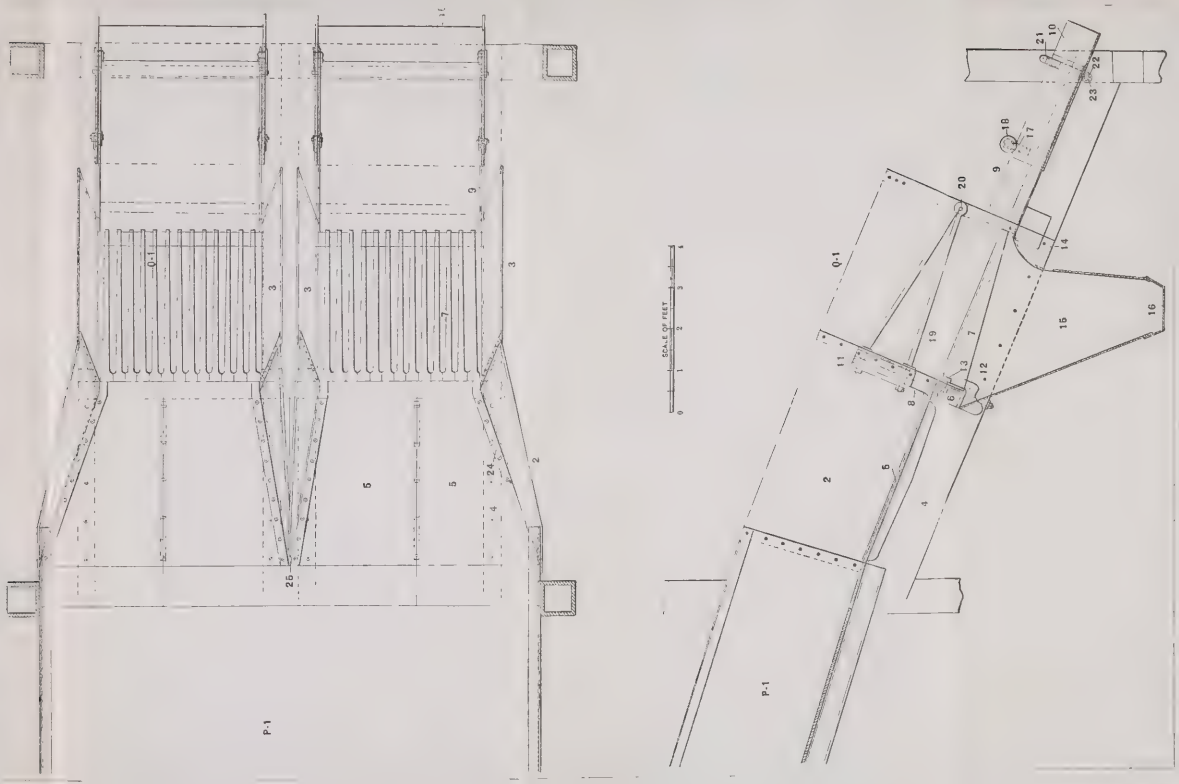
Crushed-Coal Elevator.

wheels (4). The buckets (3) are triangular in cross-section and are so constructed that when the two chains are travelling horizontally the coal will fall out and the bucket clear itself. It will be observed that between the two traction-wheels and the bottom (17) there is a distance of about 7½ feet, along which the buckets slide like a conveyor. They move upon a sheet of steel which is bent up at the point where the buckets begin to rise, so that they will touch it without being caught. As this piece of steel is very wide, if the coal is allowed to slide on it the buckets will carry along the coal and as they rise, fill themselves, carry up to the top, turn over and discharge the coal upon the plate (8) above, which is horizontal. From this upper plate a chute (11, 12 and 13) can be taken off at any angle; all that is necessary is that the steel plate on top be cut diagonally, so as to join the chute carrying away the coal in a horizontal line. Of course, when the bucket dumps on the top, if the carrying-away chute begins exactly where the bucket dumps, but is at an angle to the direction of the chain, only a small portion of the coal falls into the chute; but, as the bucket moves forward, it pushes the remaining coal forward, and as each particle reaches the point where the carrying-chute starts, it is carried away. The coal is carried into the elevator by a chute which just passes over the chain, and has therefore a very slight fall, which is still further reduced by the fact that when the machine is in operation, there is generally a certain amount of coal in the bottom. The great advantage of this elevator, which is a new thing as far as we know, is the fact that very little breakage occurs, and that the coal can be discharged with no fall at will, at one or more points, and at any angle between the traction-wheels (5) and sprocket-wheels (4).

The drags may be either horizontal or inclined. There is one, M, which carries the slate and other refuse from the jigs to a point in the breaker where they will run to the slate-conveyor, N. As a great deal of water falls from the jigs, the flights of this drag work in cast-iron troughs, which, at certain points, have perforated sides for letting out this water. The slate-conveyor, N, which is shown in Plates XXXIV. and XLI., carries away all the refuse that does not float off with the water. There is a double drag, M_1, that carries the screenings from the loading-lips into a car, X (Plate XXXIV.), under the breaker, which is hoisted to the top of the breaker. There is also a supplementary drag, M_3, Plates XXXIII. and XLI. passing under the loading-lips, Q_2 to Q_6, of some of the prepared-coal pockets, which discharges into the main screenings-drag, M_2. Under the settling tank, Z, which is outside and separate from

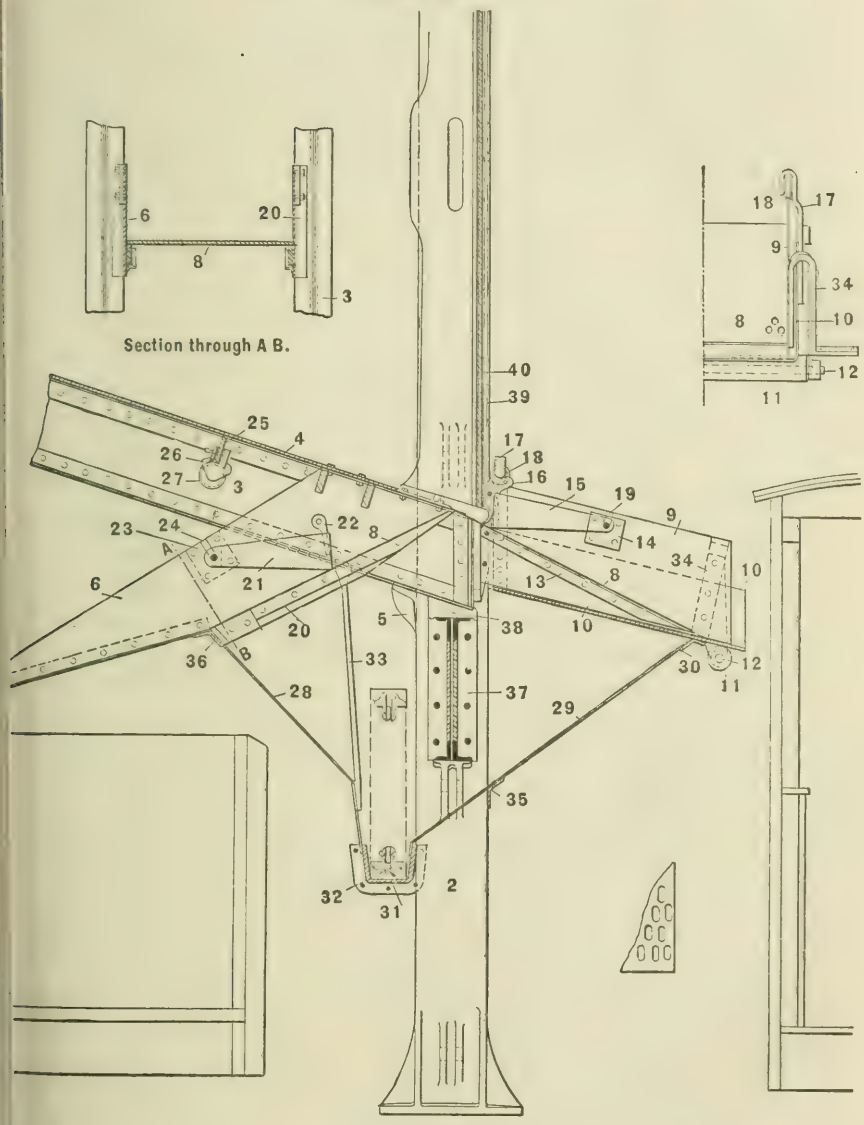


Steamboat Loading-Lip



Steamboat Loading-Lip.

PLATE XXV.

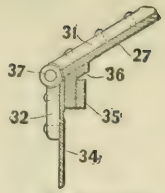


Pocket Loading-Lip.

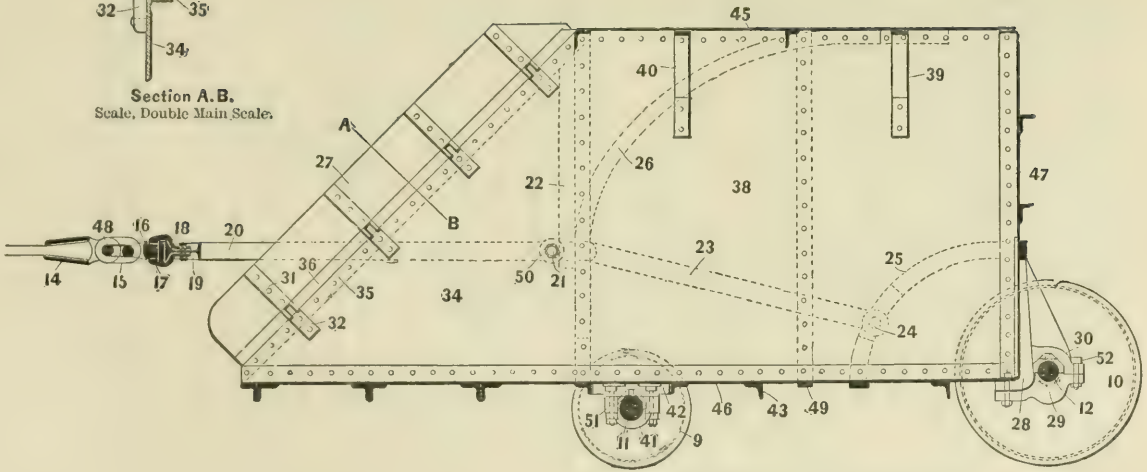
the breaker, there is another drag, M_3, which carries the settlings, when they are drawn off, into the main slate-drag or conveyor, N.

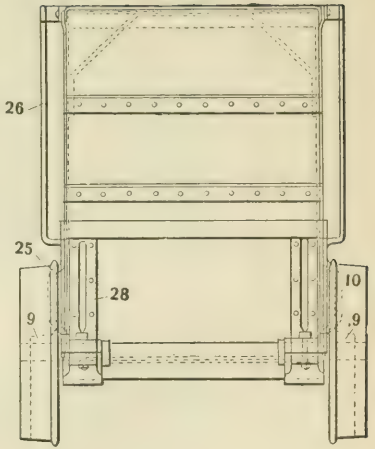
All these drags are constructed of the Dodge conveyor-chain, with the exception of the supplementary drag, M_1, under the pockets, which is constructed with the Ewart chain. As these are all well illustrated and explained in the catalogues of the parties who make them, it is not necessary to dwell upon them further. In regard to the main slate-drag, however, there is one point to which I would call attention, namely, the construction of the truss that carries the drag. As the drag runs out from three to five hundred feet, and as the thrust is in the direction of the axis of the truss, it is very important to have this very stiff. Formerly, much more attention was paid to supporting the drag than to the resistance in the line of the axis; but experience has shown us that it is really much more important to have the strength in the line of the thrust, as there is comparatively little weight to be supported.

There are two types of loading-lips: one for lump and steamboat coal and the other for the prepared coal. The lump and steamboat loading-lips (Plate XXIV.), of which there are two in each chute, consist essentially of a gate (8) and (11), below which there is a set of bars (7) and below these a sheet-iron trough (9), a portion of which (10) is movable and can be slid into the car. When cars are to be loaded the gate is raised; the coal is allowed to slide slowly over the bars (7), which take out the small coal which has been formed in the chute; and the coal that is cleaned slides through the chutes into the cars. Of course, it is necessary to draw the lump-chute into the loading chute gradually, so as to avoid corners, edges and other hindrances, which occasion breakage in the coal. We are arranging also a special loading-lip for large cars, so that the coal can be carried as close to the bottom of the car as possible, instead of dropping over the edge. This has become necessary in consequence of the great increase in the size and depth of the cars. The loading-lips for prepared coal are of two types: one for loading the ordinary open-top cars and the other for loading either the open-top cars or box-cars. The two types are shown in Plate XXV.; the one on the left hand being for open top-cars and the other for box-cars. The coal which falls out above (8) through the opening in the bottom at the extreme end passes down over (8), a perforated plate, the form of which is shown to the left of the lower part of the box-car. Through these perforations the fine dirt falls out and



Section A.B.
Scale, Double Main Scale.





Gunboat—Side and End Elevations.

is carried away by the drag flights (32), the coal sliding through the chute (6) into the car. The only difference on the other side is that the chute (10) slides back on the rollers (11), so as to allow the car to pass. The gates serve to check and regulate the flow of coal. This chute can also be used to load ordinary open-top cars, but the fall is pretty great; and it is probable that some new device will have to be adopted for loading the high cars, so as to avoid the excessive fall. This, I understand, is now being done at the Lykens collieries.

III. Hoisting Coal to the Top of the Breaker.

The breaker is situated at the mouth of the slope known as No. 2, at Drifton, the two tracks of which run directly to the top of the breaker. As this slope is very steep, over 60 degrees, it is not economical to hoist the coal in the mine-cars for the following reasons:

1. When hoisting up a slope of this pitch the strain on a car is very great. This tends to cause accidents, unless the car is in perfect order, and the cost of repairing the cars is greatly increased.

2. Every car has to go up the slope several times daily, and as they reach the slope after having been loaded in various parts of the mines, it is very difficult to be sure that they are in thoroughly good condition. A disabled car is very likely to cause an accident.

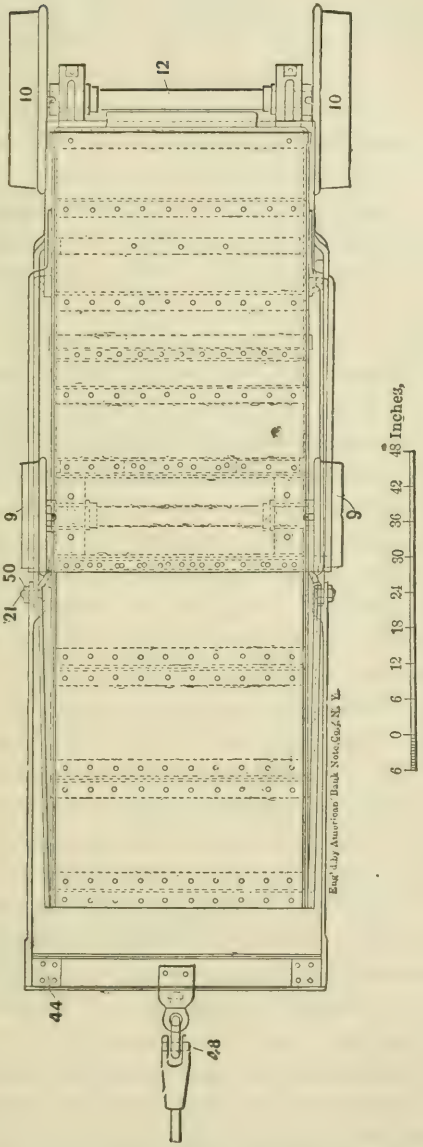
3. A great deal of coal falls out of the cars, which renders it necessary to clean the slope frequently; if the slope is not cleaned the car is often thrown off the track.

4. In order to prevent the coal from falling out of the car, it is necessary, on a steep pitch, either to place covers on the cars at the bottom, which is a very onerous operation and also expensive, or a number of cross-rails must be put on the top of the cars, which renders them very difficult to load and also prevents them from being properly filled.

For these reasons, when the pitch is much over 30 degrees, we prefer to hoist with a gunboat (Plates XXVI. and XXVII.), which is the local name of an iron car very much like what is known as a skip in metal-mining. It is described by Mr. Chance in volume A C of the Pennsylvania Geological Survey. In this case the cars only circulate on the level gangways, and are dumped into the gunboat either at the bottom of the slope or at any other convenient point upon it.

The dump employed at the bottom of the slope for dumping the mine cars into the gunboat is shown in Plate XXVIII. It consists

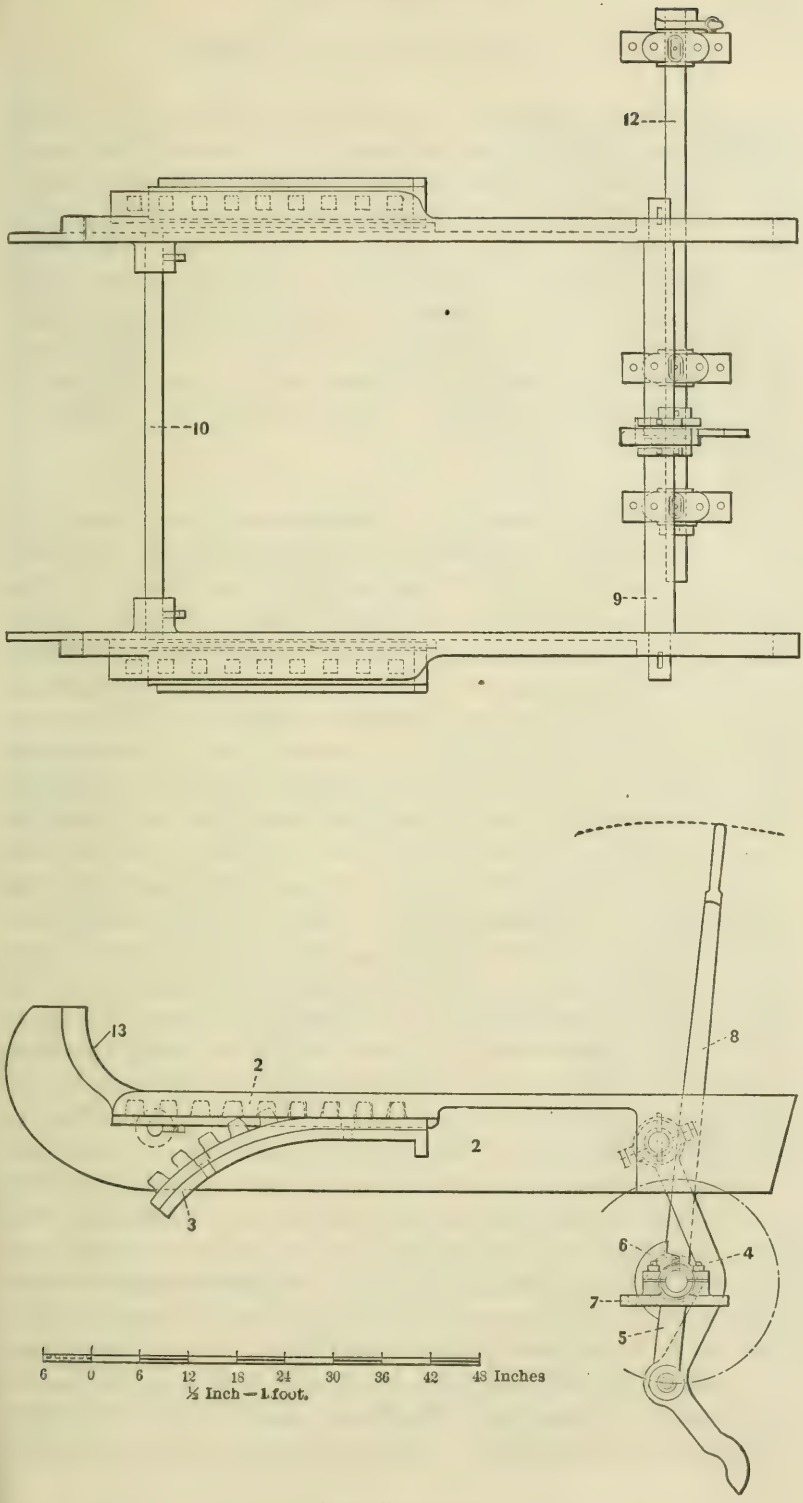
PLATE XXVII.



Gunboat—Plan.

essentially of two cast-iron sides, which form the track and which are connected by two bars, which hold them at the proper distance apart. Upon the sides of each of these cast-iron bars there is a horizontal rack, and upon the timber frame there is a curved toothed segment. When the car is placed on the iron frame, which is known as the cradle, so that the front wheels touch against the horn (13), which is curved upward, the center of gravity of the whole mass is such that the car and cradle tend to roll down and dump. This is prevented, however, by the curved arm (4), for the line joining the two pins in the curved arm passes to the left of the shaft connected with the lever, and therefore prevents the cradle from rising; but, if the lever (8) is thrown to the left, the cam (6), which is on this shaft, moves the bar (5), to the left until the joint between bars (5) and (6) is thrown to the right of the line connecting the center of the shaft and the point of connection with the cradle. When this position is reached there is nothing to prevent the shaft from turning still further to the right and the cradle from rising up, which allows the car to come into a dumping position; and the gate of the car being opened by a latch-lifter, the car empties itself into the gunboat. The cradle is so constructed that, when the car is emptied, the cradle and car tend to return to a horizontal position, but are prevented by the hook in the lower end of the arm (4), which rests against the lever-shaft until the lever is thrown in the opposite direction, thus again bringing the line of thrust of the bar (4) to the right of the central shaft, and allowing the cradle to return to the position from which it started. The empty car is then taken away and another put in its place. By the rolling motion we are enabled to place the car in the cradle before the gunboat arrives, without having the car come in contact with the gunboat, so that the cars can be changed and the loaded car brought on to the cradle while the gunboat is being hoisted. In this way hoisting can be done very rapidly, the car being always ready; and the only thing to be done by the man in charge of the dump when the gunboat arrives is to throw the lever over so as to break the toggle-joint. The coal slides very rapidly into the gunboat; he then reverses his lever; the car goes back to its first position and the gunboat can be immediately hoisted. This dump is very effective, allowing one car a minute to be very easily dumped into the gunboat and hoisted. There are two gunboats and two tracks; but the two tracks come together at the bottom, so that both gunboats run into the same dump.

The gunboat is so constructed that the back wheel has a much

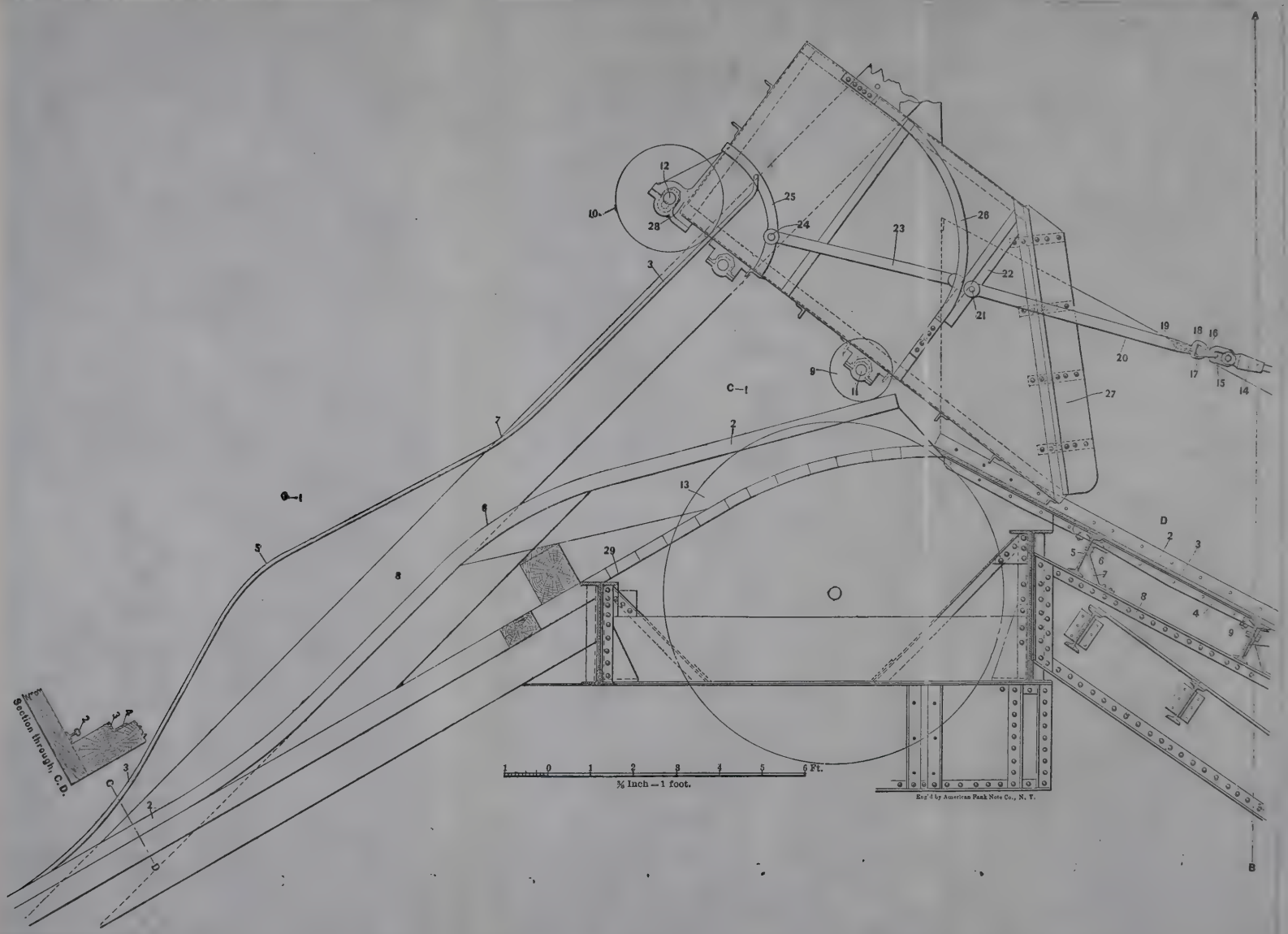


Mine-Car Dump.

wider tread than the front. The rails from the bottom (2) on Plate XXIX. continue to the dump at the top of the breaker, but at a point a little to the left of C D a second track commences, which is formed by an angle-iron (3) and (4), arranged in such a manner that, while the front wheel (9) can go through, the back wheel (10), with a much broader flange, is obliged to follow up this angle-iron. The result of this is that the hind wheel (10) gradually rises up, so that when the front wheel reaches the end of the track (2) the hind wheel is in the position shown on the figure, and the coal slides out of itself without shock, the gunboat merely forming the prolongation of the dump.

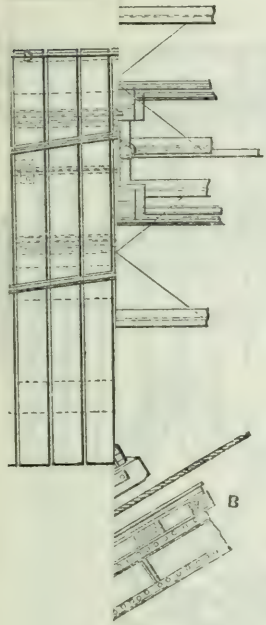
The Barney-Bridge and Dump.

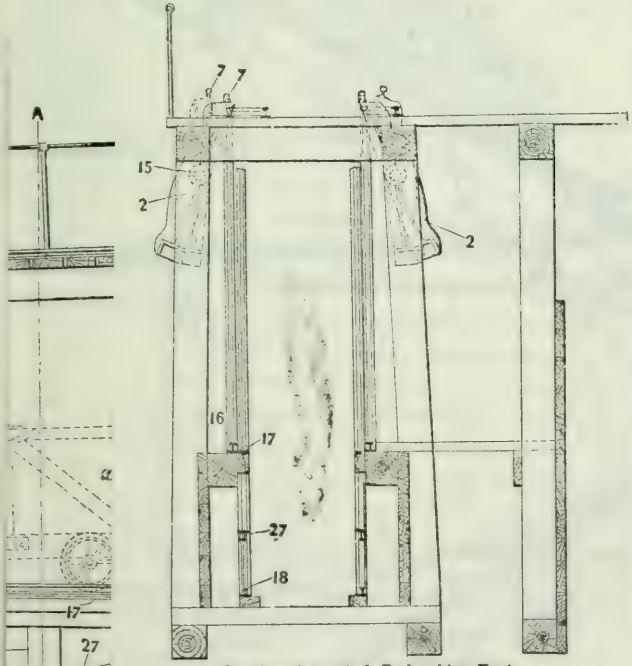
Part of the coal prepared in the breaker comes from a mine about half a mile away. The loaded mine-cars are brought by a small locomotive upon the surface to the foot of the breaker-plane, B, from which they are hoisted to the top of the breaker and dumped. Plate XXX. shows the arrangements at the bottom of the iron breaker plane, the commencement of which is shown at the top of the stone wall to the right. The loaded cars are pushed by the locomotive upon a siding which falls to the point where the car, X, is represented by dotted lines. The empty cars pass over the movable bridge, A_1, to the empty track, which is above the loaded one for a short distance and then runs alongside of it, with a grade in the opposite direction, to a switch where it joins the main track and where the locomotive takes off the empty cars. The hoisting-rope is attached permanently to the "barney" (A 2_2, Plate XXXI.), which is the local name given to the small car that pushes up the larger one. The gauge of the mine-car is 4 feet, while that of the barney is 3 feet 6 inches. The barney-track (18) runs continuously and unbroken from the left-hand corner of the barney-pit, A, to the dump, C, on top of the plane, while the mine-car track (16) runs from the end of the loaded-car siding up the plane, B, when the bridge, A_1 is open, as shown by dotted lines in the section through A B looking east; but when the bridge is closed, as shown by the full lines, the track (7), which runs to an edge at the right hand end, lies over the rail (16) and makes the main-car track on the plane continuous upon the bridge, A_1, to the empty-car track. When the empty car, X, descends from the dump, C, it finds the bridge closed and passes over the bridge to the empty-car siding, continuing to run on until it comes to the end of the siding and is stopped, or until it reaches the empty cars already on the siding. The barney being narrower passes be-



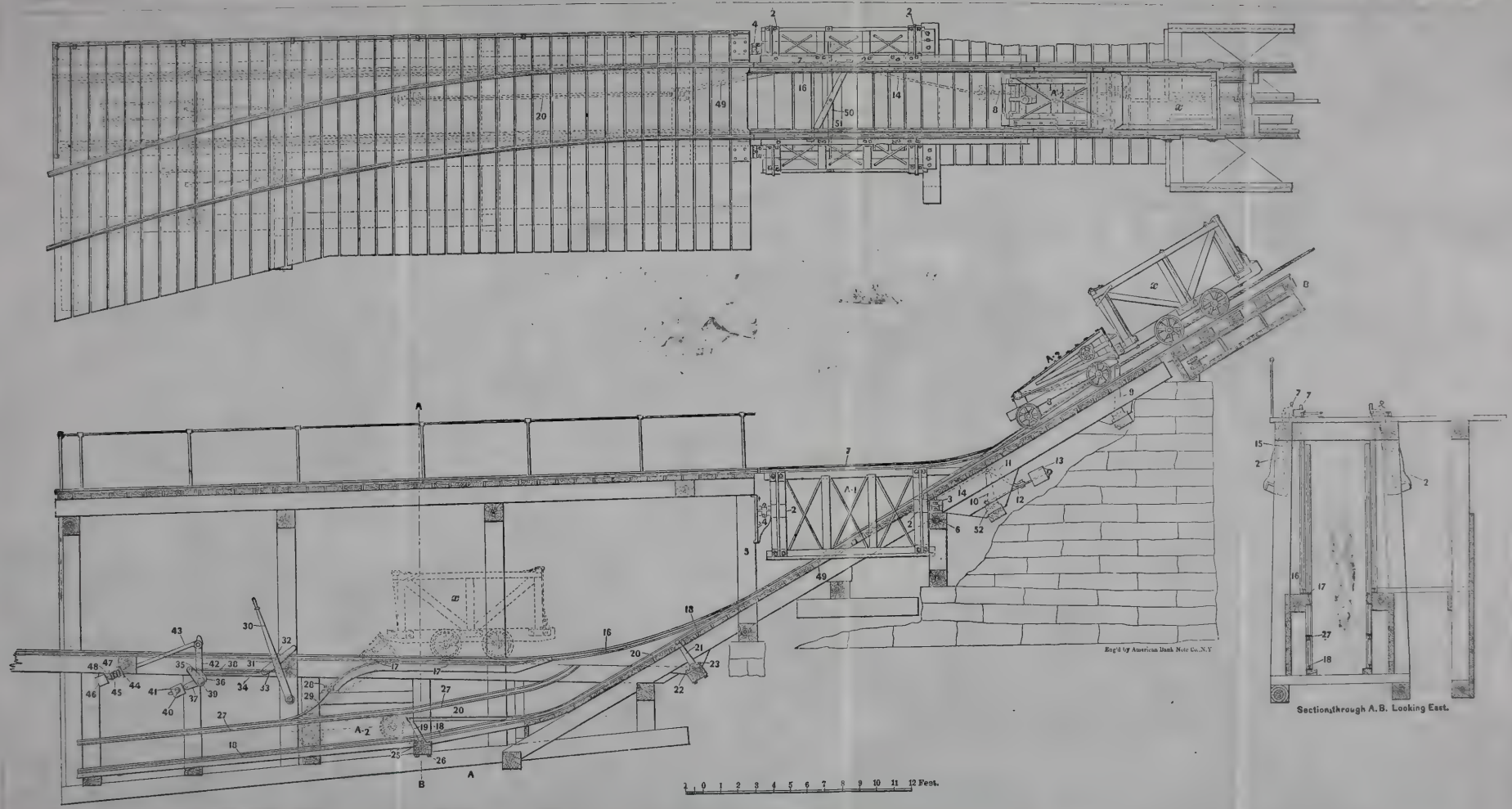
Gunboat-Dump.

COXE.—Plate XXX.



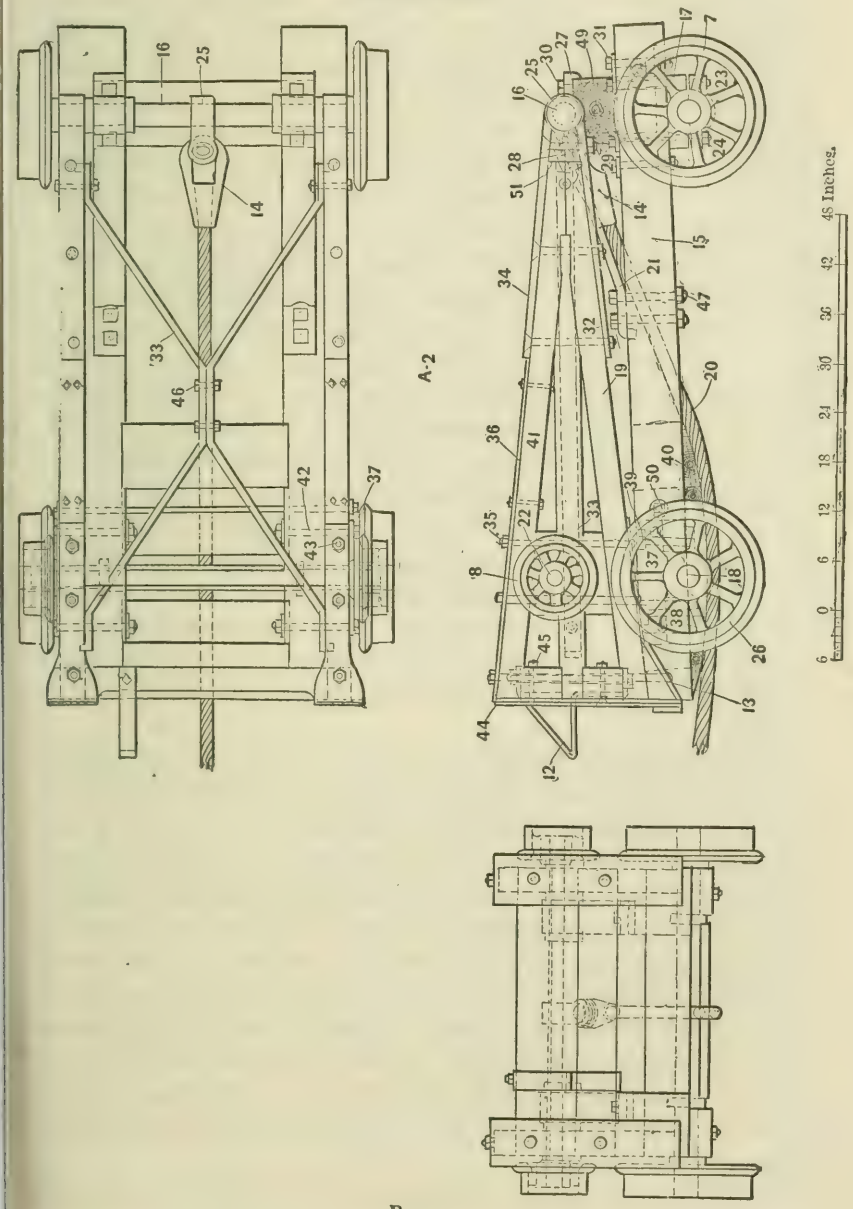


Section through A.B. Looking East.


A
B
A-1
A-2
Engd by American Bank Note Co. N.Y.
Section through A.B. Looking East.
0 1 2 3 4 5 6 7 8 9 10 11 12 Feet.

tween the sides of the swinging bridge, between the rails of the loaded track (16), down the barney-track (18) to the back of the barney-pit. The bridge, A_1, consists of two trusses, each truss

PLATE XXXI.



Barney.

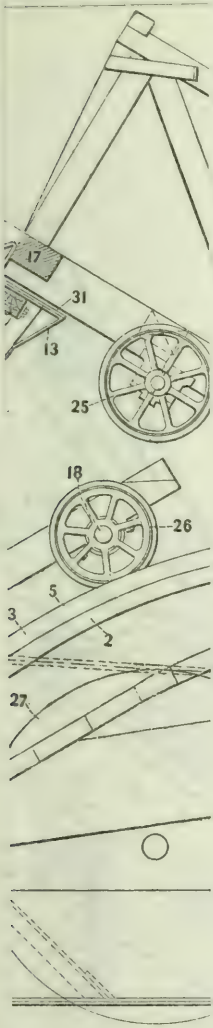
having a bearing at either end (3 and 4), so that by swinging around them the bridge is opened and closed. When the barney descends it finds the bridge closed and when it reaches the bar (20) its axle pushes forward by means of (20) the levers (19) and (21) and thus opens the bridge for the full car to go up. When the barney is in the position shown on the right side of the figure, it pushes forward the levers (9) and (10) and closes the bridge, so that it will be ready for the empty car when it descends. The upper part (or pusher) of the barney is hinged to the back-end, and at the front-end carries two small wheels which, as the barney descends, lift the latches (29) which are hinged at their upper end and fall freely, so that when the engine is reversed and the barney is hoisted, these latches catch the wheels and force the pusher to open. As the wheels are obliged to follow the track (17) the pusher comes in contact with the car and forces it forward and then up the plane, so that when the car gets on the real pitch of the plane, the barney and car are in the position shown on the right-hand side of the drawing. It will be observed that the large wheels of the barney, while in the pit, have a rail (27) above as well as below them. This is to prevent the wheels from rising and the barney from getting off the track. From here on, the car and barney continue up to the dump, the action of which will be explained below. The advantages of this system of hoisting are:

1. The rope is never attached and detached.

2. The full cars run to the bottom by gravity; and the car to be hoisted can be put in position as soon as the one before it has started, so that when the barney descends it can return immediately without delay.

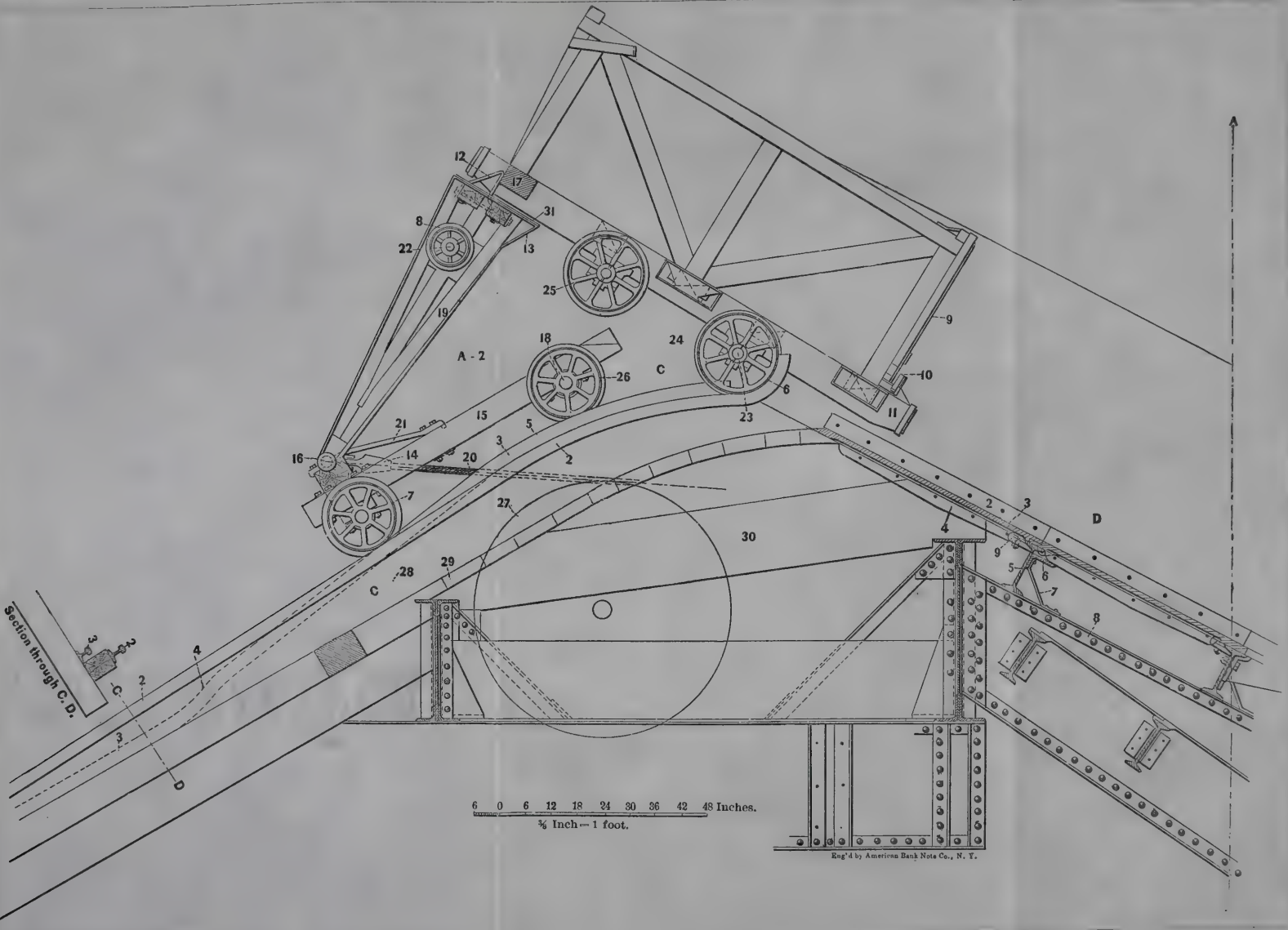
3. The empty car runs off by gravity, if the grades are properly arranged, and the bottom man has nothing to do but to drop in the full car. The system of levers and catchers (30) to (48) is a self-acting arrangement, to drop in a car at a time. The trip of cars is held by the hook at the end of the rod (43); when the lever (30) is moved forward (43) drops and allows the trip to move forward; but the same motion turns the bar (33) upon its axis (32), raising the right-hand end, so that when the axle of the front car reaches (33) it pushes it forward, turning it around its axis (33) and raising (43) so as to catch the next car by its axle and stop it. It is proposed to extend (30) to the top of the bridge or empty track, so that it can be worked by a man there, who can also look after the empty cars.

The mine-car, X, when it arrives on top (Plate XXXII.), follows a

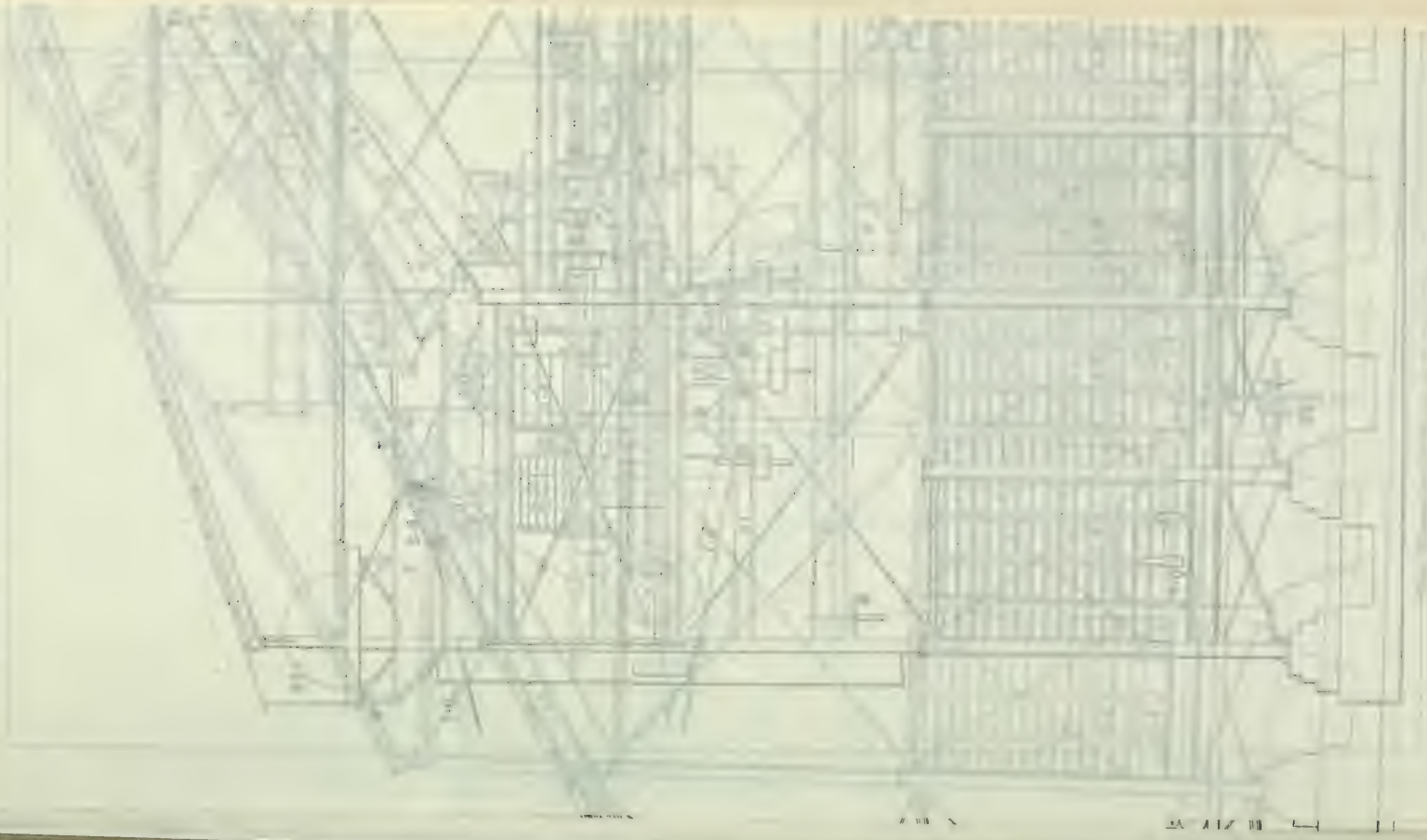


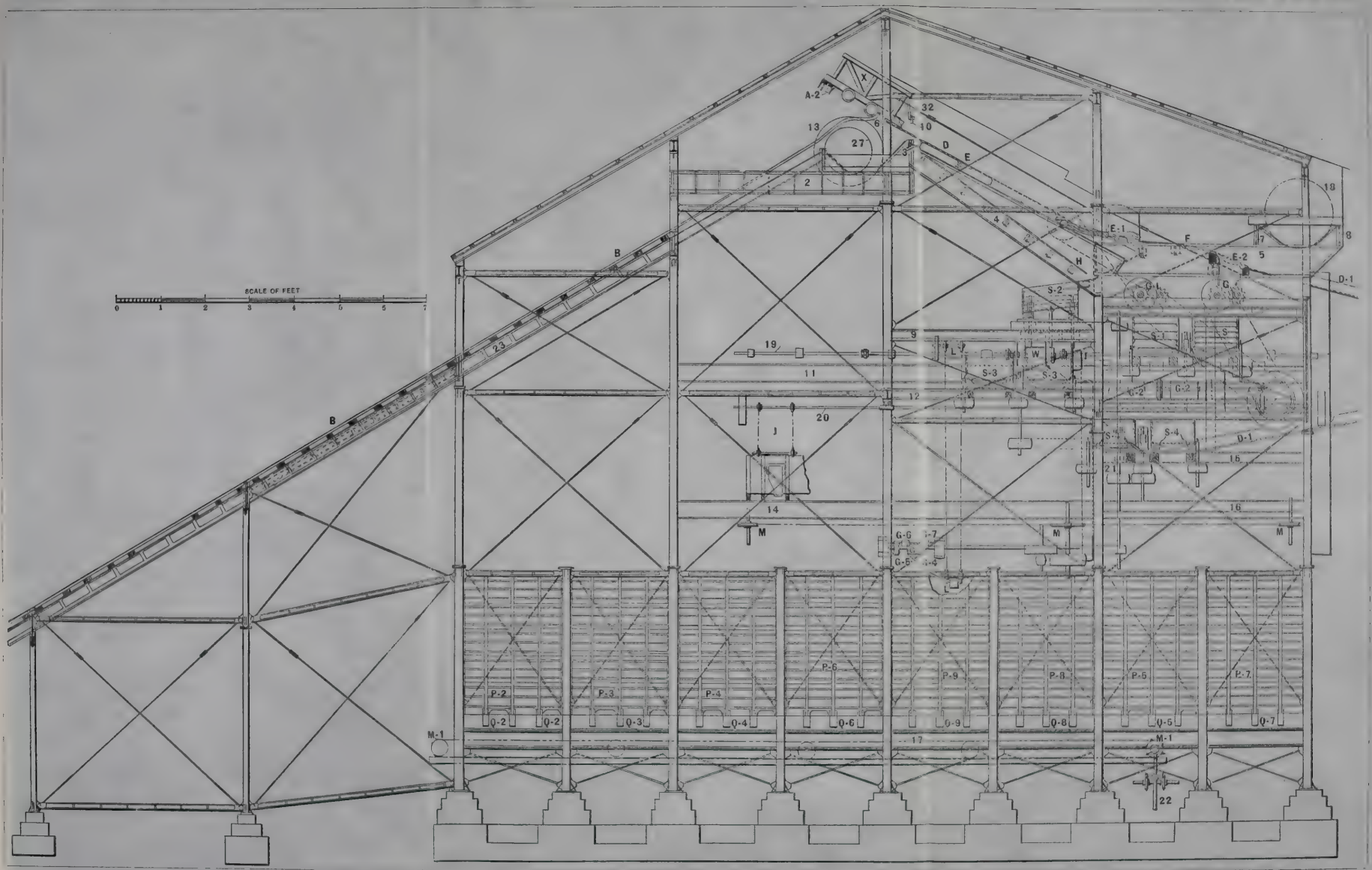
0 6 12 18 24 30

3/8 Inch = 1 foot.

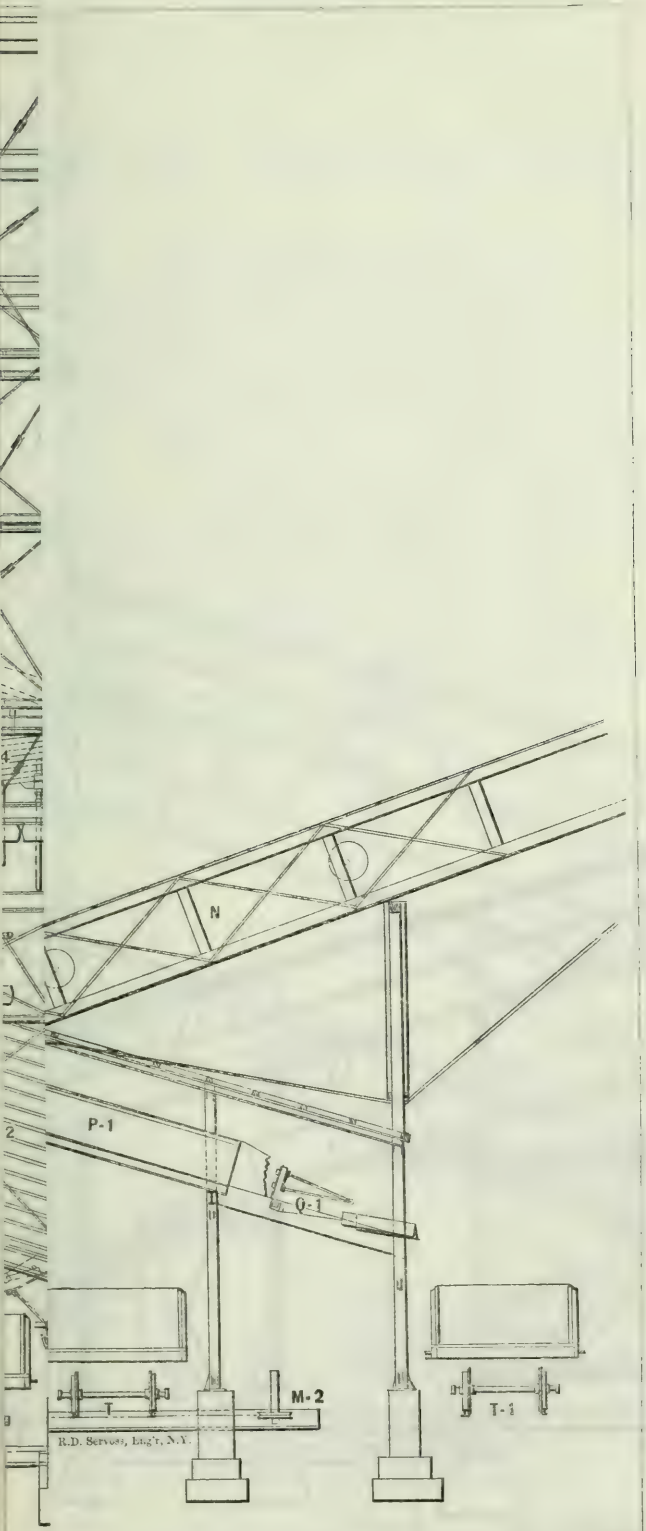

A
A - 2
C
D
Section through C. D.
6 0 6 12 18 24 30 36 42 48 Inches.
⅜ Inch = 1 foot.
Eng'd by American Bank Note Co., N. Y.

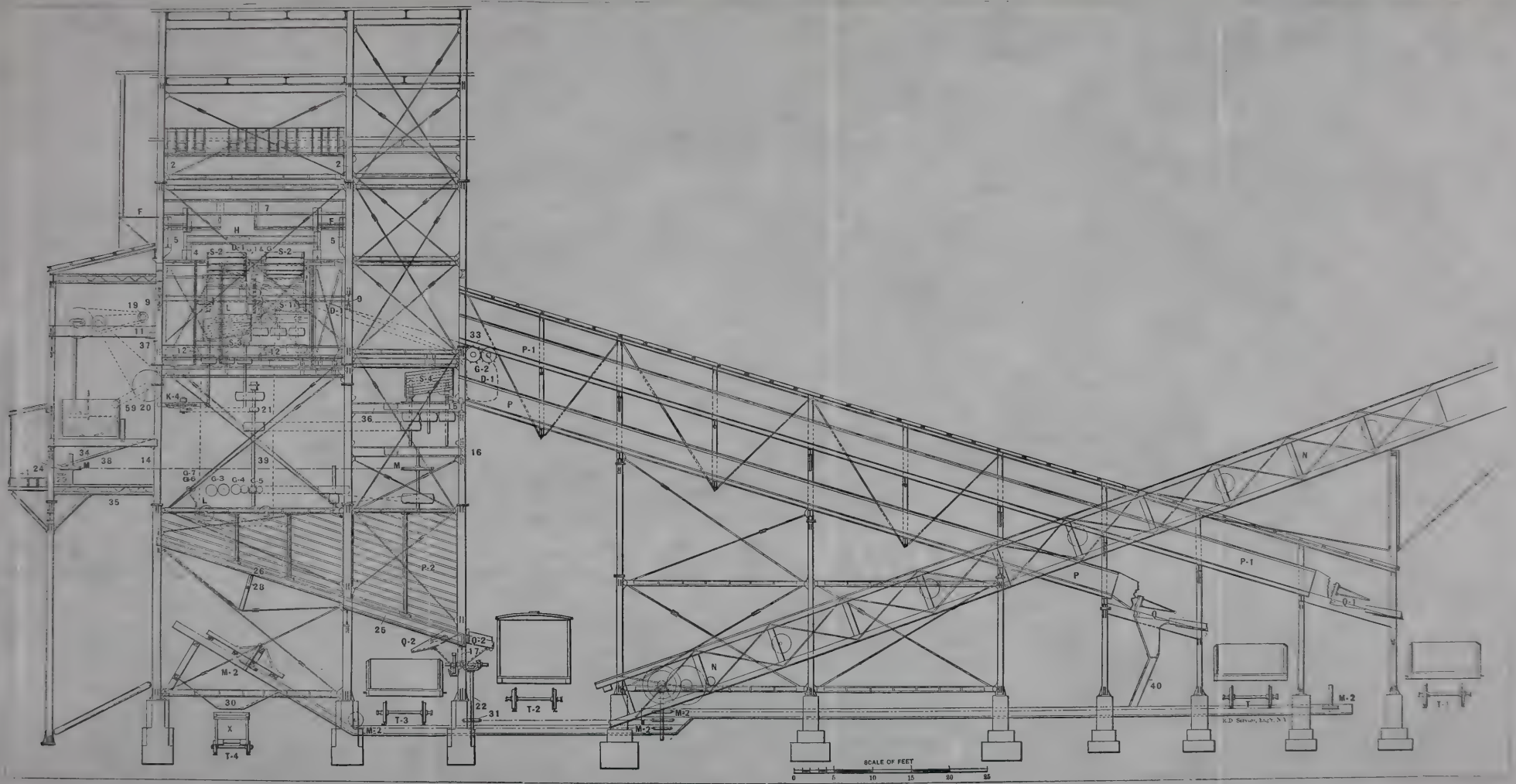
Barney-Dump.





Side Elevation of Breaker, looking West.

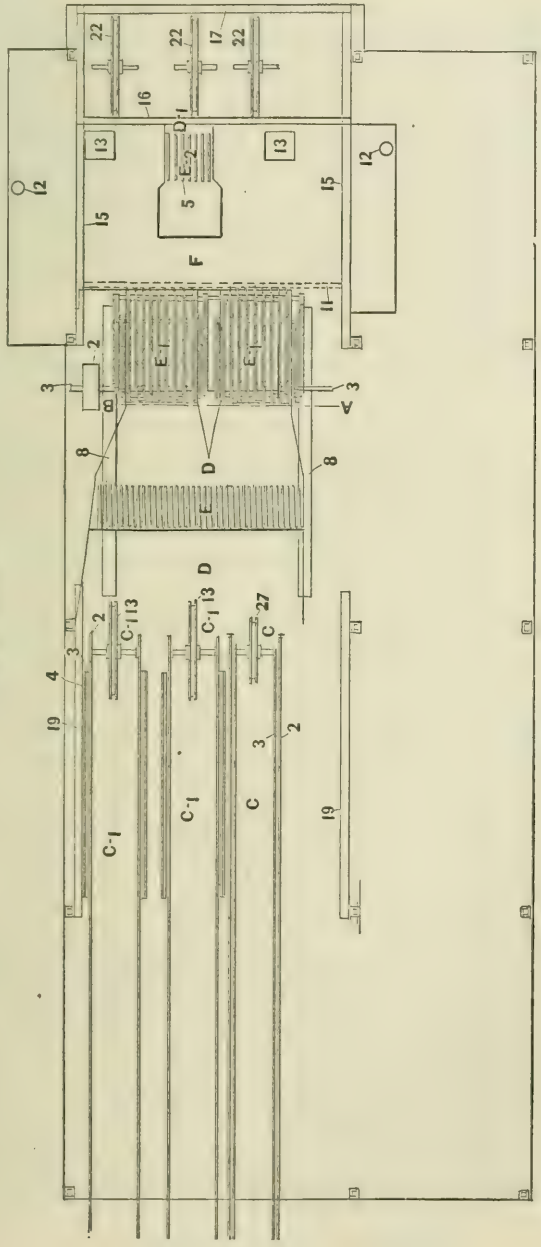

N
P-1
Q-1
M-2
T
T-1
R.D. Servoss, Engr, N.Y.



End Elevation of Breaker, Jig-House and Pockets.

rail (2) until the front wheel reaches a point (6) where the rail turns up. The hind-wheel having reached a point just in front of that occupied by the front barney-wheel in the figure, and the barney being on

PLATE XXXV.



Plan of Upper Part of Breaker.

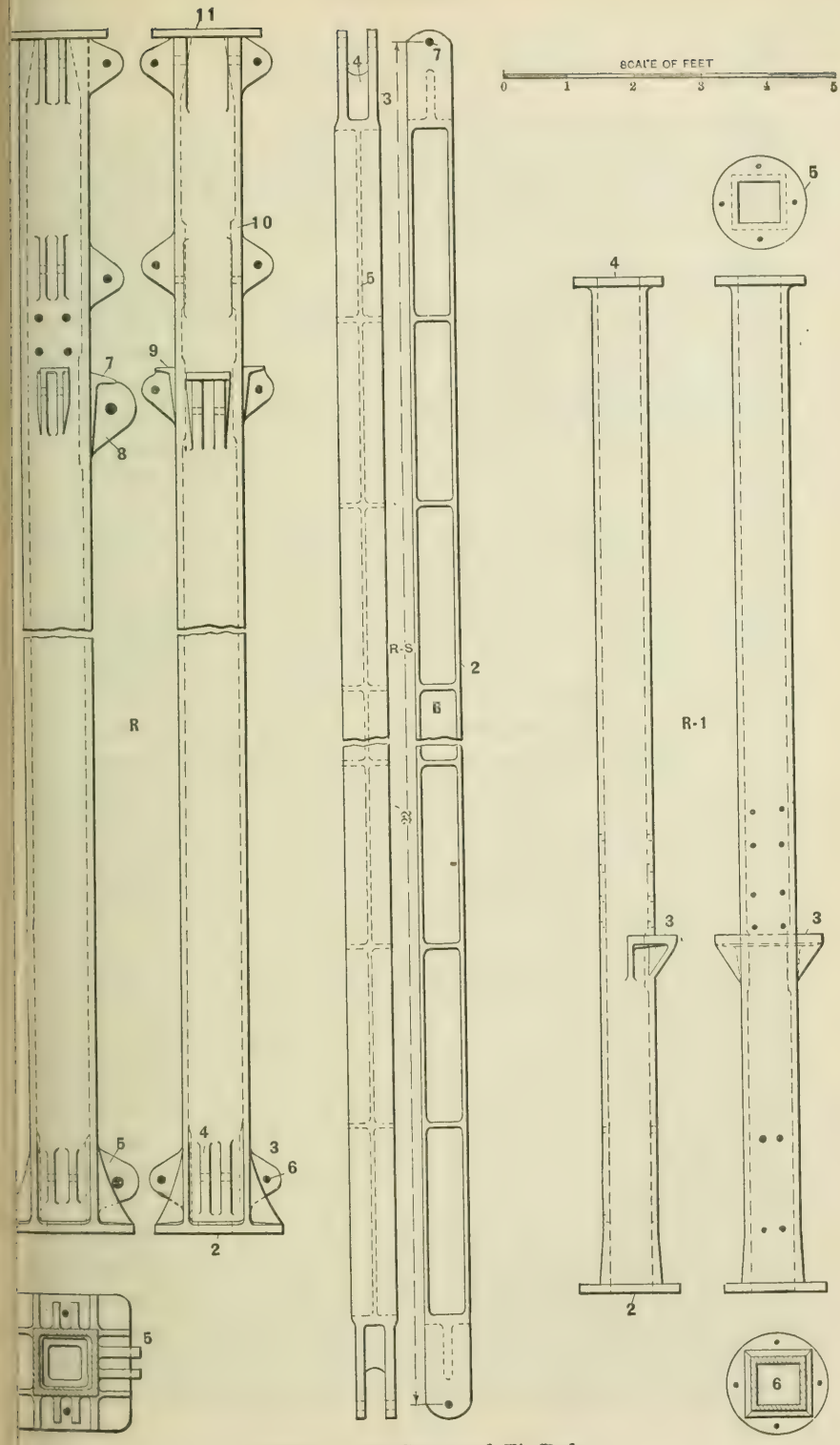
the track (3) where it is steepest, the line of thrust of the pusher is no longer in the line of the axis of the bumper, but in an upward direction. Hence, as soon as the front-wheel of the car is stopped at (6), the barney continuing to move, raises the back-end of the car until it assumes the position shown in the figure, the body of the car turning around the front axle (23). The door which closes the front-end of the car is closed on either side by the latch (10) which projects beyond the bumper (11). As the front-end of the car descends, revolving about the front-axle, the latch strikes against the iron latch-lifter which raises the latch, allowing the door to swing open and the car to empty. The latch-opener (32) is shown in Plate XXXIII. It is simply a U-shaped bar of iron, firmly fastened to the floor below on each side of the car, in such a way that only the projecting latch (10) can strike against it.

When the engine is reversed, the barney returns and the car gradually descends and takes its normal position behind the barney.

IV. The Iron Breaker.

The iron breaker is a pin-connected structure, the posts being of cast-iron, the struts generally of cast-iron, and the tie-rods of wrought-iron. Most of the large beams are riveted plate-girders; the smaller are rolled iron. There are three large lattice- or truss-girders.

The pockets for the prepared sizes are eight in number. They are $38\frac{1}{2}$ feet long, $11\frac{3}{4}$ feet wide, and $3\frac{3}{4}$ feet deep at the upper end, and $15\frac{3}{4}$ feet deep at the lower end. The bottoms of the pockets, which are made of cast-iron plates, are supported on beams (25) and (26) running parallel to the longer axis of the pocket and on a pitch of four inches to the foot. They are supported at four points by iron beams, three of which are between posts, and the fourth (28) running the whole length of the pockets supported by a truss, as shown in the end elevation, Plate XXXIV. The cast-iron plates forming the bottom of the pockets are three feet wide and six feet long, and extend from one beam to the other, the beams being three feet apart from center to center. In order to protect these beams from the action of the acid water in the coal, they are covered with a plate of lead, which extends beyond the edge of the beam and is turned down, so that all the water leaking through drops to the ground without falling upon the beams. The beams between the two lower beams, which support the heaviest load, are riveted girders, while the others above are rolled I-beams. The divisions between the



Details of Posts, Struts and Tie-Rods.

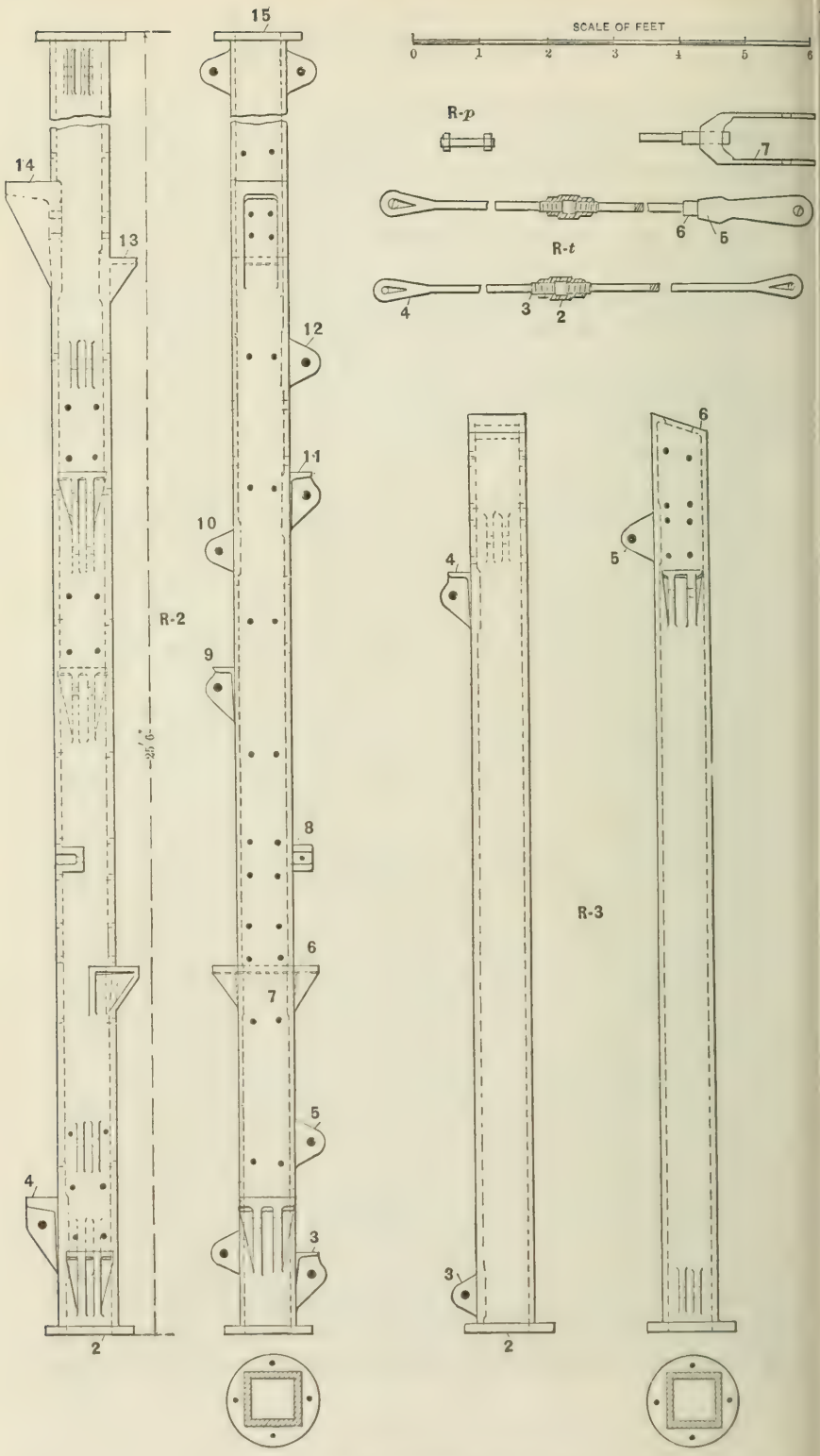
pockets are made with cast-iron plates placed between stanchions. The lower end of the pocket is also made in the same way. The upper end is simply sheet-iron, as the pressure there is very slight. All the plates are ribbed, those on which the coal slides, on one side only.

The lump-coal chute, P, consists of a pin, connected truss-structure, with two chutes, the upper one, P_1, for steamboat-coal, and the other, P, for lump-coal. They are of like construction, the sides being formed of vertical plate-girders, and the bottom of cast-iron plates, three feet long and six feet wide, supported on I-beams, which are supported on other I-beams placed between the vertical posts. Plate XXXIV. shows the general construction. It will be observed that where the posts are from 20 to 24 feet apart, there is a hanging post, which is supported by rods from the main posts, making the distance between the supports of the beams carrying the bottom of the pocket not over 12 feet. At the bottom of the lump- and steamboat-chute they are divided into two loading-chutes, Q and Q_1 (see also Plate XXIV.) in each of which there is a set of finger-bars, which takes out the fine coal and allows it to drop by means of the chute (40) into the main screenings-drag, M_2, which takes it back under the breaker, where it is loaded into mine-cars, X. Just above these bars is a gate to regulate the flow, and below the bars there is a chute, which extends out into the cars for loading. There are two types of loading-lips, one for box-cars and the other for ordinary open-top cars. The one for open-top cars turns at a right angle so as to allow coal to be fed near the bottom, and thus avoid the fall. The pitch of the lump- and steamboat-chutes is $3\frac{5}{8}$ inches. This appears to be too much, although it is less than that of any other similar chutes we had previously built. This is probably due to the fact that the cast-iron plates at the bottom are much larger and smoother than the old-fashioned ones, which are only one foot wide and four feet long. At the same time it is very doubtful whether any pitch can be adopted which will not allow the coal to slide too fast when it is by itself, and at the same time will allow it to flow freely when the chute is full and the coal lies in a body.

The plane upon which the coal is hoisted is also a pin-connected structure, as shown in Plate XXXIII., the posts, struts and tie-rods being of the same general character as those employed in the lump-chute and the rest of the building. Each track is carried on two riveted girders (23) 18 inches high, which are connected together $27\frac{1}{2}$ feet apart, by division-plates and tie-rods. Over the tops of these, wooden

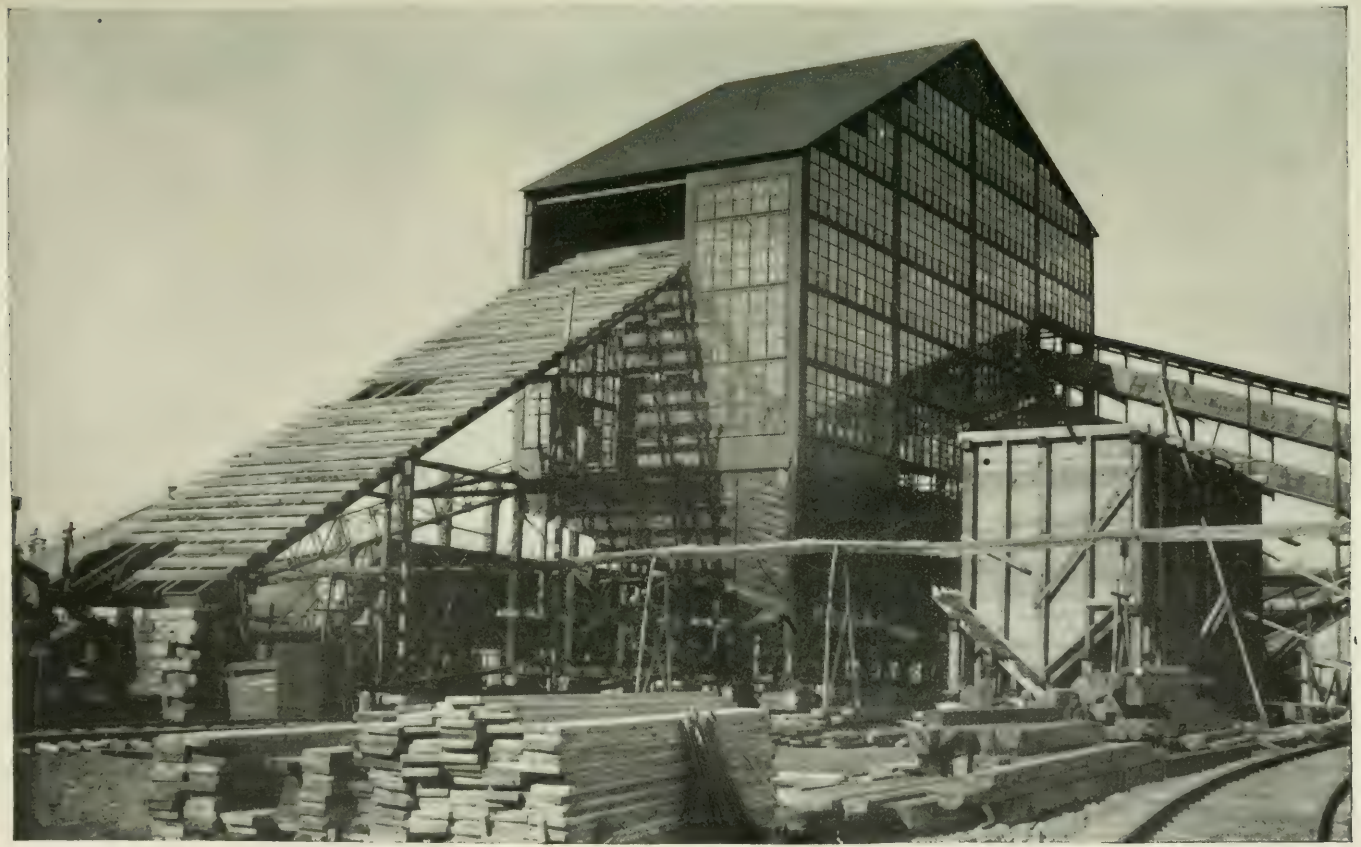
sills are laid, which are also shown on the drawing. Each gunboat track is immediately over a pair of girders. It was intended when the breaker was constructed to build the track from the surface in the same way; but it was found only necessary to put up one track instead of two, as we can hoist easily a car a minute with the single track; and the arrangements at the top and the bottom were much simplified by adopting the single track. Had the length of the plane been very much greater, two tracks would have been necessary, and in that case these tracks also would have been immediately over the girder. In other words, it was not necessary to build the hoisting-plane quite as wide as it is, and the one pair of girders could have been dispensed with.

The main structure, or that part of the breaker which contains the machinery, is built immediately over the pockets. The foundations under the whole breaker were put in with a great deal of care, particularly those under the pockets. The foundations were first dug out to a depth safely below the frost-line. About 2 feet of small broken stone was put in, laid in layers of 6 to 8 inches, and each layer was rammed with a heavy wooden rammer until it did not show any signs of sinking further. Above this broken stone walls of heavy dressed conglomerate stone were laid up, being about 5 feet thick below the surface of the ground and 4 feet above. The wall went the whole length of the pockets, but at a certain distance above the ground, piers 4 feet square were built up under each post. The same general plan was pursued for the posts of the plane and lump-coal chute, except that in this case the piers were 4 feet square, and the walls were not continuous. The end of the plane rests on a wall extending the whole width of the plane, and reinforced by wing-walls (Plate XXX.), in the direction of the slope. The ends of the plane-girders are connected by means of an expansion-joint, allowing the building to contract and expand without strain. The pockets are composed of nine rows of posts with three in each row. The two, which are in the prolongation of the line of the plane-posts, are 24 feet from center to center, and the other posts to the east of these two are 14½ feet. The rows of posts are 12 feet apart, thus making the pockets 12 feet wide, less the thickness of the partitions. All the posts are fastened to the foundation by 1½-inch bolts, built up and solidly cemented in the piers and walls. On the top of the pockets is erected the part of the building upon which the machinery is placed. The posts immediately above the pockets are 10 inches square, and those on top of these, which have nothing to do but to carry the roof, or



Details of Posts, etc., Continued.

Completed Breaker, from the southwest, showing Plane and Jig-House.



View of Breaker from the southeast, before completion, showing Settling-tank and Lump-chute.

are subjected to a comparatively light strain, are 8 inches square. The posts above the pockets, instead of being 12 feet from center to center in the direction of the axis of the plane, are 24 feet, giving a very large amount of unobstructed space for the machinery and picking-chutes. The truss-rods vary from 1 inch to 2 inches square, depending upon the strain to which they are subjected. Most of the beams that support the machinery are 15-inch rolled beams, with the exception of the riveted girders that carry the platform, the girders that support and form the dump and mud-screen pocket H, one truss-girder in the extreme northern bent, one large truss-girder immediately below the mud-screen pocket H, where it is deepest, one lattice-girder extending from those two which carry one side of the two steamboat-rolls G_2, and one side of the two gyrating screens S_4, receiving coal from these rolls, and those marked (9), (11) and (14). Outside of the line of the pockets there is an addition, supported on one side by four cast-iron posts, and on the other by the western posts of the pockets, in which addition are arranged the twelve jigs, as shown on the general plan, Plate XLI. These posts are 8 inches square, and the jigs are carried on riveted girders running parallel to the axis of the plane (Plate XXXIII.).

It has been found important not to set the gyrating screens on girders with a long span, unless they are very stiff ones, as there is a tendency to vibration in the long fifteen-inch beams. For this reason we have supported by posts these fifteen-inch beams, where they carry the gyrating screens. The swinging up and down of the beam tends to throw the running plates out of parallelism and causes great strains on the shafts. The lump-coal is carried in the chute D_1, previously described, from the platform to the lump-coal chute P, this chute being supported partly on brackets and partly by a lattice-post entirely outside of the breaker.

The sheaves which carry the hoisting-ropes are all supported, as shown, on riveted girders (8). The sides and roof of the plane and of the breaker proper are of corrugated iron throughout, except where the corrugated iron is replaced by glass (Plates XXXVIII. and XXXIX.), as is the case as far as possible around the pockets, the idea being to make the inside as light as possible. In this respect the desired result has been obtained. There are no dark corners in the breaker, particularly around the machinery, which is very accessible. All the staircases are of iron, as well as the gangways and platforms. The men who are not in charge of the machinery are not required to pass near it. The building will be heated entirely by exhaust-steam from the engines.

I give here a list of all the principal posts, struts, tie-rods, beams, etc., used in the construction of the breaker, with their dimensions. These with the general drawings and the detail drawings, Plates XXXVI. and XXXVII. of one post consisting of 4 pieces, R, R_1, R_2 and R_3, of one strut, R-s; of two tie-rods, R-t, of different kinds, and one pin, R-p, which are types of the principal members (excepting the beams), will give probably a better idea of the structure than a mere description in words.

The height of the breaker from the railroad-track, T_3, under the pockets to the peak of the roof, is 91½ feet; to the dump, 79 feet. The greatest amount of coal that has been passed through and cleaned in the breaker is a little over 260 tons per hour. The breaker has not been running long enough to determine accurately its maximum capacity. I shall endeavor to give this information as well as the capacity of the rolls, screens, jigs, etc., at a future time.

Bill of Material of the More Important Iron Members of the Breaker Structure.

Part of Breaker to which they belong.	No. of Pieces.	Name of Member.	Size.	Length.	Core.
			Inches.	Ft. In.	In.
Gunboat-plane	2	Cast columns.	8 x 8	23 0	6½
" "	2	" "	8 x 8	20 9	"
" "	2	" "	8 x 8	16 2	"
Pockets	27	" "	12 x 12	24 8	10¼
Machinery and screen-room	3	" "	10 x 10	20 3	8¼
" " "	3	" "	10 x 10	15 2	"
" " "	3	" "	10 x 10	20 2	"
" " "	3	" "	10 x 10	28 9⅞	"
" " "	3	" "	10 x 10	21 0	"
" " "	3	" "	10 x 10	20 0	"
" " "	3	" "	10 x 10	22 0	"
" " "	3	" "	10 x 10	20 0	"
" " "	3	" "	10 x 10	29 0	"
" " "	3	" "	10 x 10	13 8⅝	"
" " "	3	" "	10 x 10	15 3	"
" " "	3	" "	10 x 10	20 0	"
" " "	3	" "	10 x 10	11 8	"
Jig-house	4	" "	8 x 8	31 0	6½
"	4	" "	8 x 8	29 5⅛	"
Lump-coal chute	2	" "	8 x 3	16 5½	"
" "	2	" "	8 x 8	20 0⅝	"
" "	2	" "	8 x 8	23 11¾	"
" "	2	" "	8 x 8	11 8⅛	"
" "	2	" "	8 x 8	15 10½	"
" "	2	" "	8 x 8	15 8	"
" "	2	" "	8 x 8	15 10⅜	"
" "	2	" "	8 x 8	22 11	"
" "	2	" "	8 x 8	15 10⅜	"
" "	2	" "	8 x 8	15 8	"
" "	2	" "	8 x 8	14 6	"
" "	2	" "	8 x 8	15 10⅝	"

LIST OF MATERIAL—*Continued.*

Part of Breaker to which they belong.	No. of Pieces.	Name of Member.	Size.	Length.
			Inches.	Ft. In.
Gunboat plane	9	Cast struts	8 x 8	22 10
" "	2	"	8 x 8	23 $0\frac{3}{8}$
" "	2	"	8 x 8	26 8
Pockets	9	"	10 x 8	22 6
"	9	"	8 x 8	22 6
"	9	"	6 x 6	13 0
"	24	"	6 x 6	10 6
Machinery and screen room	8	"	6 x 6	10 6
" " " "	23	"	8 x 8	22 8
" " " "	1	"	8 x 10	22 8
" " " "	1	"	8 x 8	9 8
Jig room	4	"	8 x 8	12 6
"	4	"	8 x 8	14 $0\frac{5}{8}$
Lump-coal chute	8	"	8 x 8	22 10
" "	19	"	6 x 6	11 $7\frac{3}{4}$
" "	2	"	8 x 8	18 8
Pockets	24	Cast stanchion	10 x 6	15 $1\frac{1}{2}$
"	9	"	10 x 6	13 0
"	9	"	10 x 6	8 3
"	9	"	10 x 6	6 3
Lump-coal chute	6	"	5 x 6	16 $1\frac{3}{8}$
" "	1	Rolled beam	$10\frac{1}{2}$ x $4\frac{7}{8}$	12 $1\frac{3}{4}$
" "	16	" "	9 x $4\frac{1}{8}$	12 $1\frac{1}{2}$
" "	6	" "	6 x $3\frac{3}{4}$	20 $10\frac{3}{4}$
" "	12	" "	6 x $3\frac{5}{8}$	25 $0\frac{5}{8}$
" "	3	" "	6 x $3\frac{3}{8}$	26 $1\frac{7}{16}$
" "	6	" "	6 x $3\frac{3}{8}$	14 3
" "	2	" channels	6 x $2\frac{1}{2}$	21 $4\frac{13}{16}$
" "	4	" "	6 x $2\frac{1}{2}$	25 $2\frac{11}{16}$
" "	2	" "	6 x $2\frac{1}{2}$	26 $3\frac{11}{16}$
" "	2	" "	6 x $2\frac{1}{2}$	26 $7\frac{7}{8}$
" "	4	" "	6 x $2\frac{1}{2}$	22 0
" "	8	" "	6 x $2\frac{1}{2}$	25 $2\frac{9}{16}$
" "	2	" "	6 x $2\frac{1}{2}$	26 $3\frac{1}{8}$
" "	4	" "	6 x $2\frac{1}{2}$	14 3
" "	4	Rolled plates	39 x $\frac{3}{8}$	22 $10\frac{3}{8}$
" "	8	" "	39 x $\frac{3}{8}$	25 $0\frac{1}{4}$
" "	2	" "	39 x $\frac{3}{8}$	26 $1\frac{7}{16}$
" "	4	" "	39 x $\frac{3}{8}$	13 7
" "	4	Rolled angles	$2\frac{1}{2}$ x 3 x $\frac{3}{8}$	22 11
" "	8	" "	$2\frac{1}{2}$ x 3 x $\frac{3}{8}$	25 $0\frac{7}{8}$
" "	2	" "	$2\frac{1}{2}$ x 3 x $\frac{3}{8}$	26 $1\frac{7}{16}$
" "	4	" "	$2\frac{1}{2}$ x 3 x $\frac{3}{8}$	13 7
		Tie-rods:		
Gunboat plane	4	1 in. square upset to	$1\frac{1}{2}$	30 $10\frac{7}{8}$
" "	2	" "	$1\frac{1}{2}$	34 $6\frac{1}{2}$
" "	2	" "	$1\frac{1}{2}$	28 $11\frac{13}{16}$
" "	2	" "	$1\frac{1}{2}$	36 0
" "	2	" "	$1\frac{1}{2}$	24 $3\frac{9}{16}$
" "	4	" "	$1\frac{1}{2}$	29 $11\frac{3}{8}$
" "	2	" "	$1\frac{1}{2}$	26 6
" "	8	Built beams	18 x $5\frac{1}{2}$	29 3
" "	24	"	18 x $5\frac{1}{2}$	28 8
" "	4	"	29 x $10\frac{1}{2}$	23 4
" "	20	"	$46\frac{3}{4}$ x $5\frac{1}{2}$	3 $10\frac{3}{4}$
" "	8	Rolled beams	9 x $3\frac{1}{2}$	14 8
Pockets	8	Built beams	24 x $6\frac{1}{2}$	11 0

LIST OF MATERIAL—*Continued.*

Part of Breaker to which they belong.	No. of Pieces.	Name of Member.	Size.	Length.
			Inches.	Ft. In.
Pockets	8	Built beams	24 x 8½	11 0
"	9	" "	15 x 8½	14 6¾
"	24	" "	15 x 6½	16 3
"	9	" "	9 x 8½	24 4⅛
"	6	" "	24 x 6½	12 0
"	2	" "	24 x 6½	12 4¼
"	8	Rolled beams	12 x 4⅝	11 0
"	24	" "	9 x 4	25 0¼
Machinery and screen-room	2	Built beams	30 x 8½	27 5 3/16
" " "	1	" "	15 x 8½	23 10⅞
" " "	2	" "	16 x 8½	23 1⅞
" " "	2	" "	15 x 8½	34 4½
" " "	1	" "	16 x 8½	23 1⅞
" " "	1	" "	28 x 10½	22 1¼
" " "	1	" "	24 x 12½	23 3
" " "	1	" "	46½ x 10½	23 2
" " "	2	" "	46 x 8½	28 7¼
" " "	2	" "	63 x 10½	27 8½
" " "	1	" "	24 x 10½	23 9
" " "	3	Rolled beams	12 x 4⅝	17 8
" " "	1	" "	12 x 4⅝	16 5
" " "	8	" "	12 x 4⅝	16 9
" " "	4	" "	9 x 4¼	27 6 5/32
" " "	4	" "	9 x 4¼	25 1 1/16
" " "	4	" "	9 x 4¼	27 4½
" " "	4	" "	9 x 4¼	24 11½
" " "	5	" "	9 x 4¼	13 1
" " "	5	" "	15 x 5⅛	23 1¾
" " "	1	" "	9 x 4¼	24 0
" " "	1	" "	9 x 4¼	14 2
" " "	1	" "	9 x 4¼	13 4 3/32
" " "	1	Built beams	12 x 8½	24 4 3/32
" " "	3	" "	15 x 8⅛	23 1⅛
" " "	2	Rolled beams	15 x 5⅛	23 11 1/16
" " "	1	" "	15 x 5⅛	23 1 1/16
" " "	1	" "	20 x 6¼	23 1 3/32
" " "	4	" "	15 x 5⅛	13 7 3/32
" " "	2	" channels	9 x 2½	28 2
" " "	2	" "	9 x 2½	27 4½
" " "	2	" "	9 x 2½	25 1 1/16
" " "	2	" "	9 x 2½	25 11
" " "	1	Truss	77½ x 10½	23 1½
" " "	1	"	158 x 10½	23 2
" " "	1	"	172½ x 10½	23 2
Jig-room	3	Built beams	36 x 8½	23 4
"	2	" "	24 x 8½	25 7
"	1	" "	24 x 8½	24 0
"	2	" "	24 x 8½	25 5
"	8	Lattice beams	12 x 6½	13 1
"	3	" "	12 x 6½	23 4
"	3	Rolled beams	12 x 4⅝	23 4
"	2	" "	12 x 5½	24 5
"	1	" "	12 x 5½	24 0
"	11	" "	6 x 3½	14 6
"	4	Built beams	6 x 8½	5 2
"	4	" "	24 x 8½	24 5
		Tie rods:		
Pockets	9	2⅛ in. square upset to	3	24 10½

LIST OF MATERIAL—*Continued.*

Part of Breaker to which they belong.	No. of Pieces.	Name of Member.	Size.	Length.
		TIE RODS:	Inches.	Ft. In.
Pockets	9	$2\frac{1}{2}$ in. square upset to	3	28 $9\frac{3}{4}$
"	18	$1\frac{5}{8}$ " "	$2\frac{3}{8}$	24 10
"	18	1 " "	$1\frac{1}{2}$	22 $8\frac{3}{4}$
"	18	1 " "	"	24 $9\frac{3}{4}$
"	18	1 " "	"	19 $9\frac{1}{2}$
"	18	1 " "	"	16 $10\frac{1}{2}$
"	16	1 " "	"	11 2
"	16	1 " "	"	18 3
"	16	1 " "	"	13 5
"	16	1 " "	"	14 $5\frac{3}{4}$
"	16	1 " "	"	21 $2\frac{1}{2}$
"	16	1 " "	"	10 11
Machinery and screen-room	6	1 " "	"	26 $5\frac{1}{2}$
" " "	6	1 " "	"	30 $8\frac{3}{4}$
" " "	6	1 " "	"	25 $10\frac{15}{16}$
" " "	4	1 " "	"	26 $2\frac{15}{16}$
" " "	4	1 " "	"	24 $2\frac{1}{4}$
" " "	2	1 " "	"	30 $8\frac{3}{4}$
" " "	2	1 " "	"	24 $10\frac{3}{4}$
" " "	2	1 " "	"	24 $7\frac{7}{8}$
" " "	2	1 " "	"	31 $2\frac{1}{8}$
" " "	2	1 " "	"	12 $4\frac{5}{8}$
" " "	2	1 " "	"	22 $11\frac{1}{4}$
" " "	14	1 " "	"	30 $1\frac{15}{16}$
" " "	4	1 " "	"	28 $3\frac{1}{4}$
" " "	2	1 " "	"	28 $0\frac{3}{8}$
" " "	2	1 " "	"	18 $5\frac{1}{8}$
" " "	2	1 " "	"	25 $3\frac{3}{16}$
" " "	2	1 " "	"	23 $5\frac{1}{16}$
" " "	2	$1\frac{1}{8}$ " "	$1\frac{7}{8}$	18 $4\frac{9}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	27 $2\frac{7}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	17 $7\frac{1}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	19 $11\frac{5}{8}$
" " "	2	$1\frac{1}{4}$ " "	"	30 $1\frac{3}{4}$
" " "	2	$1\frac{1}{4}$ " "	"	23 $9\frac{9}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	24 9
" " "	2	$1\frac{1}{4}$ " "	"	18 $10\frac{3}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	13 $10\frac{11}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	13 $7\frac{7}{8}$
" " "	2	$1\frac{1}{4}$ " "	$2\frac{1}{4}$	17 $0\frac{5}{8}$
" " "	2	$1\frac{1}{4}$ " "	"	17 $1\frac{1}{2}$
" " "	2	$1\frac{1}{4}$ " "	"	18 $4\frac{3}{16}$
" " "	2	$1\frac{1}{4}$ " "	$1\frac{7}{8}$	30 $10\frac{1}{2}$
" " "	2	$1\frac{1}{4}$ " "	"	23 $0\frac{15}{16}$
" " "	2	$1\frac{1}{4}$ " "	"	14 $5\frac{3}{4}$
" " "	4	1 " "	"	30 $1\frac{15}{16}$
" " "	2	$1\frac{1}{2}$ " "	$2\frac{1}{4}$	23 $9\frac{15}{16}$
" " "	2	$1\frac{1}{2}$ " "	"	29 4
" " "	2	$1\frac{1}{2}$ " "	"	23 $7\frac{7}{8}$
" " "	2	$1\frac{1}{2}$ " "	"	28 $6\frac{1}{4}$
" " "	2	$1\frac{3}{4}$ " "	$2\frac{1}{2}$	25 $0\frac{1}{16}$
" " "	2	1 " "	$1\frac{1}{2}$	16 $11\frac{1}{4}$
" " "	2	1 " "	"	27 9
" " "	8	$1\frac{1}{2}$ " "	$2\frac{1}{4}$	27 9
Lump-chute	2	1 " "	$1\frac{1}{2}$	27 $0\frac{1}{2}$
"	2	1 " "	"	27 $8\frac{1}{4}$
"	2	1 " "	"	24 4
"	2	1 " "	"	15 $8\frac{3}{4}$
"	2	1 " "	"	16 $2\frac{3}{4}$
"	18	1 " "	"	13 $8\frac{1}{2}$
"	4	1 " "	"	13 $1\frac{1}{2}$

I give here two views from photographs of the breaker: Plate XXXVIII. being taken from a point southwest of the breaker and representing the breaker when finished. It shows the barney and gunboat on the plane; and on the left-hand side of the breaker, the jig-house, which projects from the main line of the building, is very distinctly shown. The truss-bridge running from the ground to the top of pockets (about 26 feet from the ground) is the main entrance to the breaker for the workmen. The advantage of this plane is that it brings the men to the ground beyond the cars and machinery, thus diminishing the risk of accidents. Plate XXXIX. is a view taken from a point southeast of the breaker. This was taken when the breaker was only partially completed. It shows on the right hand the lump- and steamboat-chutes very distinctly, and the plane in its unfinished condition with the girders carrying the sills. The large wooden structure to the right is the settling-tank for settling the slimes coming from the breaker. The water, after depositing in them the most of the fine coal and slimes, is carried off in pipes. It will be noticed in both these plates, particularly in Plate XXXIX., that, in consequence of the great number of windows all around the breaker, one can see through it distinctly. The thoroughness with which the interior is lighted will therefore be evident. The breaker is also equipped with an incandescent electric light plant of 200 lamps of 16 candle-power.

V. The Preparation of Coal in the Iron Breaker at Drifton.

Plates II., III., IV. and XXXIII. to XLIII. illustrate this description. The coal arrives at the dump, C or C_1, Plates XXIX. and XXXII., either in gunboats or in mine-cars. (The gunboat and barney-dump have been described above.) It is emptied into the dump-chute, which is a reservoir for the breaker; passes first over the fixed finger-bars, E, which allow most of the smaller coal to fall into the mud-screen pocket, H, while the larger accumulates in the dump-chute, D, above the oscillating bars, E_1. The gate, the position of which is shown in the plan, Plates III. and XXXV., by the line A B above the bars, (30) to (32), Plate II., serves to regulate the flow of the coal upon the bars, particularly when the car or gunboat is dumped.

The oscillating bars carry within certain limits about the same quantity of coal per minute, as long as the speed of the shaft which drives them is the same, and they thus aid as a regulator in feeding

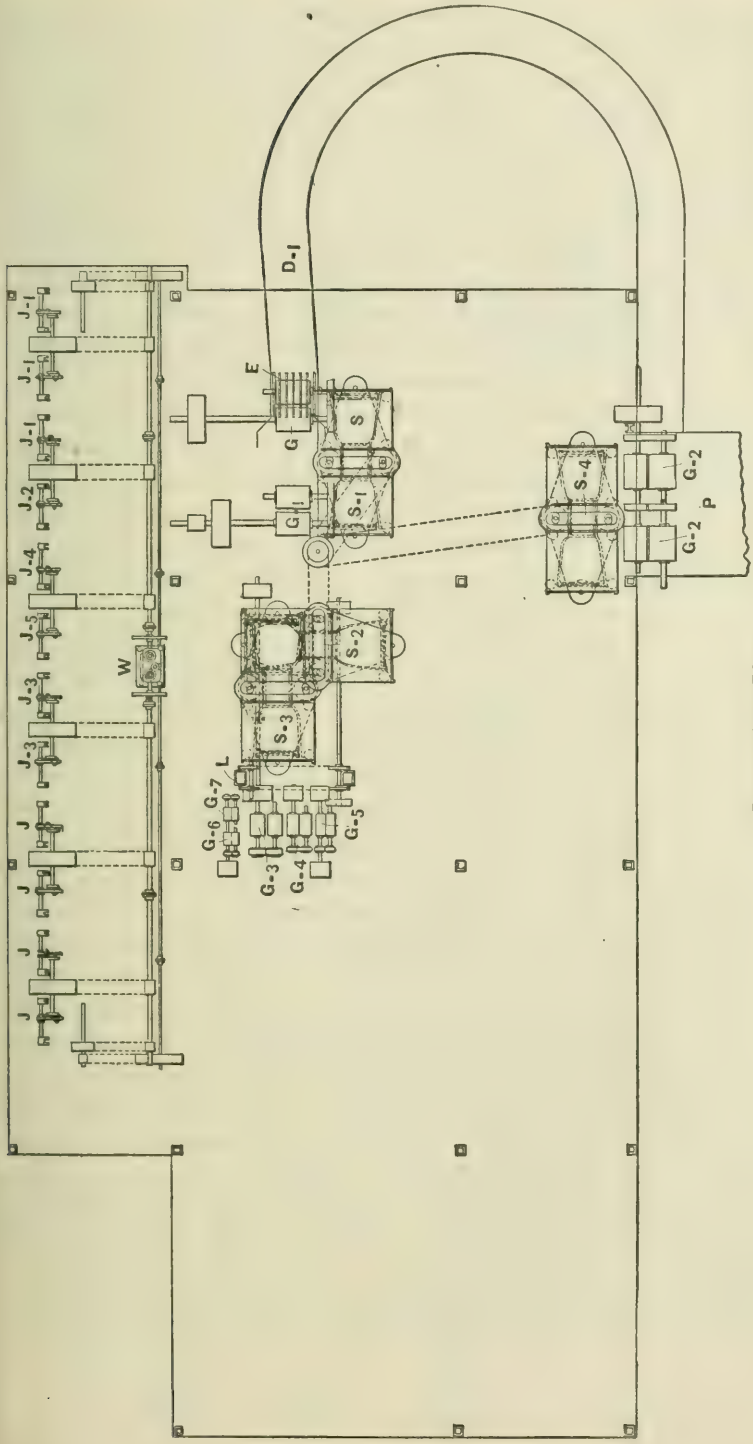
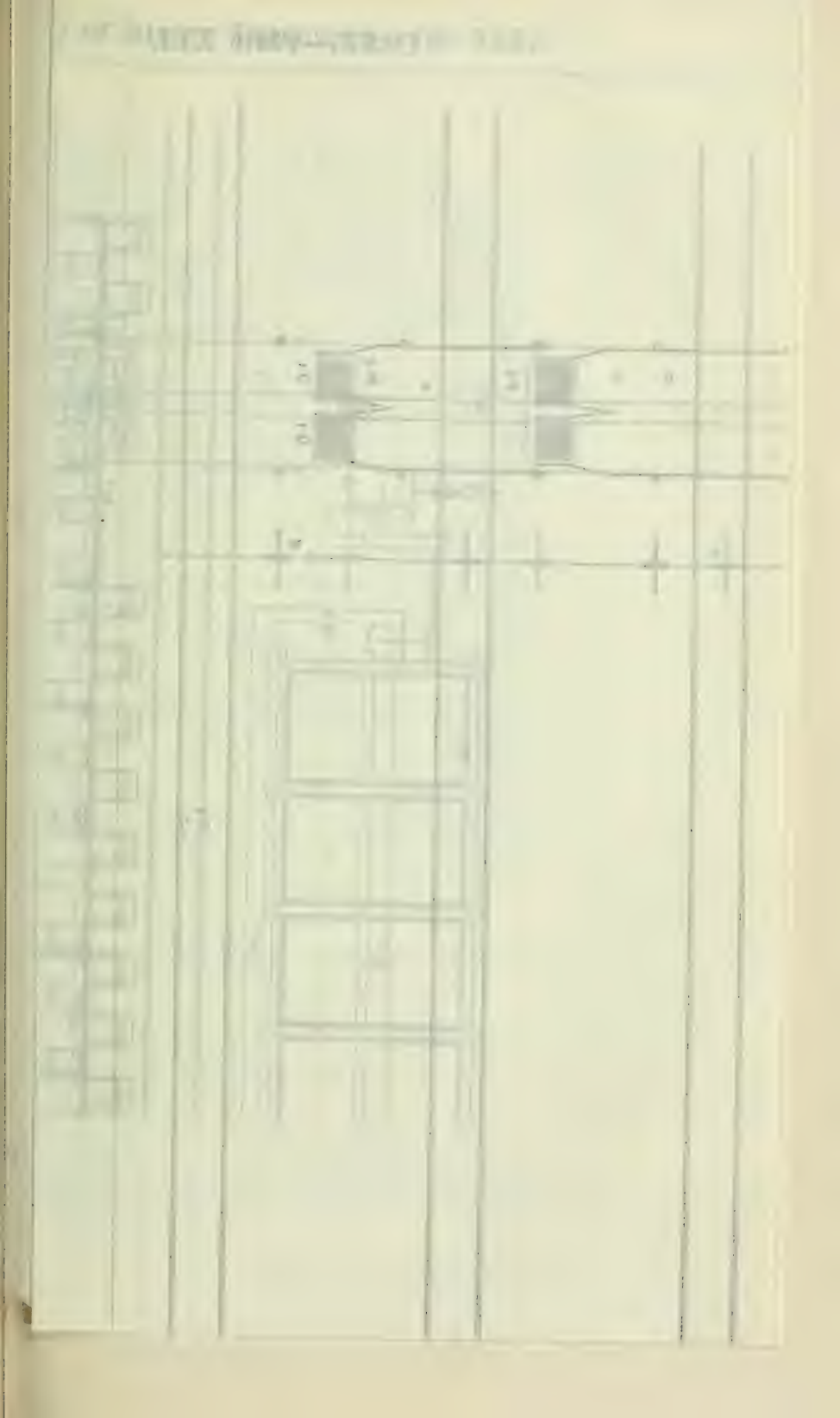
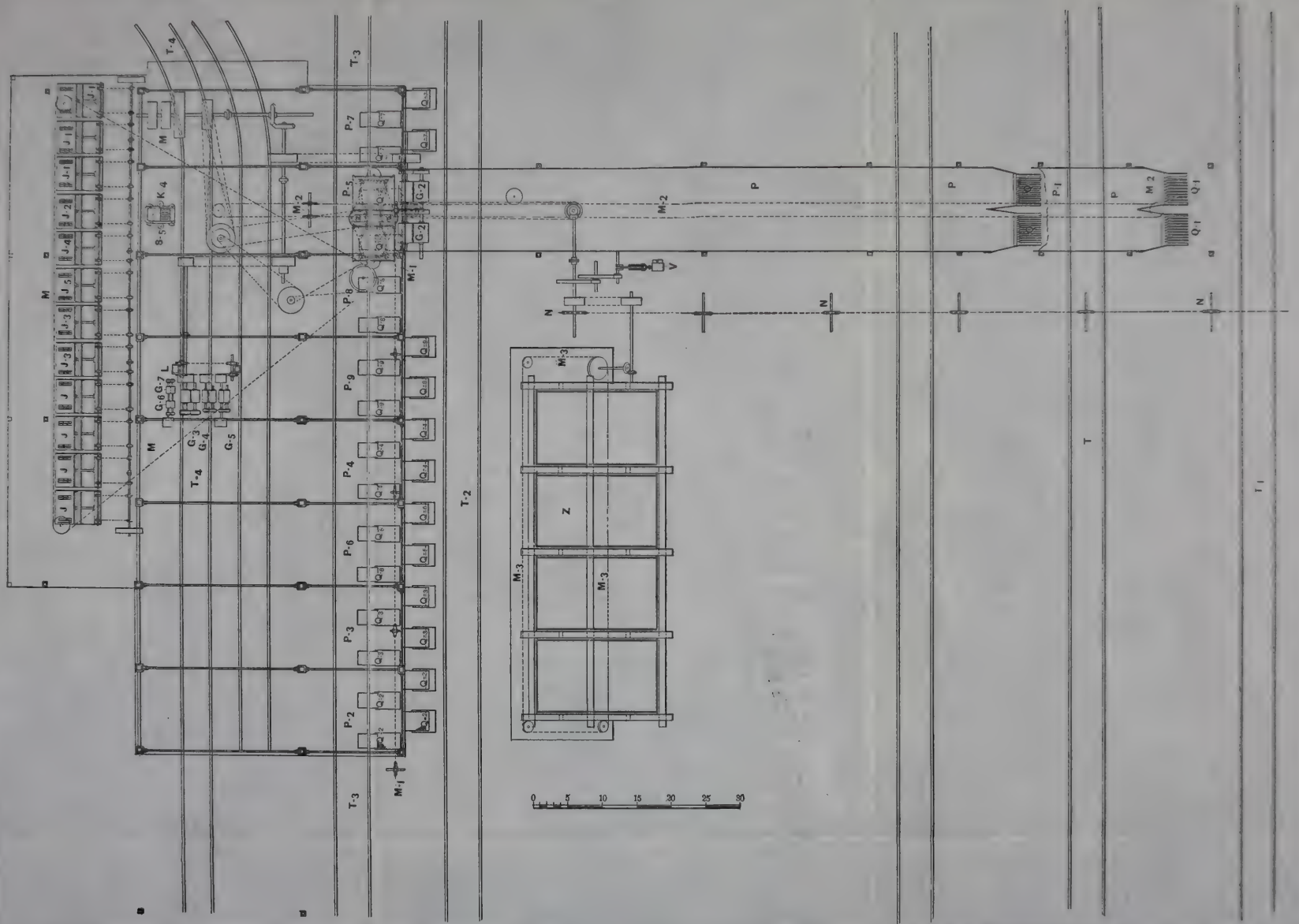


PLATE XL.

Intermediate Plan.

the breaker. Practically, all the coal smaller than lump-coal passes through these bars, and only the lump-coal arrives at the platform, F. Of course there is a certain amount of small coal and dirt carried by these large lumps which did not happen to turn over. All this fine stuff, together with the fine stuff made on the platform, is pushed either back into the mud-screen pocket or into the screen under the slate-coal rolls. The lump-coal that arrives on the platform, F, is divided into three sorts: the first being the slate, which is put into the slate-chutes (12), Plates II., IV., and XXXV., and is carried to the drag, N, Plate XXXIV.; and the second, the pure coal, which is put into the lump-coal chute, D_1, and, if there is market for it, is loaded into cars and sold as lump-coal. At the head of the chute, D_1, leading to the lump-chute there is a set of bars, E_2, the distance between which can be varied when any lump-coal is to be broken up. The bars are kept four or five inches apart. If lump-coal is to be broken up, the opening between the bars is made larger, so that sufficient coal passes between the bars to reduce the quantity of lump-coal to what can be sold. The third product consists of those lumps which I have called slate-coal, containing too much slate to go to market in their existing condition. These lumps are slated by hand; that is, if the slate can be easily detached from them with a pick, it is split off and the slate goes to the slate-chute, (12), while the remaining coal, if pure, goes down the lump-chute, D_1. However, there is always a large amount of coal which cannot be cleaned in this way. This is put through a set of rolls, G_1, the product from which goes into a gyrating screen, S_1, while the pure coal from the pure rolls, G, goes into another gyrating screen, S, which runs alongside of the slate-coal gyrating screen, S_1. All the coal that has gone through the stationary bars, E, and oscillating bars, E_1, above the platform, F, is accumulated in the mud-screen pocket, H, or reservoir. From this the coal is conveyed to the two mud-screens, S_2, the feeding being regulated by means of gates at the mouth of the pocket, H. There are two mud-screens, S_2, running together, as explained above, each of which makes steamboat, broken and egg. The stove and smaller coal passes to another pair of screens, known as the stove or wet screens, S_3, which are situated a little lower down. The steamboat-coal from the two mud-screens passes into a chute, U, upon one side of which are a number of small troughs through which the coal passes; and during the passage through these small troughs the slate and slate-coal are taken out, as are also any pieces of flat or unmarketable shape. The coal passed through these



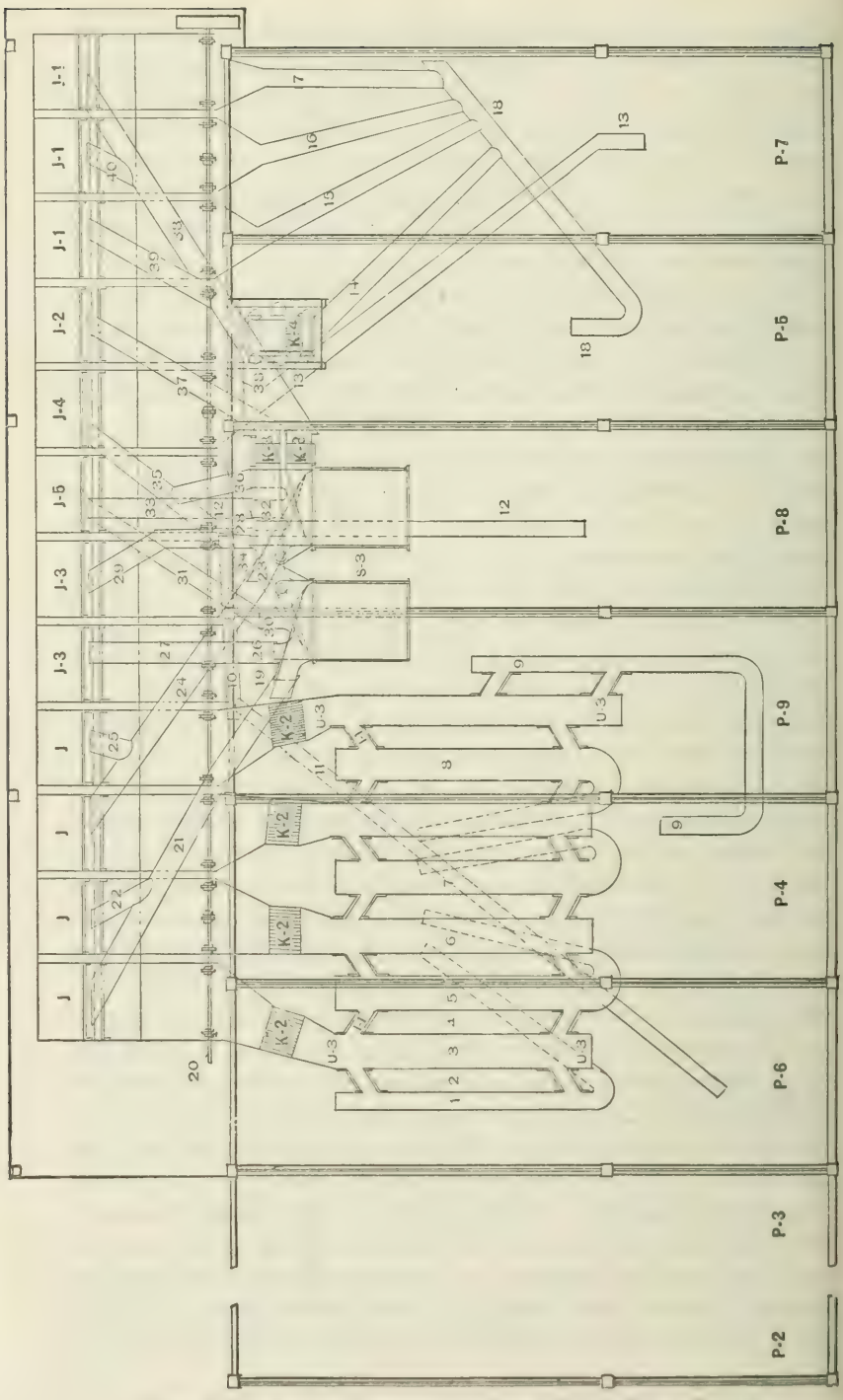


Ground Plan of Breaker.

chutes enters into another chute, directly in the line of the steamboat-chute, P_1, Plate XXXIV, and slides down into the steamboat-chute, provided all the steamboat-coal can be sold. If it cannot, a portion is turned into the rolls, G_2, that lie at the head of the steamboat-chute, and goes from there into the dry broken-coal screens, S_4, which separate the coal into broken, egg, stove, chestnut, pea, buckwheat No. 1, No. 2, No. 3 (which go to the pockets direct) and dust, which runs into the slate-drag M.

The pure coal from the platform, F, which has been broken, passes from the rolls, G, into a gyrating screen, S, as has been explained before, which screen makes steamboat, broken, egg, stove and chestnut. These go directly to the pockets, P_1 to P_5, with the exception of steamboat coal which passes down into the steamboat-chute, P_1, and can also be turned into the rolls, G_2, that lie at the head of this chute. The remaining sizes, from pea to dust, go to the broken-coal screen, S_4, or, if this is not running, to the elevator, L, which carries them to the wet-screen, S_3. The slate-coal which has been broken under the platform, passes into another gyrating screen, S_1, which makes steamboat, broken and egg. What is smaller than egg goes into the wet-screen, S_3, which is under the mud-screen, S_2, and which gets all the stove and smaller coals from the mud-screen. The steamboat-coal from the slate-coal screen, S_1, passes through two chutes, in which the slate and doubtful coal are picked out, and then into the steamboat-chute, P_1, but can be turned on its passage if necessary into the rolls, G_2, as can be also the other steamboat-coal. The broken and the egg coal pass directly to the pockets, P_2 and P_3, but have their slate and slate-coal picked out on the way and thrown into their respective chutes. The broken coal from the mud-screen goes to fixed (automatic) slate-pickers, K, Plate XXI., that which passes over going into two long picking-chutes, U_1, Plates XVIII. and XIX., where the slate and slate-coal are picked out and thrown into the chutes, a little above and parallel to the other, while that which passes through goes to a picking-table where it is separated into slate-coal and slate, the former going to a pair of rolls, G_3, and the latter into the slate-chute. The same thing takes place with the egg-coal from the two mud-screens, except that there is but one chute for the egg-coal. All the coal that passes through the steamboat-rolls, G_2, at the head of the steamboat-chute, as well as that below chestnut-size from the pure broken-screen, passes into a pair of screens, S_4, which make broken, egg, stove, chestnut, pea, buckwheat, No. 1, 2 and 3 and dust; this coal having been made by breaking up

PLATE XLII.



Plan of Chutes from Wet-screens to Jigs, and from Jigs to Pockets.

steamboat-coal, which has already been cleaned, does not require any further preparation. (See diagram, Plate XLIII.) It is simply sized and goes by telegraphs to the pockets.

We have now disposed of all except the coal that goes to the screens, S_3, under the mud-screen. This pair of screens makes stove, chestnut, pea and No. 1, 2 and 3 buckwheat and slime. They are worked wet, *i.e.*, a large amount of water is put on them, as the coal which comes from the mud-screen is very often wet, containing mud and other impurities, and in order to make a good separation, it is necessary to wash it. In addition, all the wet coal from this screen is jigged after it comes from the screen. One of these screens has on one side one more jacket than on the other, the jacket being raised above the level of the top of the adjoining screen. This elevated jacket holds back egg-coal and is put on because in breaking up the larger sizes of slate-coal, such as steamboat, broken and egg, we must make a certain amount of egg. If we undertake to break these larger sizes directly into stove, the waste is too great. The egg-coal made on this screen does not go to a jig, but passes down a separate chute, U_2, and out of it the slate and slate-coal is picked, the slate-coal going back to the rolls. The stove-coal from the two screens, S_3, passes down the chutes (21) and (24) to four jigs, J, Plate XLII., the construction of which has already been explained. The chestnut-coal from each of the screens, passes from the discharge-lips over the attached automatic slate-pickers, K_3, which also partakes of the gyrating motion of the jackets. This separates the chestnut-coal into flat coal and slate (a large proportion being slate) and into round or cubical pieces of coal. The flat coal and slate pass down the chute (37) into one jig, J_2 (flat jig), and the rest of the chestnut-coal goes down the chute (38) into three other jigs, J_1. The pea-coal goes direct to two jigs, J_3, the buckwheat to another jig, J_4, and the No. 2 buckwheat to a third jig, J_5. The No. 3 buckwheat goes to a jig of the second type. When the stove-coal comes out of the jigs, J, it passes over a fixed slate-picker, K_2, which takes out the flat coal and the slate, which goes to a picking-table. A boy separates the slate from the slate-coal, the latter passing through a pair of rolls, G_6 (chestnut-rolls). The chestnut-coal which comes from the jig that gets the flat stuff, passes first over a small gyrating separator, K_4, Plate XXII., with holes one-half inch in diameter, the construction of which has already been explained. This allows the pea-coal, if there is any, to fall through while the chestnut passes over a gyrating slate-picker with a smaller slit than the one in S_3. All the coal not

allowed to go out by this smaller slate-picker, goes direct to the pockets, while the flat stuff which is let out goes to a picking-table. The slate, if any remains, is picked out from this, and the flat pieces are allowed to run through a pair of flat chestnut-rolls, G_7. The coal that goes over the slate-picker, on its way to the pocket, has any slate that may still remain in it picked out by hand. All the other smaller sizes go direct to the pockets after leaving the jigs.

All the slate and slate-coal thrown out from the steamboat-coal picking-chutes goes to a table, where any piece that contains enough coal to make it worth while is picked out, broken in two with a hammer and sent to the large broken-coal rolls, G_3, at the foot of the elevator, L. The same takes place with the refuse taken out of the broken and egg-coal, both from the mud-screen and from the upper slate-coal screen. These slate-coal re-breaking rolls, which are all placed together, discharge the material which is passed through them into an elevator, L, Plate XL., which elevates and discharges it into the south wet stove-coal screen, where it mixes with the coal from the mud-screen, is then screened and goes (with the exception of egg-coal) to the jigs. All the slate from the various picking-chutes is carried by iron chutes into the conveyor, N, which carries it out and deposits it on the dirt-bank. The coal from the jigs, as already described and shown in Plate XX., is taken out in front by the conveyor-chain and goes to the picking-chute. The slate comes out through the hopper (9) in front and below, and passes to the back-end of the jig, where it is picked over, the slate, if any, being thrown into the drag-trough (29), which runs under the jigs. The coal is carried by chutes into the pockets. When the bottoms of the jigs are cleaned, the sediment which settles goes also into the drag. The drag is provided with openings in two places, by which the water that comes out of the jigs is taken away and carried to the settling-tank, Z, while the slate and dirt, which has been more or less dried, goes direct to the conveyor, N, and is thus carried out to the dirt-bank.

Plate XL. is a plan showing the principal machinery used in the preparation, with the exception of the jig-tanks and the oscillating bars, E_1. The upper side shows the shafting and driving-apparatus of the jigs. G and G_1 are the large rolls; S and S_1 are the screens which take the coal from them; G_2 are the re-breaking rolls for steamboat, and S_4 the screen which handles all the coal broken by these rolls. S_2 is the mud screen, and S_3 is the wet-screen immediately below it, which takes all the coal from it below egg, and also

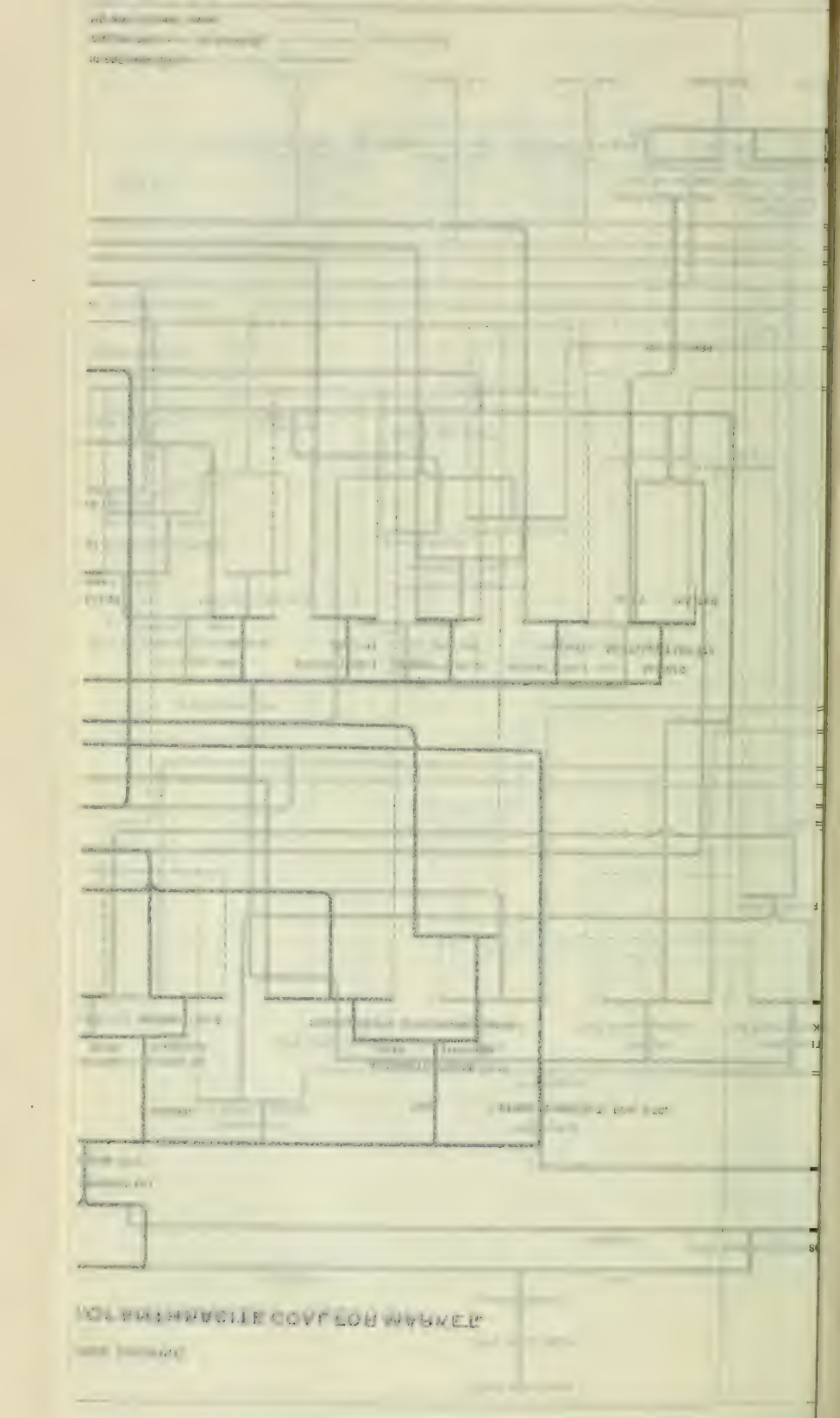
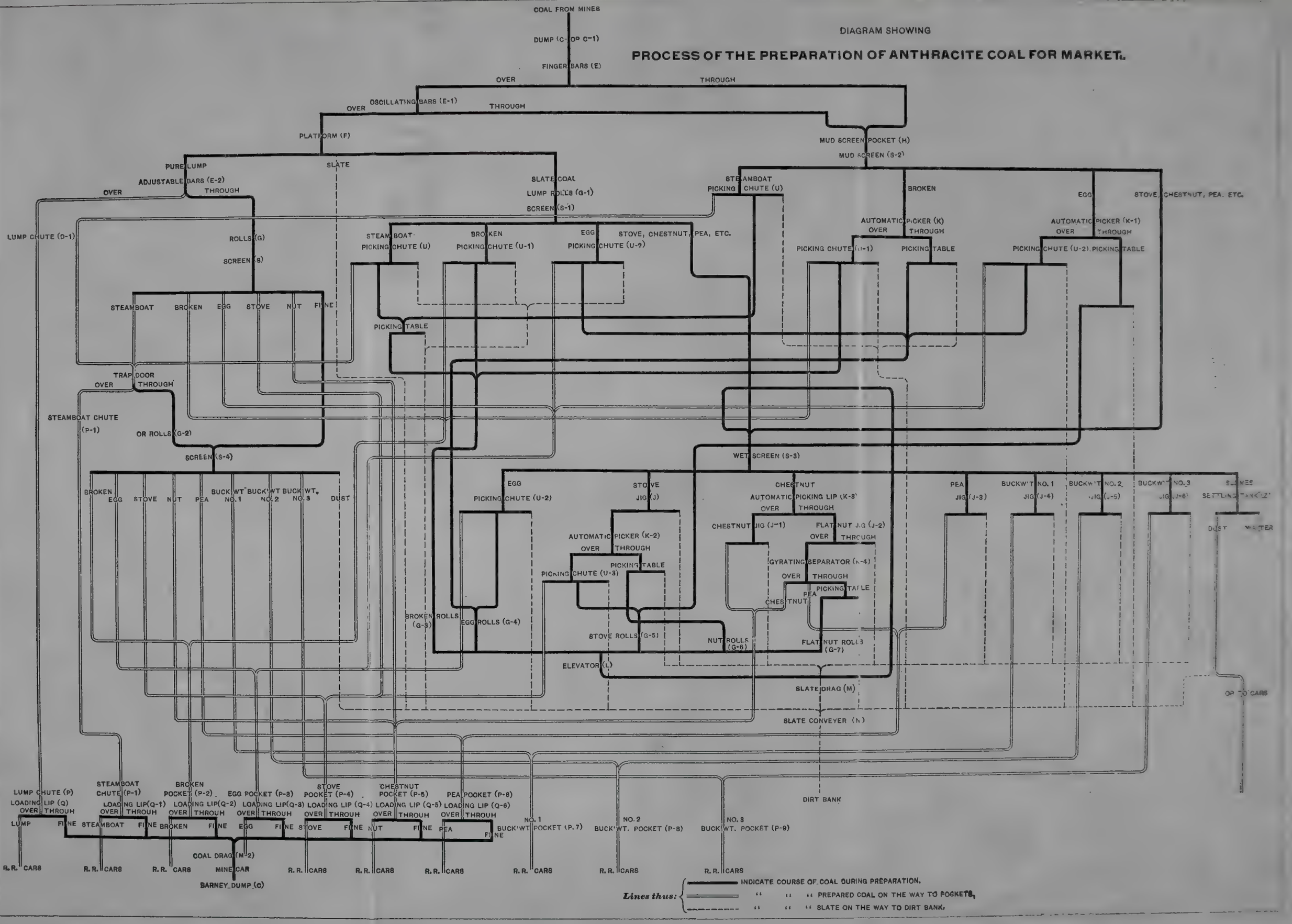


DIAGRAM SHOWING

PROCESS OF THE PREPARATION OF ANTHRACITE COAL FOR MARKET.



the coal below egg from S_1, and all the coal rebroken by the rolls G_3, G_4, G_5, G_6 and G_7. When the breaker is not breaking up steam-boat-coal, G_2 and S_4 do not run, and all the screening is done by the three pairs of screens S and S_1 (which form a double set), S_2 and S_3.

Plate XLI. is a ground-plan of the breaker, showing the relative position of pockets, lump-coal chute, settling-tank, all the drags, mouths of pockets and loading-tracks.

Plate XLII. shows the course of the coal from the wet screen, S_3, to the jigs, and from the jigs, J to J_5, to the pockets. The chutes (24) and (21) carry the stove-coal to the four jigs, J; the chute (38) in connection with (39) and (40) carries the chestnut-coal, which has gone over the slate-picker, K_3, to the three jigs, J_1; the chute (37) carries the coal to the flat chestnut jig, J_2, from which it goes to the slate separator, K_4; the chutes (27) and (29) carry the pea-coal to the two jigs, J_3; the chutes (35) and (36) carry the buckwheat-coal to the jig, J_4; and the chutes (31) and (33) carry the No. 2 buckwheat-coal to the jig, J_5. The coal from the four stove-coal jigs goes to the slate-pickers, K_2, and then to the picking-chutes below, and from there to the pockets. The chestnut-coal is carried by the chutes (14), (15), (16), (17) and (18), into the pocket, and the coal from the various other jigs is carried, as shown, to the different pockets. The size of coal for which each of the pockets is intended is shown by the letters P_2, P_3, etc. (See lists, page 464).

Plate XLIII. is a diagram of the preparation, showing the progress of the coal through the breaker. These two plates will be found useful in tracing the course of the coal through the breaker.

Concluding Remarks.

While in the description of the preparation I have shown (what is actually the case) that the No. 1 and No. 2 buckwheat are cleaned in the jigs described above, we do not intend to continue this practice. We have jigs of another type now running and giving satisfaction, but they have been in operation only a few weeks. Should they continue to work satisfactorily, we shall use the large jigs for pea, chestnut, and stove, only. The new ones are much less expensive and much simpler, and seem to work better with small coal. I will describe them hereafter in another paper, should we adopt them definitely.

All the plans for the structural work of the breaker, which were very full, accurate, and satisfactory, were made by Wilson Bros. & Co., of Philadelphia. They include everything relating to the

breaker as a structure. The other plans, including the general arrangement of the breaker, the machinery, etc., were made in our own construction-office, under the direction of Mr. Samuel Salmon. The trussed and riveted girders, the rolled beams and tie-rods, were made by the Pencoyd Iron Works, from Wilson Bros. & Co.'s plans. The rest of the work, including the machinery, etc., was built in our own shops at Drifton. The building was erected, and the laying out and drilling of the posts, struts, etc., constructed by us, was done by Mr. James W. Levan, of Eckley, Pa., without accident to any of the workmen under him. In conclusion I wish to say, that I am greatly indebted to my scientific assistant, Mr. John R. Wagner, M.E., for the material aid he has given me in the preparation of this paper. The drawings were all prepared and arranged under his direction, and much of the other work had the advantage of his careful supervision.

LIST OF PLATES.

I. Details of bars.
II. Side elevation of continuation of dump, bars, platform, etc.
III. Plan of continuation of dump, finger and oscillating bars.
IV. Plan of platform, etc., shown in Plate II.
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VI. Single gyrating screen (with various styles of cones and tracks).
VII. Side elevation and plan of single gyrating screen.
VIII. Front elevation of single gyrating screen.
IX. Box bed-plate for single gyrating screen.
X. Front view of double gyrating screen, as originally built.
XI. Double gyrating screen; plan, front and side elevation.
XII. View (from photograph) of double gyrating screen, with lips and three slate-pickers.
XIII. Adjustable corrugated rolls.
XIV. Fixed-distance corrugated rolls.
XV. Taper corrugated rolls.
XVI. Sections of roll-teeth.
XVII. Egg and stove-coal intermediate picking-chutes.

XVIII. Combination of egg and stove-coal intermediates, etc. (upper cut), and broken coal intermediate picking-chute (lower cut).

XIX. Combination of egg and broken intermediates, U_1.

XX. Jig.

XXI. Automatic slate-picker.

XXII. Gyrating separator.

XXIII. Crushed-coal elevator.

XXIV. Steamboat loading-lip.

XXV. Pocket loading-lip.

XXVI. Gunboat; side and end elevations.

XXVII. Gunboat; plan.

XXVIII. Mine-car dump.

XXIX. Gunboat-dump.

XXX. Barney-pit and bridge.

XXXI. Barney.

XXXII. Barney-dump.

XXXIII. Side elevation of breaker (looking west).

XXXIV. End elevation of breaker, jig-house and pockets.

XXXV. Plan of upper part of breaker.

XXXVI. } Details of posts, struts and tie-rods.
XXXVII. }

XXXVIII. View (from photograph) of completed breaker from the southwest, showing plane and jig-house.

XXXIX. View (from photograph) of breaker from southeast, showing settling-tank and lump-chute.

XL. Intermediate plan, showing the principal machinery used in the preparation of coal, except the jig-tanks and oscillating bars.

XLI. Ground plan of breaker.

XLII. Plan of chutes from wet-screens to jigs, and from jigs to pockets.

XLIII. Diagram of preparation.

CATALOGUE OF MACHINES AND STRUCTURES AND THEIR PARTS.

The various machines and structures are designated throughout these plates by letters, a given letter always meaning the same thing; and in the detail drawings the several parts are indicated by numbers, a given number in any drawing of the same machine or structure

always meaning the same part. The following lists will facilitate the consultation and comparison of the drawings:

List of Letters.

A, barney-pit; A_1, barney-bridge; A_2, barney; B, plane; C, barney-dump; C_1, gunboat-dump; D, dump-chute; D_1, lump-chute, circular; E, fixed finger-bars in dump-chute; E_1, oscillating bars; E_2, adjustable bars; E_3 and E_4, fixed finger-bars in lump- and steamboat loading-lip; F, platform; G, lump-rolls (pure); G_1, lump-rolls (slate-coal); G_2, steamboat (pure); G_3, broken (slate-coal); G_4, egg (slate-coal); G_5, stove (slate-coal); G_6, chestnut (slate-coal); G_7, flat chestnut-rolls; G_8, tapered corrugated rolls; H, mud-screen pocket; I, broken-coal intermediate picking-chutes; I_1, egg and stove-coal intermediate picking-chutes; J, stove-jigs; J_1, chestnut-jigs; J_2, flat chestnut-jig; J_3, pea-jigs; J_4, No. 1 buckwheat-jig; J_5, No. 2 buckwheat-jig; J_6, No. 3 buckwheat-jig; K, broken-size automatic slate- and flat-coal pickers; K_1, egg-size ditto; K_2, stove-size ditto; K_3, chestnut-size gyrating picking-lip; K_4, gyrating flat chestnut-separator; L, elevator; M, slate-drag (jigs); M_1, supplementary coal-drag (from loading-lips under pockets); M_2, main coal-drag; M_3, dust-drag (settling-tank); N, slate-conveyor; O, picking-table for steamboat and broken; O_1, ditto for egg; O_2, picking-table under automatic stove-pickers; P, lump-coal chute; P_1, steamboat-chute; P_2, broken-coal pocket; P_3, egg-coal pocket; P_4, stove-coal pocket; P_5, chestnut-coal pocket; P_6, pea-coal pocket; P_7, P_8, P_9, coal-pockets for Nos. 1, 2 and 3 buckwheat respectively; Q, loading-lip for lump-coal chute; Q_1, ditto for steamboat-chute; Q_2, ditto for broken-coal pocket; Q_3, ditto for egg-coal pocket; Q_4, ditto for stove-coal pocket; Q_5, loading-lip for chestnut-coal pocket; Q_6, ditto for pea-coal pocket; Q_7, Q_8, Q_9, ditto for No. 1, No. 2 and No. 3 buckwheat respectively; R, one cast column, lower section; R_1, ditto second section; R_2, ditto third section; R_3, ditto upper section; R-s, one cast strut; R-t, two styles of tie-rods; R-p, pin; S, pure crushed-coal screen, north box; S_1, crushed slate-coal screen, south box; S_2, mud-screen; S_3, wet-screen; S_4, dry broken pure-coal screen; T, lump-coal loading-track; T_1, steamboat ditto; T_2, outside ditto; T_3, inside ditto; T_4, reloading mine-car track; U, steamboat picking-chutes; U_1, broken-coal picking-chutes; U_2, egg- and stove-coal picking-chutes; V, slate-drag engine; W, Westinghouse engine for running jigs; X, standard mine-car; X_1, gunboat; Y, mine-car dump; Z, settling-tank.

Plate I.—Details of Bars.

Numbered Parts.—On adjustable bars E_2: 2, web; 3, base of bar-bearer; 4, V-shaped groove in bar-bearer to receive V-shaped end of web (8) on bar; 5, circular head of bar; 6, slot to receive flange (7) of bar-bearer; 7, lower flange of bar-bearer; 8, V-shaped end of web on right of bar; 9, V-shaped end of web on left of bar.

On finger-bars E, E_3 and E_4: 2, face curved to fit under flange of I-beam forming bar-bearer; 3, point or free end of bar; 4, semi-circular bolt-holes; 5, lateral edge of fixed end of bar; 6, fixed end of bar; 7, recess to allow a chipping-face along (2).

On oscillating bar E_1: 18 and 19, hook-hub on short- and long-nosed bar respectively; 24, planed roller-plate; 27, planed roller-surface; 28, roller; 35, upper inclined surface of bar; 36, upper horizontal surface of bar; 37, face of hook-hub; 38, projection of hub; 42, distance-guides; 43, short nose; 44, long nose; 45, swivel; 46, tie-rods; 47, vertical brace; 48, square head on tie-rods.

Plates II. and III.—Elevation and Plan of Dump-Chute, Bars, Mud-Screen Pocket, Platform, Etc.

Numbered Parts.—On dump-chute D and finger-bars E (see Plate I.): 2, sides on chute-bottom; 3, cast-iron chute-plates; 4, web on chute-plates; 5, 12-inch I-beam supporting chute-plates; 6, angle-iron on I-beam; 7, brace for I-beam; 8, top flange of dump-girders; 9, fish-plate on upper side of I-beam; 10, clamp-lug on under side of chute-plate.

On oscillating bars E_1 (see detail, plate I.): 2, driving-pulley (see plan, Plate III.); 3, driving-shaft; 4, pedestals for driving-shaft; 5, eccentric sheave for long eccentric-rod (7); 6, eccentric sheave for short eccentric-rod (8); 7, long eccentric-rod; 8, short eccentric-rod; 9 and 10, suspension-links for long and short eccentric-rods respectively; 11 and 12, carrying-shafts for long and short eccentric-rods respectively; 13 and 14, upper suspension-pins in links (9) and (10) respectively; 15, suspension-plate, overhead; 16 and 17, lower suspension-pins in links (10) and (9) respectively; 18, hook-hub on short-nosed bar; 19, hook-hub on long-nosed bar; 20, fulcrum-pin for gate; 21, sheet-iron plate on gate; 22 and 23, discharge-ends of long-nosed and short-nosed bars respectively; 24, planed roller-plate (see Plate I.); 25 and 26, eccentric-straps on long and short rods respectively; 27, planed roller-surface on end of bar; 28, steel

rollers; 29, fulcrum-plate for gate; 30, gate-arms; 31, angle-stiffeners for gate; 32, diamond-pointed gate-bars; 33, lifting-hook on gate; 34, lifting-link on gate; 35, upper inclined surface of bar; 36, upper horizontal surface of bar; 37, face of hook-hub; 38, projection of hub; 39, side-plates enclosing bars; 40, wedge in center of bar-approach; 41, cast-iron beam above suspension-plates; 42, distance-guides (see Plate I.).

On mud-screen pocket H: 2, twelve-inch I-beam for supporting bottom-plates; 3, connecting-angles for I-beams; 4, bottom-plates; 5, joint of bottom-plates; 6, connecting-angle for I-beam (7); 7, twelve-inch I-beam for supporting plate of lower end of pocket; 8, stiffener for I-beam (7); 9, plate for end of pocket; 10, stiffener for I-beam and support for pocket-plate; 11, I-beam for supporting upper end of platform and end of pocket.

On Platform F: 12, slate-hole; 13, slate-coal hole for coal going to lump-rolls, G_1; 14, chute for slate-coal from platform to lump-rolls, G_1; G_1, slate-coal lump-rolls; 2, guards to prevent coal from leaving space between rolls.

On adjustable bars, E_2, at head of lump-chute: 2, web of bar; 3, base or shoulder of bar-bearer; 4, V-shaped groove in bar-bearer; 5, timber for supporting upper bar-bearer; 6, short post supporting (5); 7, fifteen-inch I-beam for supporting upper bar-bearer; 8, sheet-iron guard-plate under adjustable bars; G, pure-lump rolls; D_1, lump-chute.

PLATES VII., VIII., and IX.—SINGLE GYRATING SCREEN.

Numbered Parts.—2, U-bed-plate; $2\frac{1}{2}$, box-plate; 3, driving-pulley; 4, driving-shaft; 5, crank-pin on shaft; 6, sleeve over crank-pin; 7, brass bushing over sleeve; 8, screen-box; 9, front leg on screen-box; 10, back leg on screen-box; 11, cone-track attached to screen-box; 12, gyrating cones; 13, cone-tracks attached to bed-plate; 14, cone-track bolts through bed-plate; 15, cone-track bolts through screen-box; 16 and 17, journal-brasses for upper and lower bearings, respectively; 18, cap-bolts for bed-plate; 19, step-bearing for driving-shaft; 20, lug on cap for stud-bolt; 21, counterbalance; 22, top-jacket; 23, 24, and 25, second, third, and lowest jackets respectively; 26, top discharge-chute; 27, 28, 29, and 30, second, third, fourth, and bottom discharge-chutes, respectively; 31, receiving-chute; 32, cap for lower bearing; 33, cap for upper bearing; 34, clamp-bolt for fastening jacket to screen-box; 35, strip $\frac{5}{8}$-inch square on side of jacket for fastening jacket to screen box; 36, weight in counterbal-

ance (21); 37, bolts for cap on step-bearing; Plate IX., box bed-plate.

Plate XI.—Double Gyrating Screen.

Numbered Parts.—2, gyrating cones; 3, cone-tracks; 4, front and back bed-plate castings; 5, connecting box-castings, for bed-plate; 6, middle balance-pulley; 7, vertical eccentric-shafts; 8, eccentric; 9, eccentric pedestal; 10, screen-box; 11, receiving-lip (on back end); 12, jacket-supports; 13, discharge-lips (front); 14, bushing over crank-pin on side balance-shaft; 15 and 16, side balance-shaft, and balance-pulley; 17, counterbalance; 18, crank-pin on counterbalance-shaft; 19, side bed-plate braces, 20, step-pedestal on counterbalance-shaft; 21, step-pedestal on eccentric-shafts; 22, upper eccentric-shaft pedestal-caps; 23 and 24, outer and inner laps of balance-belt; 25, upper balance-shaft pedestal; 26, step-pedestal cap; 27, cone-track bolts; 28, face-plates on bottom of screen-boxes for cone-tracks; 29, eccentric-pedestal caps; 30, driving-pulley; 31, balance crank-pin pedestal; 32, wrought-iron straps enveloping counterbalance.

Plate XIII.—Adjustable Rolls.

Numbered Parts.—1, pinion; 2, short shaft; 3, long or driving-shaft; 4, body of rolls; 6, bed-plate; 7, movable pedestal; 10, bed-plate bolts; 12, fixed pedestal-bolts; 13, long bevel-wheel shaft; 14, large bevel-wheel; 15, break-piece; 16, short bevel-wheel shaft; 17, square nut on short bevel-wheel shaft; 18, journal-boxes in movable pedestal; 19, long bevel-wheel shaft-bracket; 20, cap on movable pedestal; 21, plate over break-piece (15); 22, fixed pedestal-cap; 24, small bevel-wheel; 25, chilled points of roll-teeth.

Plate XIV.—Fixed-distance Rolls.

Numbered Parts.—1, pinions; 2, short or driven-shaft; 3, long or driving-shaft; 4, body of rolls; 5, cotter or gib; 6, bed-plate; 7 and 8, pedestal for long and short shafts respectively; 9, pedestal-cap; 10, bed-plate bolt; 11, lid on oil-cavity in pedestal-cap; 12, pedestal-bolts; 25, chilled points of roll-teeth.

Plate XV.—Tapered Corrugated Rolls.

Numbered Parts.—1, bevel-wheels; 2, short shaft; 3, driving or long shaft; 4, body of rolls; 6, bed-plate; 7 and 8, pedestals for long and short shafts respectively; 10, bed-plate bolts; 25, chilled points of roll-teeth.

Plate XVI.—Sections of Roll-Teeth.

(Scale ¼ of actual size.)

Numbered Parts.—4, body of rolls; 25, showing extent of chill on points of roll-teeth; 26, roll-teeth.

Plates XVII., XVIII. and XIX.—Picking Chutes.

Numbered Parts.—2, sharp angle in I and round angle in I_1; 3, inclined or curved portion of bottom; 4, low or delivery-end of intermediate chute I and I_1; 5, bolt-holes in flanges of I; 6, flanges of intermediates; 7, side of trough at angle (2); 8, arrow in receiving-chute of U_1 and U_2; 9, arrow showing direction of coal into intermediate I; 10, delivery-chute; 12, bottom of intermediate I_1; 13, side of intermediate cut out for convenience of slate-pickers; 14, space for slate-pickers to sit in, U_1 and U_2; 15, free sides of receiving-chutes; 16, receiving-end of intermediates; 17, sheet-iron bottom of receiving-chutes; 18, upper end of delivery-chutes.

Plate XX.—Jig.

Numbered Parts.—2, plunger-chamber; 3, part of tank directly under plunger (40); 4, dirt and water-chamber; 5, space into which coal enters; 6, dash-plate; 7, space into which pure coal rises; 8, middle jacket-support; 9, slate-hopper for sizes below stove; 9½, slate-hopper for stove-coal; 10, upper slate discharge-gate frame; 11, upper slate discharge-gate; 12, vertical rod for moving slate-gate (11); 13, lower discharge gate-frame for sizes below stove; 13½, ditto for stove-coal; 14, wood facing in (15); 15, lower slate discharge-gate for sizes below stove; 15½, ditto for stove-coal; 16, horizontal lever connected with (15); 17, bell-crank for operating slate-gate (11); 18, lever for operating bell-crank; 19 and 20, water- and dirt-discharge-gate-frame and gate respectively; 21, lever for operating dirt-and water-discharge-gates; 22, sheet-iron covering; 23, table on which coal is picked from slate; 24, sheet-iron chute upon which slate is dropped; 25, sprocket-wheel shaft; 26, sprocket-wheel for slate-drag; 27, slate-drag chain with attachments; 28, flight of chain; 29, cast-iron slate drag-trough; 30, cast bracket for bell-crank shaft; 31, cast regulating-plate for slate-exit; 32, cast overflow-plate; 33, cast draining-plate in bottom of coal-drag; 34, sprocket-wheels for coal-drag; 35, ditto, connecting pipe-rods; 36, eye-bolt on sprocket-wheel shaft into (35); 37, perforated jacket forming bottom of jigging-chamber; 38, plank floor to carry sheet-iron chute; 39, water-tight floor (white pine boards); 40, plunger; 41, cast plunger-rod foot-

end; 42, wrought-iron portion of plunger-rod (oak filling); 43, fly-wheel pulley; 44, slotted arm; 45, slotted crank-disc; 46, plunger-rod arm; 47, sliding block; 48, crank-pin and nut for crank-disc; 49, driving-shaft; 50, plunger-rod shaft; 51, pedestal for (50); 52, pedestal for (49); 53 and 54, sprocket-wheel driving-shaft and pulley (coal-drag); 55, pedestal for (53); 56 (in plan), sprocket-wheel counter-shaft; 57, slot in 44; 58, sliding-block bushing.

Plate XXI.—Automatic Slate-Picker.

Numbered Parts.—2, wider inclined face; 3, abrupt face; 4, taper slots; 5, transverse supporting-bars.

Plate XXII.—Gyrating Separator.

Numbered Parts.—2, screen-box; 3, running-plate on box; 4, support for bottom running-plate; 5, bottom-running plate; 7, wrought-iron pedestal for crank-pin; 8, cast pedestal for crank; 9, crank-shaft; 10, counterweight-pulleys; 11, discharge-chute, under slate-picker (14); 12, angle-iron supporting jacket; 13, screen-jacket, $\frac{1}{2}$-inch mesh; 14, automatic picker; 15, discharge-chute for what goes over.

Plate XXIII.—Crushed-Coal Elevator.

Numbered Parts.—2 and 3, elevator-chain and bucket; 4, sprocket-wheel; 5, traction-wheel; 6, upper pedestals; 7, upper pedestal-caps; 8, main part of bottom of upper conveyor-trough; 9 and 10, long and short sides of upper trough; 11, 12 and 13, flaring sheets to form discharge-lip from upper conveyor-trough; 14, sprocket-wheel shaft; 15, spur-wheel on sprocket-wheel shaft; 16, flare on uptake-plate, to prevent spilling; 17, sheet-iron forming bottom of lower trough; 18, sole-plate for lower movable pedestal; 19 and 20, adjustable lower pedestal and cap; 21, lower tension traction-wheel; 22, receiving-lip on screen-box (S_{33}); 23, curved-down end of bottom of upper trough on uptake side; 24, sheet-iron casing on down-take side.

Plate XXIV.—Steamboat Loading-Lip.

Numbered Parts.—2, flaring sides of lip above bars; 3, flaring sides of lip on either side of bars (7); 4, timber; 5, cast-iron plates in mouth of lip; 6, bar-bearer; 7, finger-bars (see E_4, Plate I); 8, diamond-pointed gate-bars; 9, stationary part of spout; 10, sliding part of spout; 11, planking, forming breast of gate; 12, flanges on

end of bar-bearer (6); 13, finger-bar bolts; 14, stiffeners on end of sheets (3); 15, screenings-hopper under bars; 16, inclined trough leading to screenings-drag M_2 (see Plate XXXIV.); 17, bracket for carrying friction-roller (18) running on (9); 18, friction-roller running on (9); 19, gate-arms; 20, gate-fulcrum; 21, hanger for long friction-roller under movable spout; 22, long friction-roller under sliding spout; 23, shaft for long friction-roller; 24, bottom flange on side sheet (2); 25, point of partition-wedge.

Plate XXV.—Loading-Lips for Pockets.

Numbered Parts.—2, post of breaker; 3, I-beam supporting plates forming floor of pockets; 4, floor of pockets; 5, bracket on post to support I-beam; 6, side-sheets on inside spout; 8, screen-jacket forming bottom of lip; 9, side-sheet of stationary part of spout on outside lip; 10, side-sheet of adjustable spout on outside lip; 11, long roller under adjustable lip; 12, axle for long roller (11); 13, angle-iron support for screen-jacket for outside lip; 14, fulcrum-bracket for gate; 15, gate for outside lip; 16, bracket for attaching rope; 17, frame for friction-rollers, rolling on (9); 18, friction-rollers, running on (9); 19, slot to admit gate-fulcrum; 20, angle-iron support for screen-jacket on inside lip; 21, gate for inside lip; 22, bracket for attaching rope; 23, fulcrum-bracket for gate; 24, slot to admit gate-fulcrum; 25, rib on floor of pocket to attach rope-pulley; 26, pedestal for rope-pulley; 27, rope-pulley; 28, sheet-iron forming inside of drag-hopper; 29, sheet-iron forming outside of drag-hopper; 30, angle-connection between drag-hopper-side and outside spout; 31, cast drag-trough; 32, drag-flights; 33, brace to support inside of drag-hopper; 34, hanger for long friction-roller; 35, angle-connection between post and outside sheet of drag-hopper; 36, angle-connection between inside sheet of drag-hopper and spout; 37, built-up 24-inch I-beam; 38, cast-iron chair supporting I-beam (3); 39, rabbet-strip cast on side of post; 40, cast plates in end of pockets fitting in rabbet-strips (39).

Plates XXVI. and XXVII.—Gunboat.

Numbered Parts.—9, front wheel; 10, back wheel; 11, front axle; 12, back axle; 14, rope-socket; 15, link connecting rope-socket and swivel; 16, swivel; 17, clamp over pulling-beam; 18, head of swivel-pin; 19, spreader-bar (I-beam); 20, front pulling-bar; 21, pin for front pulling-bar; 22, front side-stiffeners; 23, back pulling-bar; 24, pin for back pulling-bar; 25, stirrup for

pulling-bars; 26, guide for pulling-bars; 27, wings on mouth of gunboat; 28, pedestal for back axle; 29, pedestal-cap for back axle; 30, brass bearing for back axle; 31, long side of hinges for wings; 32, short side of hinges for wings; 34, angle-part for side-sheet; 35, stiffeners for mouth at wings; 36, rest for wings; 37, pin for wing-hinges; 38, side-sheet; 39, back brace for inside; 40, front brace for inside; 41, pedestal-cap for front axle; 42, pedestal for front axle; 43, angle-iron stiffeners; 44, riveting-plate on spreader-bar (19), Plate XXVII.; 45, top sheet; 46, bottom sheet; 47, back sheet; 48, pin in cone-socket; 49, side- and bottom-binder; 50, nut on pin for front pulling-bar; 51, bolts for front pedestal-cap; 52, bolts for back pedestal-cap.

Plate XXVIII.—Mine-Car Dump.

Numbered Parts.—2, cradle; 3, curved rack; 4, locking-arm; 5, connecting-link; 6, cam; 7, pedestals; 8, hand-lever; 9, back cradle-shaft; 10, front cradle-shaft; 12, lever and cam-shaft; 13, horn.

Plate XXIX.—Gunboat-Dump.

Numbered Parts.—2, rail for front wheel; 3, flat strap of iron upon which rear wheels of gunboat run; 4, flat iron strap to protect timber and act as guide; 5, second abrupt change in back-wheel track; 6, abrupt change in front-wheel track; 7, third abrupt change in back-wheel track; 8, timber upon which back-wheel track is built; 13, eight-foot rope-sheave; 29, floor of upper part of plane.

On gunboat: 9, front wheel; 10, back wheel; 11, front axle; 12, back axle; 14, rope-socket; 15, link connecting rope-socket and swivel; 16, swivel; 17, clamp over spreader-bar; 18, head of swivel-pin; 19, spreader-bar (I-beam); 20, front pulling-bar; 21, pin in front pulling-bar; 22, guide-strip to prevent pin (21) from striking; 23, back pulling-bar; 24, pin-connection between stirrup and back pulling-bar (23); 25, stirrup for pulling-bars; 26, guide for pulling-bars; 27, wings on mouth of gun-boat; 28, back pedestal.

On dump-chute: 2, sides on chute-bottom; 3, cast-iron chute-plates; 4, web on chute-plates; 5, I-beam supporting chute-plates; 6, angle-iron stiffener on I-beam; 7, brace for I-beam; 8, top flange of dump-girders; 9, fish-plate on upper side of I-beam.

Plate XXX.—Barney-Pit and Bridge.

Numbered Parts.—2, trunnion-stanchion; 3, trunnion-pedestal, right hand; 4, trunnion-pedestal, left hand; 5, vertical timber to

receive trunnion-pedestal (4); 6, horizontal timber carrying trunnion-pedestal (3); 7, bridge-rails; 8, upper rocker-bar, automatically shifted by barney-axle, throwing levers (9) and (10), the latter pulling (14) and closing bridge A_1; 9, upper idle rocker-lever; 10, upper acting rocker-lever; 11, brace-end supporting counterweight, 13; 12, counterweight-lever; 13, counterweight to balance bridge-shifting mechanism; 14, upper shifting-rod to close bridge A_1; 16, mine-car rail; 17, rail for elevating pusher on upward motion; 18, rail in pit for barney-truck wheels; 19, lower idle rocker-lever; 20, lower rocker-bar for opening bridge, operated by axle of barney-truck on downward motion; 21, lower acting rocker-lever; 22, bearing for (21); 23, bolts for (22); 25, bearing for (19); 26, bolts for (25); 27, overhead guard-rail for barney-truck wheels; 28, bearing for pusher-latch (29); 29, pusher-latch raised on downward motion of barney; 30, releasing-lever; 31, bearing for trip-lever (33); 32, pin for trip-lever; 33, trip-lever for raising check-bar (43); 34, trip-rod to raise levers (42) and (43); 35, rocker-arms to lower levers (42) and (43); 36, stop on link (42); 37, counterweight-lever; 38, releasing-rod; 39, rocker-shaft; 40, trip-counterweight; 41, set-screw on trip-weight; 42, check-link; 43, check-bar; 44, washer for spring-bearing; 45, check-spring; 46, check-bar buffer; 47, washer on check-bar; 48, nut on check-bar; 49, lower shifting-rod to open bridge; 50, bridge-shifter; 51, fixed bar with pin on which shifter (50) turns.

Plate XXXI.—Barney.

Numbered Parts.—7, back wheel; 8, pusher-wheel for raising pusher; 12, check-horn; 13, riding-horn; 14, wire-rope socket; 15, main truck-frame; 16, pusher-shaft; 17, back truck-wheel pedestal; 18, front truck-wheel axle; 19, lower sill of pusher-frame; 20, wire hoisting-rope; 21, pusher-shaft brace; 22, pusher-wheel axle; 23, cap for back truck-wheel pedestal; 24, cap-bolts for back truck-wheel pedestal; 25, rope-clevice; 26, front truck-wheel; 27, hook on end of pusher-shaft brace; 28, pin for rope-clevice; 29, cotter for rope-clevice pin; 30, top-bolts for pusher-shaft brace; 31, base-bolts for back truck-wheel pedestal; 32, tie-bolts for pusher-shaft strap; 33, braces in pusher-frame; 34, pusher-shaft strap; 35, binding-bolts for pusher-frame; 36, thin binding for pusher-frame; 37, front truck-wheel pedestal; 38, cap for front truck-wheel pedestal; 39, bolts for front truck-wheel pedestal; 40, guard for rope; 41, upper sill of pusher-frame; 42, base-bolts for front truck-wheel

pedestal; 43, vertical-bolts for front truck-wheel pedestal; 44, bumper-plate for pusher; 45, bolts for bumper-plate on pusher; 46, bolts for braces in pusher-frame; 47, bottom brace-bolts for pusher-shaft; 49, large timber under pusher-shaft; 50, binding-bolts for front part of truck-frame; 51, cast-iron bearing-block for pusher-shaft.

Plate XXXII.—Barney-Dump.

Numbered Parts (for parts of the barney proper, the numbers on this plate are the same as those on the corresponding parts of Plate XXXI.).—2, mine-car rail; 3, barney-rail; 4, abrupt change in grade of barney-rail; 5, second change in grade of barney-rail; 6, check-horn on end of mine-car rail (2); 27, five-foot sheave-wheel; 28, heavy timber under rails; 29, floor on top of plane; 30, beam over wrought-iron girders.

On mine-car: 9, tail-gate; 10, gate-latch; 11, bumper; 17, cross-beam on back of car; 23, front car-wheel axle; 24, front car-wheel; 25, back car wheel.

On dump-chute: 2, sides on chute-bottom; 3, cast-iron chute-plates; 4, web on chute-plates; 5, I-beam supporting chute-plates; 6, angle-iron stiffener on I-beam; 7, brace for I-beam; 8, top flange of dump-girders; 9, fish-plate on upper side of I-beam.

Plates XXXIII. and XXXIV.—Side and End Elevations of Breaker.

Numbered Parts.—2, built-up I-beam carrying dump; 3, large girder at top of mud-screen pocket; 4, taper built-up girder, forming sides of mud-screen pocket, H, and carrying dump-chute and bars; 5, built-up I-beam carrying platform, F; 6, check-horn on end of mine-car rail on dump, C, Plate XXXII.; 7, inside girder carrying sheaves (18); 8, outside girder carrying sheaves (18); 9, fifteen-inch I-beam carrying mud-screen, S_2; 10, latch on tail-gate of mine-car; 11, twenty-four-inch I-beam, carrying driving-mechanism of jigs, also screens, S and S_1; 12, fifteen-inch I-beam, carrying timbers under wet-screen; 13, eight-foot sheaves at dump for gunboat-rope; 14, twenty-four inch I-beam, carrying east end of jig-tanks; 15, I-beam, carrying broken-coal screens, S_4; 16, fifteen-inch I-beam; 17, twenty-four-inch built-up I-beam under deep end of pockets; 18, eight-foot sheaves nearest to hoisting-engines, north side; 19, line-shaft from Westinghouse engine for driving jigs; 20, line-shaft for driving drags taking coal from jigs; 21, pulley from which gyrating

separator K_4 is driven; 22, bevel-wheel shaft for driving supplementary coal-drag, M_1; 23, eighteen-inch built-up I-beams for gunboat and plane; 24, slate-chute under jigs (see PlateXXXIV.); 25, built-up I-beam under pockets; 26, rolled I-beam at shallow end of pockets; 27, five-foot sheave on dump for barney-rope; 28, built-up I-beam under (26); 29, slate-drag trough under jigs; 30, screenings-hopper above T_4; 31, sprocket-wheel for driving supplementary drag, M_1; 32, inverted ∩-iron opening latch (10); 33, trap-door over steamboat-rolls, G_2; 34, twelve-inch timber carrying jig-tanks; 35, lattice-girder for jig-house; 36, I-beam carrying S_4; 37, pockets to carry one end of upper jig-timbers; 38, chute-floor for jig-drawings, slate and slimes; 39, vertical line-shaft; 40, lump-coal screenings-chute to drag, M_2; 41, platform for slate-drawer; 59, scraper-chain of jigged coal.

Plate XLII.—Plan of Chutes from Wet-Screen to Jigs, and from Jigs to Pockets.

Numbered Parts.—1, stove-coal delivery-chute; 2, space in which intermediate picking-chutes I_1 are placed; 3, receiving-chutes; 4, same as (2); 1, 5, 7, 8 and 9, stove-coal delivery-chutes; 10, chute to gain pitch for pea-coal delivery chute; 11, pea-coal delivery-chute; 12, No. 2 buckwheat delivery-chute; 13, No. 1 buckwheat delivery-chute from jig direct; 14, chestnut-coal delivery-chute from gyrating separator; 15, 16, 17 and 18, chestnut coal delivery-chutes from jig direct; 19, stove-screen-lip on south box; 20, jigged-coal elevator shaft; 21 and 22, stove-coal chutes from screen to jigs; 23, stove-coal screen-lip on north box; 24 and 25, stove-coal chute from screen to jigs; 26, pea-coal screen-lip on south box; 27, pea-coal chute to jigs; 28, pea-coal screen-lip on north box; 29, pea-coal chute to jigs; 30, No. 2 buckwheat screen-lip on south box; 31, No. 2 buckwheat chute to jigs; 32, No. 2 buckwheat screen-lip on north box; 33, No. 2 buckwheat chute to jigs; 34, No. 1 buckwheat screen-lip on south box; 35, No. 1 buckwheat chute to jigs; 36, No. 1 buckwheat screen-lip on north box; 37, flat-chestnut chute to jig; 38, 39 and 40, cubical-chestnut chutes to jigs.

IRON AND LABOR.

BY A. S. HEWITT, NEW YORK CITY.

(Presidential Address at the New York Meeting, September, 1890.)

After an interval of fourteen years, saddened for all of us by the death of David Thomas, the father of the anthracite iron-trade, first president of the Institute, and by the untimely loss of his successor, Alexander L. Holley, whose memory we are about to honor by loving services, the members of the American Institute of Mining Engineers have conferred upon me for the second time the office of president, under circumstances similar to those which led to the choice in 1876. Then the whole world was represented at the great exposition of industry which was held in Philadelphia to celebrate the one hundredth anniversary of our declaration of political independence. In 1890 it is our privilege to receive as guests the members of the British Iron and Steel Institute, who, with singular and graceful felicity, have appointed a meeting in the United States in the year when we pass from the second to the first place in the international column of the production of iron, and thus celebrate our industrial independence. For the honor of being chosen to extend to our distinguished guests the right hand of fellowship and a hearty welcome I return to my associates the grateful acknowledgment of one whose chief claim to your favor is to be found in his love for the professional occupation in which we are all engaged, and in his efforts, public and private, to promote its advancement into new fields of usefulness.

In order to indicate the full significance of the visit of our foreign guests at this time, it will be proper to explain briefly the origin, aims and work of the Iron and Steel Institute. It unquestionably owes its birth to the International Expositions held in London in 1851 and 1862, and in Paris in 1867. These expositions served to break down the jealousies of nationality and to diffuse a better knowledge of industrial processes. Trade-secrets, which had been carefully guarded, became common property; manufacturers who had previously regarded themselves as rivals formed bonds of union and ties of friendship, as the results of an intercourse which

broke down all prejudices and led them to see that their highest interests would be promoted by the free interchange of experience and ideas. The progress of the world in the arts of civilization since 1851 has been phenomenal. In fact, the face of the industrial world may be said to have been entirely reconstructed since the time when the first international exposition was held. The figures which I shall adduce in regard to the development of the iron and steel industries during the last decade, will make it clear that new agencies have been at work for the amelioration of humanity and the progress of civilization. Among these agencies I can think of none more powerful than the influence exerted by the Iron and Steel Institute through its meetings, its discussions and the publication of its transactions. The comprehensive nature of its work, and the absolute honesty and frankness with which it has been carried on, afford the best illustration of the catholicity of the age in which we are so fortunate as to live, and give promise of the good time when all barriers to the intercourse of men shall be broken down, and it shall be demonstrated that the true interests of mankind are best promoted by the free play of the laws of nature.

So far as I know, this was the first considerable organization of a special industrial interest, deliberately formed for the acquisition and diffusion of scientific and practical knowledge, independently of the question of profit and commercial results. I venture to say, that in the history of the human race there is nothing approaching the records of the Institute in the disclosure to the whole world of the most valuable and intricate processes, in previous times regarded as special secrets and carefully concealed as the property of private establishments.

Although British in origin, the Institute has invited and secured the co-operation of the ablest men, both practical and scientific, in all countries; and its membership is as widely diffused as are the materials of the great industry to which it is devoted. Its organization dates from 1869, when it was formed with 292 members; to-day it numbers over 1600. In 1870 the world's product of pig-iron was 11,900,000 tons; in 1889 it amounted to 24,869,534 tons. Meanwhile the processes of manufacture have been revolutionized, and a large portion of the world's consumption has been changed from iron to steel in consequence of the improvements which have been either originated by members of the Institute, or made the common property of mankind by the complete disclosures of new discoveries and processes made in its Journal. It is not too much to say that the tone of the trade has been raised

to a plane of intelligence and liberality which would have been absolutely incomprehensible to the past generation, and which opens for those who are to come after us a new world of generous competition in promoting the welfare of the race. The history of the achievements of the Iron and Steel Institute thus forms a complete answer to those who are inclined to take a pessimistic view of humanity; for it has shown how all the members of the largest branch of industry, lying at the very foundation of civilization, can be brought to work together for the common good and for the more equitable distribution of the products of human effort.

It has been from time to time the wise practice of the Institute to hold meetings outside of Great Britain, in countries where there is a considerable development of the iron and steel business. It has reserved, however, its meeting in this country until by the inevitable growth of population and the consequent development of our resources, we have become the largest producers of pig-iron and of steel among the nations of the world.

In recognition of this felicitous compliment, we can assure our guests that the time has been accelerated by many years in consequence of the inventions of members of the Iron and Steel Institute, chief among whom, we are glad to say, still survives in the conspicuous personality of Sir Henry Bessemer, *clarum et venerabile nomen*, whose absence on this occasion, in consequence of his advanced years, causes a feeling of profound regret throughout the length and breadth of the land. They have come largely to see the fruits of their own good work, and it will be our fault if they do not receive a hospitality which will remove from their hearts anything of regret that the primacy in the production of iron and steel is passing from the parent to the child. If they have divided their inheritance with us, we at least are ready to acknowledge the obligation, and as hosts to tender them the first fruits of the magnificent resources with which we have been endowed, and which we have been enabled to develop with the co-operation, counsel, and large experience of the mother-country.

Our own Institute, now consisting of nearly 2000 members, and the Verein Deutscher Eisenhuettenleute of over 1000 members, and represented here by a numerous delegation, both owe their existence to the example of the parent society, whose record they have striven to emulate, and whose pre-eminence is acknowledged with admiration and gratitude.

Having thus endeavored to express our profound appreciation of

the honor of their visit to this country, I proceed now, in a general way, to indicate the subjects to which their attention will necessarily be directed in order to understand the condition of the iron and steel business in this country, and its further prospects. I shall carefully abstain, however, from all description of the great establishments which have grown up in the last ten years, of the improved processes which they employ, and of the technical details which will be presented at the international meetings by representatives of both continents, fully qualified to deal with the scientific and practical phases of the business. I shall devote what I have to say to a consideration rather of the conditions under which the iron and steel business has grown to its present proportions, and to the difficulties in the way of its development on the scale demanded by the growth of the world and the progress of civilization.

The possession of fuel determines the direction, growth and remuneration of modern industry. The mineral-fuel of the world is largely under the control of the Teutonic races, and hence the iron and steel business has been most extensively developed in Great Britain, Germany and the United States. Of these three countries, the latter has the largest and most cheaply worked deposits of fuel. Of the country lying east of the Mississippi river, the coal-field occupies about 135,000 square miles, generally covered by a fertile soil, traversed by natural water-ways, and tapped by railways aggregating over 100,000 miles in length. Around the edges of this vast coal-field the other geological formations bring to the surface the deposits of iron-ore belonging to the successive strata from the crystalline rocks to the recent tertiary deposits. The accumulation of ore is, in some portions of the country, upon a scale of grandeur which may well excite the wonder of the beholder. The connection between these vast deposits of ore and the fuel required for smelting them is, as a rule, remarkably convenient and easy. The magnetites of New York and New Jersey have been connected by rail with the anthracites of Pennsylvania, so that they may be brought together upon favorable conditions. The hematites and specular ores of Lake Superior reach the coals of Illinois and Ohio by a water-communication which has been so perfected as to reduce the cost of transport below one dollar per ton. The magnificent deposit at Cornwall, in Pennsylvania, which our guests will visit, is within 40 miles of anthracite-coal, and accessible to coke at rates which leave nothing to be desired. When we pass to the South, we find in Alabama that the coal and ore are usually

within 25 miles of each other, and sometimes to be found lying one over the other upon the same property. Between Pennsylvania and Alabama the ranges of ore extend in and along the Appalachian chain, in close proximity to admirable fuel, which, during the last few years, has been made accessible by railways and canals.

When the government of the United States was organized under the Constitution in 1789, the existence of these coal-fields was unknown, and only a few deposits of iron-ore skirting the coast had been opened for the use of charcoal as a fuel. The first census of the United States, made just a hundred years ago, disclosed a population of not quite 4,000,000 inhabitants residing near the coast, without capital and with no industries developed, except those of a domestic nature, carried on according to the rude methods of a primitive civilization. No steam-engine had as yet been erected within our borders, and no other means of transportation existed except the water-ways which had been provided by nature, or the rough roads over which communication was kept up with difficulty between sparse and distant settlements. Except in natural resources, no civilized people was ever poorer than was this nation at the close of the Revolutionary war, but none was ever endowed with so much hope, courage and intellectual activity, or with so grand a field for the exercise of these good qualities. Moreover, our fathers were God-fearing men, who thoroughly believed that "the earth was the Lord's, and that they were His people." In 1776 they had proclaimed political independence, but it was not until 1789 that they became a nation. The same instrument which decreed the fusion of the States into one Union as the fundamental condition of nationality, to be subsequently maintained by a war which cost eight thousand millions of dollars and the sacrifice of more than a million of lives, and is attested to-day by a pension roll of over one hundred millions of dollars per annum, laid the foundation of industrial progress and independence by declaring that "no State shall pass any law impairing the obligation of contracts," and that "no person shall be deprived of life, liberty or property without due process of law." With the assurance of protection thus afforded, the young nation started boldly out to conquer sea and land. The genius of Alexander Hamilton set in motion the wheels of progress by the honest funding of the public debt, and by encouraging the energies of a race of men ready alike to vex the ocean with their sails and to drive back the Indian by their steady march beyond the Alleghanies, disclosing the mineral-wealth that was stored within the mountain fastnesses. Roads were built, over

which toiled the cumbrous Pennsylvania wagons; and the men who had marched with bleeding feet from New Jersey to Yorktown crossed the Alleghany mountains and occupied the rich regions of Ohio, Tennessee and Kentucky. Their toilsome paths were over the coal-beds, and in their train followed the canals and railroads which have enabled us so to assemble the raw materials as to place ourselves within a century at the head of the nations in the production of iron and steel. In these early days there was no time for strikes and no provocation for boycotts. There were no capitalists; and the only walking-delegates to be found were those who travelled on foot to represent their communities either in the Legislature or in Congress. They were all Americans, made brothers by mutual trials and struggles; not a mongrel mass of foreign races ignorant of the true basis of personal liberty which our fathers had incorporated into the Constitution as the perpetual guaranty of freedom and progress. In vindication of these fundamental principles of our national existence, we may yet have to make greater sacrifices of men and money than were required for the maintenance of the Union. For, let our visitors from abroad be assured, the people who have been able to conquer a continent in a hundred years will not allow themselves to be deprived of the right to life, liberty and the pursuit of happiness at the dictation of any organization, either of capital or of labor, however powerful.

Until communications by canal or rail had been established between the coal-fields and the ore-beds, the progress of the iron business was necessarily slow. Its subsequent story is told in the following table, showing the comparative rate of increase in population and in the production of iron:

Year.	Production of Pig-iron. Gross Tons.	Rate of Increase.	Population.	Rate of Increase.
1830	165,000	.....	12,866,020	32.51
1840	315,000	91 per ct.	17,069,453	33 52
1850	564,000	79 "	23,191,876	35.83
1860	821,223	46 "	31,443,321	35 11
1870	1,696,429	106 "	38,558,371	22.65
1880	3,835,191	126 "	50,155,783	30.08
1890	8,552,679*	123 "	64,000,000	28.00

* This is the product for the year ending June 30, according to the census returns, whereas the preceding figures are for calendar years.

This table brings out the striking conclusion that the production of pig-iron has always increased more rapidly than the population, and that the ratio is an increasing one. Between 1830 and 1860 the production of iron increased twice as fast as the population. Between 1860 and 1890 it increased four times as rapidly, thus proving that the national wealth continues to grow from decade to decade at a rate of acceleration of which the world affords no previous example. Inasmuch as during all this time we have imported iron in addition to our production, it follows that the consumption per capita has also increased more rapidly than population. In 1855, according to careful calculations which I made at that time, we were consuming iron at the rate of 117 pounds per head; whereas in 1890 the consumption has increased to rather more than 300 pounds per head, the whole of which, for the first time in our history, we are producing within our own borders.

Great Britain, on the other hand, produces more iron than it consumes, and is still the largest per capita producer in the world. In 1889, with a production of 9,321,563 tons of 2000 pounds, and with a population estimated at 38,000,000, the production reached the large figure of 495 pounds per head. Deducting the exports, Great Britain is now consuming 250 pounds per head against a consumption of 144 pounds in 1855. But the production of iron in Great Britain appears to be now very nearly stationary, as will appear from the following table, giving the quantity produced since 1880:

	Gross tons.
1880,	7,749,233
1881,	8,144,449
1882,	8,586,680
1883,	8,529,300
1884,	7,811,727
1885,	7,415,469
1886,	7,009,754
1887,	7,559,518
1888,	7,998,969
1889,	8,245,336

This table discloses the fact that in 1889 Great Britain, while making an increase over 1888, was not able to reach the product of the years 1882 or 1883. It may, therefore, be concluded that no considerable increase of production is to be expected, especially in view of the facts that the present production is only maintained by the importation of foreign iron-ores to the extent of 20.4 per cent. of the total ore-consumption, and that the ratio of foreign ore consumed in Great Britain is a steadily increasing one.

In the United States the case is otherwise. We have been able to increase our total product year by year, without increasing the importation of foreign ores, none of which are in fact necessary to the existence and growth of the business, although in some localities, near the coast, foreign ores are desirable for the production of Bessemer pig. Assuming, then, that the production of Great Britain is not likely to be increased, and that the growth in the consumption of iron is to go on with the increase of population and the progress of industry in the future as in the past, we are in a position to estimate the demands which will be made upon the resources of the United States for the next ten years. The population in 1900, allowing the same rate of increase of 28 per cent. as in the last decade, will be 82,000,000. A consumption of 300 pounds per head will require 24,600,000,000 pounds of iron, equal to 12,300,000 tons of 2000 pounds. If, in accordance with the accelerated ratio of consumption, which has trebled since 1855, the per capita demands shall rise to 400 pounds, the total tonnage required will be 16,400,000 tons of 2000 pounds, equivalent to 14,800,000 gross tons. The consumption of the world in 1889 is estimated at 24,869,534 gross tons, of which we produced 30 per cent., and Great Britain 33 per cent. The consumption of the world has increased in eleven years from 14,117,902 tons to 24,869,534 tons, or nearly 11,000,000 tons. Assuming that the coming ten years will require an equal increase (and it is likely to require more), the United States must supply 7,000,000 tons, and the rest of the world 3,000,000, in order that the two continents may not be forced to draw supplies from each other. I think it is safe to estimate, therefore, that in 1900 the world will require 35,000,000 gross tons of iron, of which the United States must supply 45 per cent., and the other iron-producing countries the remainder, in the proportion of half to Great Britain and half to Germany, France, Belgium and the other smaller producers.

The question presents itself, whether this vast demand can be met without such an increase in price as will tend to restrict the use of iron within narrower limits. The increased call upon the European countries is too small to make any embarrassment, except such as may arise from the fuel- and food-questions, both of which are serious problems in the European industrial world. The answer to the question must, therefore, come from the United States; and in view of the changed relations of the two continents in regard to the supply of iron, it will be seen that tariff-legislation will henceforth play no part in the solution of the problem. According to the traditions

of the Institute of Mining Engineers, it is not permissible to discuss commercial questions; but there is no impropriety in calling attention to the fact that one of the perplexing elements in the consideration of the future of the iron business is removed from the problem when the conclusion is reached that henceforth it will task the ability of Europe to supply its own demand for crude iron, and that the United States must look to its own resources for the supply of the great demands of the coming century. It is quite evident that the only effect of transferring any considerable portion of this demand from the United States to Europe will be to raise the price of iron, so that thenceforth the competition in the open markets of the world will be more favorable to our manufacturers than it has been in the past.

So far as we are concerned, then, the question is substantially whether this country can nearly double its production in the next ten years without so seriously increasing the present cost of iron as to restrict the consumption, and arrest the rate of progress at which the world is now moving forward.

The production of iron involves the five elements of fuel, ores, capital, labor and skill.

Fuel.—Upon reference to the map of the United States* on which Mr. Kunhardt has outlined the coal-fields, showing their relations to the deposits of iron-ore, it becomes evident that there is practically no limit to the quantity of coal which can be supplied on demand. The growth of this product corresponds very closely with that of the production of pig-iron:

	Coal Mined, Tons.
1870,	28,312,581
1880,	65,883,000
1889,	137,455,172

More than doubling itself in each decade. The capacity for production is always so far in excess of demand that it is often necessary to limit the amount forwarded to market, by the action of the great corporations engaged in the mining and transportation of coal. Doubtless there will be required to meet the demand in A.D. 1900 nearly or quite 300,000,000 tons of coal. This can readily be had from the fields which are now open and have direct communication with the deposits of iron-ores.

Iron-Ores.—The map shows also the regions from which we now derive our present supplies. That these can be doubled in ten years does not admit a doubt. Probably the most remarkable points of interest to our guests will be the great deposits of Cornwall, in

* This map was exhibited in Chickering Hall during the New York Meeting.

Pennsylvania, of the Marquette, Menominee, Gogebic, and Vermilion ranges in the Lake Superior region, and the ores of Tennessee and Alabama, extending in an unbroken bed for hundreds of miles along the flanks of the great Appalachian coal-field. The following statement of the growth of the business in the Lake Superior region will serve to show the facility with which the supply can be increased:

	Tons Mined.
1885,	2,466,372
1886,	3,568,022
1887,	4,730,577
1888,	5,063,693
1889,	7,292,754

Showing that in five years the quantity has increased threefold.

The South is practically a virgin country, in which the production of ore is in its infancy; but the development is already phenomenal, and even if the other regions ceased producing ore, the Southern States could readily supply the deficiency. It is remarkable also that the vast deposits of hematites and red fossiliferous ores with which the South is endowed are adapted to the "basic" process, while the ores of Lake Superior are suited to the "acid" process. Thus the two sections are practically the complements of each other in the work of supplying the needs of the country for steel. It will doubtless excite surprise in the minds of our visitors to find that the basic process has made little progress in this country. The delay has been due partly to the recent development of the Southern ores, and partly to the illiberal spirit in which the basic patents have been managed. But it will not longer be possible to arrest the manifest destiny of the South which is now erecting a large number of furnaces, the product of which must find a market through the basic process.

Capital.—The total wealth of the United States will be reported in the census just completed, but the figures are not yet available. In 1880 the amount was $43,642,000,000, which is equal to $870 per head of population. The rate of increase of population for the previous decade was 30.13 per cent., and of the per capita wealth 45.47 per cent. During the decade ending in 1890, the rate of increase of population is 28 per cent., and if the ratio of increase in wealth is only the same as in the previous decade, it will amount to 42 per cent., making the per capita wealth $1235. The actual figures will undoubtedly show a larger amount for each inhabitant.

In Great Britain, according to Robert Giffen, the wealth per capita, at present, is £270, equal to $1300, so that the two countries

are probably on an equality of wealth as to each inhabitant; but the aggregate wealth of this country now, and for the first time, exceeds that of Great Britain, although the amount of floating capital is larger there than here. But it must be remembered that the floating capital of the world is now practically mobilized, so that, if a deficiency exists in the United States, it is promptly supplied from abroad.

Now, the wealth of Great Britain has been adequate for the annual production of 495 pounds of pig-iron per head. It cannot, therefore, be doubted that, with equivalent wealth, we could meet a demand of the same extent. If such a result should be reached in 1900, we should produce 19,000,000 gross tons of iron, which exceeds the estimate already made as to the probable requirements by more than 1,000,000 tons. The very large absorption of capital in the erection of new furnaces in the Southern States, during the last two years, supplying a capacity of at least 1,000,000 tons per annum, when the furnaces now under construction shall have been completed, has been readily met; and this goes to show that there is no practical difficulty in getting the means to supply any quantity of iron which the market will take.

Labor.—For the supply of the raw materials and the smelting of our present product of pig-iron, about 200,000 men are required. The labor of one man, therefore, now suffices to produce rather more than 40 tons of pig-iron per annum. To produce, in 1900, double the quantity now produced, will require the labor of 200,000 additional men. This is not more than the number of male emigrants who come annually to our shores, and it is but a small percentage of the normal increase of our population, which between now and 1900 will reach at least 16,000,000 of persons. So far, therefore, as the supply of labor for the increased production is concerned, we need be under no apprehension.

Skill.—Our foreign visitors are about to make a critical survey of the iron- and steel-works of the country, and their judgment as to efficiency and management will be accepted as final. In 1876, when this country had not fully entered upon the manufacture of steel, in which it now leads the world, producing one-third of the whole supply, the foreign engineers bore testimony to the superiority of our appliances, and to the greater yield per man. Our methods and labor-saving machines were at once copied by the best European establishments. I think I am safe in saying that we have not gone backwards in the interval, and that our guests will still find some-

thing of value to be given in exchange for the priceless contributions which, since 1876, they have so generously made to the progress of metallurgy in this and other countries. At any rate, I think we have been quite ready to learn, and to take advantage of every advance, at home and abroad, so that our technique will be found to be fully up to the highest known standard of excellence.

It seems to be clear, then, that in all the elements required to meet the increased demands of the world for iron and steel, the United States are abundantly equipped. But it is not enough to have adequate supplies of ore and fuel. They must be so situated as to be brought cheaply together at the place of production. This condition has recently formed the subject of an investigation by the Commissioner of Labor, the Hon. Carroll D. Wright, and his results, so far as published, serve to show that the assemblage of material required per ton of pig-iron can be made with as little labor and expense, on the average, in this country as in any country of the world. Indeed, it may be asserted that in no other country can the quantity required for the production of 20,000,000 tons per annum be brought together so cheaply, if at all.

The most remarkable fact in this connection is the constant reduction in the cost of transportation, which has been mainly accomplished by the extension and improvement of the railroad-system of the country. It appears by the table herewith appended, that the average rate of freights on all classes of goods since 1882, has been reduced from 1.236 cents per ton per mile to 0.976 in 1889.

The following statement, from *Poor's Manual*, shows the volume of freight-traffic on all the railroads of the United States during the eight years, 1882–1889 :

Years.	Tons Freight moved.	Tons Freight moved one mile.	Average Rate Per Ton Per Mile.	Aver'ge Haul Per Ton.
			Cents.	Miles.
1882	360,490,375	39,302,209,249	1.236	109.02
1883	400,453,439	44,064,923,445	1.236	110.04
1884	399,074,749	44,725,207,677	1.124	112.07
1885	437,040,099	49,151,894,469	1.057	112.46
1886	482,245,254	52,802,070,529	1.042	109.49
1887	552,074,752	61,561,069,996	1.034	111.51
1888	590,857,353	65,423,005,983	0.977	110.72
1889	619,137,237	68,604,012,396	0.976	110.80

The rates on iron-ore, coal, limestone and pig-iron are probably not more than one-half of the average rate, because they are raw materials of the lowest class. This showing, which compares most favorably with the rates on European railways, is the more remarkable because it is accomplished in the face of a higher rate of wages, thus indicating that other elements besides wages paid, enter into the determination of final cost, and must be taken into account by economists and law-makers when they deal with the subject. The principal factor, however, in producing this desirable result has undoubtedly been the use of steel rails, due to the genius of Bessemer. His contribution to American prosperity will form the subject of consideration in another place, but it would be less than justice if we failed to record here that among all the agencies which have produced the phenomenal development of the United States during the last ten years, there is none which approaches in importance or is so far-reaching in its influence as the process which has enrolled the name of Sir Henry Bessemer among the great benefactors of mankind.

But when the materials, the men, the money and the skill have been brought together, it still remains to secure such harmony of action between labor and capital as will insure steadiness of employment and continuity of operations. The final answer to our inquiry as to the ability of the United States to supply the iron required for the continued progress of the country and the march of civilization throughout the world depends therefore upon the establishment and maintenance of friendly relations between the employers and employed engaged in the work of production. Otherwise, it is quite conceivable that no considerable addition can be made to the present annual product. Indeed, this consideration throws much light upon the fact that Great Britain, with abundance of fuel and with access to adequate supplies of foreign ores, has not been able to maintain the product which was reached in 1882. We are thus brought face to face with the most serious problem of our age, because if we cannot increase our output, the growth of wealth, which now increases in a higher ratio than the increase of population and is necessary to the amelioration of social conditions, must become stationary.

It cannot be denied that, throughout the world, the relations between capital and labor are far from satisfactory. They are undoubtedly undergoing a process of readjustment not unlike that which followed the abolition of serfdom at the close of the Middle Ages. During the process of evolution leading to a new era there must necessarily be unrest, agitation, sometimes violence and generally

severe loss on both sides, to the great detriment of society at large. The solution, when it comes, must be based upon justice; and it cannot come until public opinion is definitely made up as to the rights and duties of the contending parties, or until the contention shall cease to exist by the voluntary action of the combatants. Meanwhile, the severity of the struggle may be greatly mitigated and the final outcome accelerated if certain fundamental principles which have been established by the experience of mankind are kept steadily in view, and rigorously applied as each new complication shall arise. While the propositions which I shall state may be disputed by extremists, I think they will be generally regarded as axioms ingrained in the very constitution of human nature, and therefore to be accepted as standards of right and wrong to which all contentions may be referred.

I. Individual liberty consists in the right of each person to control his own life and to use the products of his labor in his own way, so long as he does not interfere with the equal rights of any other person.

II. Individual liberty implies the right of two or more persons to combine together and to use their property and faculties as they may see fit, so long as they do not interfere with the equal right of other individuals or combinations of individuals.

III. As population grows, there will necessarily be interferences among individuals and combinations of individuals, which must be adjusted; and hence the necessity for government and for tribunals whose judgment must be final.

IV. In countries where law expresses the will of the majority, and in which it can be amended as often as the majority may desire, there is no justification for resort to private or personal force in order to rectify wrongs, correct abuses, and maintain the rights of men. If the courts of justice have not adequate jurisdiction, it is the duty of the legislature which represents the public will to supply it, and all agitation should be directed to secure such legislation; and no man or set of men should be allowed to take the law into their own hands, to usurp the functions of the courts of justice, or to forestall the action of the legislature.

Bearing these axioms in mind, the following conclusions may be submitted as incontrovertible:

1. It is the equal right of employers and employees to make combinations among themselves respectively, or with each other to

advance or reduce wages, or to establish or resist legislation which either or both may regard as essential, desirable, or objectionable.

2. Neither party has the right to coerce the other into submission, except through the action of the courts or tribunals duly constituted to hear and decide upon causes of action submitted to them by either or both parties.

3. The right of workmen to refrain from labor and the right of the employer to cease to employ are correlative rights; but no one has the right to compel any other workman to cease from labor, nor has the employer any right to lock out his workmen in order to compel submission to obnoxious rules.

4. Strikes and lock-outs are therefore equally indefensible on the ground of justice, and can only be tolerated in the absence of provisions for the submission of grievances to the adjudication of competent tribunals.

5. No man has the right to compel another man to combine with him in any organization, and when a man declines to combine it is a violation of right to refuse to work with him, and to deny him the means of earning a living. It is equally wrong for employers to black-list men, so that others will not give them employment.

6. A boycott cannot be defended under any circumstances whatever. It is in effect a declaration of private war, which is a crime of the Hatfield-McCoy class, to be stamped out by prompt and severe punishment.

7. The claim of any body of men that under any circumstances they have the right to stop the operations of business by the issue of an order in the name of organized labor or associated capital cannot be tolerated. When such an order is given in regard to any railway or any other means of communication, it is a direct assault upon the common weal; and the failure to arrest and punish the offenders thus usurping the executive functions of the State and the judicial power of the Courts, is proof of cowardice on the part of the public officials and of degeneracy in public opinion, which excuses or permits the violation of the principle of the common law, that "not even the king can obstruct the highway."

And yet we live in a country and under a government professedly of law founded upon public opinion, in which all these abuses go unpunished. If they continue, disorders will increase, and capital will retire from business subject to such outrage and disturbance.

The iron business, as now organized, is a field in which capital and labor are brought into direct and immediate contact. It requires a

capital of at least one thousand dollars for each man employed. It has grown up under the wages-system, in which one party hires the other at an agreed price, and all the risks and profits of the business are assumed by the owners. Under the modern system of industry, its operations are conducted on a scale of such magnitude as to require the association of capital in corporate organizations which have almost entirely superseded private firms and ordinary partnerships.

As a rule the workmen have formed unions for the care of their interests, and especially to secure a satisfactory rate of wages. The formation of such unions is alike a right and a duty; and so long as they confine themselves to the assertion of the rights and the protection of the interests of their members they are to be commended and encouraged. The employers, on the other hand, have also various associations for the promotion of their own commercial interests, but no general organization, so far as I am informed, for the regulation of wages. Both sides are thus prepared for argument; and in this fact is to be found the starting-point from which may be readily reached the ground of conciliation and arbitration which ought to make strikes and lock-outs a memory of the past, to be recalled as a warning and not as a menace. In England, which has taught us how to make iron and steel cheaply and well, the system of voluntary arbitration has been in operation since 1869 and has worked, in the main, in a manner satisfactory to both parties, and with decided advantage to the public. Official arbitration, under the law which was passed in 1872, has not been found to be acceptable to either workmen or employers, and no case has ever arisen under the provisions of the law from which enthusiasts expected the most beneficent results. On the other hand, too much is not expected or attempted in the voluntary arbitrations which have sufficed to settle most of the disputes of the last twenty years.

It is admitted that the question of wages is fundamental, and that it can only be solved by the equal representation of both sides, with an umpire, whose decision shall be final after the fullest submission and discussion of complete information as to costs, sales, and the condition of the trade. Attempts of either side to get the better of the other by tricks and misrepresentation have long since ceased, so that when a result is arrived at, the award of the arbitrator is accepted by both sides as a satisfactory solution. Here it is obvious that three fundamental elements of conciliation have been evolved from the contentions which formerly resulted in strikes and lock-outs. The right of combination on both sides is admitted; the mutual

equality of both parties is conceded; and the right of both to be informed as to the actual condition of the business is acknowledged.

In view of such an example, and of the advantage of avoiding conflicts damaging alike to employers and employed, I am satisfied that we shall not be long in adopting a similar system of settling disputes by voluntary action, and that there will not be any disturbances serious enough to interfere with the rapid increase of product, which as we have seen, is required by the progress of our country.

It is manifest that this method of settlement involves publicity as to the profits of business. There is undoubtedly great reluctance and some ground of objection to the disclosure of cost and profits; but as a matter of fact, the transfer of business to large corporations has really made this information public property, and in the iron business there is no longer any pretence of concealment either from stockholders or competitors. Surely then, there remains no valid reason for denying to the workmen the information necessary to enable them to formulate reasonable demands; and it is to the interest of the owners to give this information, inasmuch as the margin of profit on manufacturing operations is now narrowed down to the smallest limits consistent with a moderate return on the capital employed. There is so much misapprehension on this point in the public mind that I am impelled to say that in the great staples of trade it is exceedingly difficult to get an adequate return for the capital employed, and the business is often conducted for long-continued periods on a basis which insures only wages for labor, without any return whatever for capital. Where large profits are realized, they are due either to the production of specialties covered by patents, or to the possession of raw material under exceedingly favorable conditions of cost or locality.

There is no feature in the business more pronounced than the excessive competition which cuts down profits to a minimum, and hence attempts have been made to control product and prices, through combinations looking to the maintenance of standard prices, and, in some cases, by the reduction of the output. In the public mind such arrangements are confounded with trusts, which have been the subject of so much recent criticism and denunciation. The objection to trusts is not to be found in the magnitude of their operations. This, in the modern development of industry, is unavoidable, and constitutes, in fact, an advantage to society by insuring lower prices and better quality, and to the workmen by providing the best appliances for labor and arrangements for the preservation of health and

the increase of comfort. It is only when trusts attempt to create a monopoly and succeed in destroying competition that they become injurious to the public welfare. It is extremely doubtful whether it is possible to maintain in this country an effective monopoly of any staple product of industry, but whether possible or not in other branches, the iron business is too widely diffused and is too vast in extent to admit any monopoly not sanctioned by law. The concentration of business, however, in special localities and the consolidation of interests in order to secure efficiency of administration, is a public benefit. The greater the organization and the larger the capital employed, the more certain it becomes that the business will be steadily prosecuted, thus avoiding the greatest evil under which workmen suffer—lack of constant employment. The principle of association developed in great industrial corporations is therefore altogether beneficial, and should have the hearty sympathy of the public and especially of the labor-organizations.

In any previous period of history such vast establishments might have been converted into devices for oppressing the workman, and for preying upon society by excessive prices; but in the presence of powerful labor organizations, whose right to demand information and whose power to obtain justice is now conceded, no oppression is possible, and no exaction can be continued under the scrutiny of an omnipresent and omniscient journalism. Society has therefore nothing to fear from the growing tendency of workmen to form unions, and of capital to centralize in great industrial corporations. But society has a duty to perform in the enactment of legislation which will regulate these organizations by a clear definition of their respective rights and duties.

Publicity, inspection and discussion are the great safeguards which the public can apply, in order to correct abuses and avoid conflicts and disastrous losses. The discouraging feature of the time is that the legislative department has shown not merely indifference but abject cowardice in dealing with the questions which from time to time require the interpretation of the law. Some of the legislation which has been recently enacted is a positive violation of the fundamental axioms which I have ventured to lay down, and of the provisions of the Constitution in reference to the liberties of the citizens, which are quoted in the outset of this address; but the greatest evil is the failure to legislate at all with reference to interferences which result in constant conflict, to the great injury of the public. What we need, therefore, is a recurrence to the well-set-

tled principles of jurisprudence, a higher order of statesmanship, and the courage on the part of our public men to stand up for the right, though for the time it may involve the sacrifice of personal popularity.

The course of procedure is clear. All organizations which avail themselves of the provisions of the law for the creation of corporations, should be required to report the result of their business and be open to the inspection and scrutiny of public officers appointed for the purpose. This principle is already recognized and enforced with reference to savings and other banks, insurance and trust companies, and railway corporations. It has not yet been applied to industrial organizations; but these now exist on so large a scale and employ so many men, disputes with whom affect the public convenience and interests so seriously, that every safeguard should be applied to prevent the disturbance and dislocation of industry. Publicity as to profits and losses would at once remove the most serious cause of strikes, which often take place when it is impossible for the employer to concede the demands of his men, because his profits will not warrant the concession. With proper information, the intelligence of the workmen may be relied upon not to make an issue which can only result in failure.

It will not be necessary to give any compulsory power of rectification to the officers charged with the duty of inspection. No real abuses can survive the criticism of the press when they have been fully investigated by an impartial tribunal. No strike can then succeed, unless it is based upon an abuse recognized and reported as a positive grievance by competent authority. All trade regulations and the rate of wages can then be safely left to voluntary agreement between the representatives of masters and men, sitting as equals on a board of conciliation, and presided over by an arbitrator who has the confidence of both.

Violations of the fundamental principles of society should be made crimes to be promptly punished. The legislature will readily respond to sound public sentiment in this respect; and a stern enforcement of the law is the best security for peace and order.

With industry under the control of great corporations endowed with adequate capital, with the workmen thoroughly organized to protect their rights and advance their interests, with proper public inspection and publicity as to the condition and results of the business, with legislation covering the grounds of conflict, and with the co-operation of the judicial arm clearly expounding and steadily

enforcing the law, it does not seem difficult to forecast the outcome of the evolution which is going on in the industrial world, and which seems to be full of promise and encouragement under the beneficent law which Edward Atkinson discovered, and which he and Robert Giffen have demonstrated, to wit: That labor is receiving a steadily-increasing share of a steadily increasing product; and that capital is receiving a steadily-diminishing share of an increasing product still insuring for it an adequate remuneration.

More than fifty years ago, John Stuart Mill laid down the proposition that when employers and employees had a common interest in the work, in the nature of a partnership, the means would exist of "healing the widening and imbittering feud between the class of employees and the class of capitalists." Since these words were written the feud has widened and the conflicts have become more frequent and more intense. On the other hand, the work of educating both employers and workmen has been going on in a bitter school of experience. Various attempts have been made to get the two classes together on some basis of organization which will make the remuneration of each directly and visibly dependent upon the profits of business. Under the existing system, wages are necessarily paid out of profits in the last analysis, but the rate and amount are not determined by the actual results from day to day. On the other hand, they constitute a prior lien upon the business, as well from necessity as now by law, and are thus exempt and guaranteed against the losses of the business.

The workman, however, fails to perceive that he is thus dependent upon the profits in order to get wages, and that he has the preference over all other claims upon the product of the business. Hence the sense of personal interest is lacking, and the success of the enterprise forms no part of the workman's current of thought. He has, in fact, no means of knowing the condition of the business, and his individuality is lost in the vast aggregation of energy which is combined in order to produce the results of modern industry. In England, it is notorious that the action of the trades-unions has been exerted in the direction of obliterating the individual to such an extent that special skill is rapidly declining, and in the finer grades of work it is almost impossible to find the experience required for the production of instruments of precision. This is a national evil of the first magnitude; and its disastrous consequences are becoming more apparent to the intelligent workman whose opportunities to rise in life are thus abridged and destroyed.

Slowly but surely, therefore, a new idea has been taking root in the industrial mind. Profit-sharing is getting to be a familiar thought both with employers and workmen, and many promising experiments in this direction are now in progress in this and other countries. The practice is to pay the current rate of wages in the usual manner, then to allow a reasonable percentage on the capital employed, and, if there be any excess after these payments, to divide it equally or otherwise between the capital and the labor, estimated by the amount of wages paid. The success of this system depends obviously upon the ability of the business to earn the current rate of wages. As this is not possible at all times, the employer must have sufficient capital to carry on business at a loss for a season, with the expectation of recouping the loss out of the future profits. It is idle to expect that workmen will be able or willing to refund losses, the risk of which must remain therefore, as it now does, with the employer. Hence the necessity and usefulness of the great organizations under the control of which the iron business is passing, by steady and irresistible progress. In such establishments the work of production will go on in bad as well as in good times, and the workman will be secured against the evils of intermittent employment.

But even this advantage is not sufficient for intelligent and ambitious men. Each man should be paid wages according to the value of his labor, and not on the mistaken basis of a dead level of mediocrity, advocated and enforced by some trades-organizations. Progress is only possible where the individual is encouraged to develop his skill and apply his labor, by a payment in proportion to the results achieved. But higher and beyond all this stands the stimulus of being engaged in a successful business and having a direct interest in its results. If the workman were a stockholder as well as a laborer working for wages, he would have such an interest; and this would tend to raise his self-respect as well as to develop his energies.

But profit-sharing, as it is called, will never be popular with the workmen, because, on the face of it, it is an act of grace from the employer. A self-respecting workman is not willing to accept charity. What he wants is justice, and any concession from the employer which does not recognize the right of the workman will be, and ought to be, rejected by independent and self-respecting men. When a workman, however, becomes a shareholder, either by payment for stock or by an agreement to pay for it out of his earnings, he stands on a level with the capitalist, and in fact, as well as in theory, is in a position

to feel that he is working for himself in doing his best to promote the success of the business in which he is engaged.

It should be a matter of congratulation, therefore, that the formation of trades-unions contemporaneously with the rapid growth of large corporations whose stock is divided into such small shares as to admit of easy distribution, clears the way for the new era when every intelligent workman will insist upon being an owner, and every well-managed corporation will see that its workmen are directly interested in the results of the business. To effect this desirable end, no compulsory legislation and no addition to the powers of corporations are needed. The educational influence of the conflicts which have occurred, has already done much, and the conferences which frequently take place as to wages and regulations, are doing more to establish a better understanding, to create harmonious action and to develop the idea that business cannot be carried on, unless both the capital and the labor employed share directly in the proceeds. The two classes are organized, as it were, into armies of observation, and occasionally they come into conflict, but the chances of collision are becoming daily smaller and will disappear altogether when their differences are merged in a sense of common ownership through the agency of corporations, admitting and cultivating the direct participation of the workmen in the profits.

It is, however, by no means necessary that all workmen should thus become shareholders. There will always be a considerable element of an unstable and unintelligent character, whose participation in the ownership is neither desirable nor possible; but I think the time is near when it will be discreditable to a workman not to be also an owner in the establishment in which he works, and that all workmen of the better class will have such an interest. It is quite conceivable that the workmen may ultimately acquire the preponderating interest, in which case the best possible solution will have been reached, in which labor hires capital at the lowest possible rate and thus becomes the main factor in the conduct of industry. This process can only succeed in establishments which have all the elements of success in the way of location and the possession of raw materials and of appliances for work. But such corporations, in the iron business at least, are so numerous as to offer abundant opportunity for the inauguration and successful application of this beneficent policy. I am glad to say that one of the greatest of our organizations, recently formed by the consolidation of several large establishments most favorably situated in all respects, the Illinois Steel Com-

pany, has made a promising beginning in the direction of interesting its workmen in its business. The outcome of this experiment will be watched with very great interest; and it may be commended to the attention of our guests as the most important and encouraging feature of our wonderful development, because it shows how the concentration of force under one management, in accordance with the modern tendency to centralization, may be made to solve, and must necessarily solve, the problem of harmonizing capital and labor engaged in the work of production, without new legislation or the application of any other than familiar and well-recognized principles of social organization.

The points which I desire to enforce by these arguments are:

First, that the industrial world has been steadily moving during the present century in the right direction for the welfare of mankind, and that the disturbances which have occurred have been necessary incidents of a beneficent evolution in the steady advance in the wages of labor, and in the distribution of the proceeds of industry upon the basis of equality and justice.

Second, that it is not necessary to invoke any new principles of government or to inaugurate any revolution in order that capital and labor may be associated together in peace and harmony. Progress is rather to be sought in diffusing a knowledge of the principles upon which government is founded, and by appropriate legislation framed in accordance therewith to meet the necessities of the complex relations arising out of advancing civilization and the unprecedented increase of riches in our day. A rigid enforcement of the laws thus formed is the necessary and sole condition for the maintenance of progress, peace and order.

Third, that the time is approaching when capitalists and laborers will more and more be joint owners in the instruments of production. That while the wages system will necessarily survive, the workmen will, to a large extent, become their own employers, and finally may hire capital as capital now hires labor. The facilities offered for the division of property, through the distribution of corporate shares, will lessen strife, develop skill, reduce cost, increase production, and promote the equitable distribution of wealth, which, it must never be forgotten, is the chief end of the social organization.

Fourth, that the invasion of government into the domain of industry must be met with uncompromising opposition. The proper function of government is supervision, regulation and adjudication. The work of production and distribution belongs to the citizen. Any

departure from this principle must result in the ruin of free government and in the substitution of despotism, the characteristics of which are communism, anarchism and nihilism.

Our contract-labor law is an example of the pernicious character of such interference. It affords probably the only instance in history since the expulsion of the Huguenots from France, in which the government has deliberately decided to deprive itself of the highest order of skill, by refusing to admit trained workmen, although it is still willing to receive ignorant and incompetent immigrants. If this law had been enforced fifty years ago, David Thomas, the first President of this Institute, whose advent to the United States has added untold millions to its wealth, would never have seen our shores. Read the contract under which this benefactor of our trade was induced to leave his home, and contrast it with the provisions of the law which fetters industry and deprives us of trained and useful citizens.*

*"Memorandum of agreement made the thirty-first day of December, 1838, between Erskine Hazard for the Lehigh Crane Iron Company of the one part and David Thomas, of Castel Ddu, of the other part.

"1. The said Thomas agrees to remove with his family to the works to be established by the said Company on or near the river Lehigh, and there to undertake the erection of a blast-furnace for the smelting of iron with anthracite coal and the working of the said furnace as furnace-manager, also to give his assistance in finding mines of iron-ore, fire-clay, and other materials suitable for carrying on iron-works, and generally to give his best knowledge and services to the said Company in the prosecution of the iron business in such manner as will best promote their interests for the term of five years from the time of his arrival in America, provided the experiment of smelting iron with anthracite coal should be successful there.

"2. The said Hazard for the said Company agrees to pay the expenses of the said Thomas and his family from his present residence to the works above-mentioned on the Lehigh and there to furnish him with a house and coal for fuel—also to pay him a salary at the rate of two hundred pounds sterling a year from the time of his stipend ceasing in his present employment until the first furnace on the Lehigh is got into blast with anthracite coal and making good iron, and after that at the rate of two hundred and fifty pounds sterling a year until a second furnace is put into operation successfully, when fifty pounds sterling shall be added to his annual salary, and also fifty pounds sterling per annum additional for each additional furnace which may be put into operation under his management.

"3. It is mutually agreed between the parties that should the said Thomas fail of putting a furnace into successful operation with anthracite coal, that in that case the present agreement shall be void and the said Company shall then pay the said Thomas a sum equivalent to the expense of removing himself and family from the Lehigh to their present residence.

"4. In settling the salary four shillings and sixpence sterling are to be estimated as equal to one dollar.

"In witness whereof the said parties have interchangeably set their hands and seals the date above written.

Witness

Erskine Hazard. [SEAL.]
For Lehigh Crane Iron Company.

David Thomas. [SEAL.]

But in condemning the interference of the Government in the actual work of production or distribution, let me guard against the inference that I am not in sympathy with the modern tendency of legislation to ameliorate the condition of the laboring class by suitable regulations of the hours of labor, by securing elementary and technical education, by improving the dwellings and providing for the general recreation of the masses. Neither do I object to the control by the Government of all functions which are of a general nature, such as the transmission of letters and the care of the public health. But even in these cases the general government should confine itself to administration and regulation, employing as far as possible the agencies provided by private enterprise.

The general tendency of the age is, however, in the right direction, and it cannot be arrested by a few temporary violations of sound statesmanship. The remedy will speedily be found when the workmen generally shall acquire a direct interest in the great industrial organizations of our day; and it is to this result that all intelligent and patriotic men should direct their efforts. The very simplicity of the plan may suggest doubts as to its efficacy, but all doubt will vanish, I am sure, if in our trade the proprietors and managers shall make an earnest effort to interest the workmen in the ownership of the property by making it easy for them to acquire shares upon the same terms as they can be purchased by capitalists. So certain am I of the disappearance of all strife when this diffusion of ownership shall become general, that I have been impelled to ask for this subject the thoughtful consideration of the representatives of the iron and steel industry of the world, now for the first time assembled in the country where the final development of this business must take place upon a scale of unprecedented grandeur.

If I have ventured to give an exceptional and unusual direction to this address, it is because I am fully persuaded that the conflict between capital and labor cannot go on without impeding, and finally paralyzing, the operations of the industrial world, and interrupting the continued progress of society in wealth, comfort and civilization. The present century, now nearing its close, has been pre-eminently an era of invention and of development in the forces of nature, enriching society and opening possibilities of general culture beyond the dreams of enthusiasm. Industrial peace is, however, necessary to the fruition of the hopes of a better adjustment of social relations, and of progress which will remove all privilege and all artificial impediments to the final establishment of equal rights. It is encour-

aging to think that this result can be reached without seeking for any new principles of government or introducing any new methods of legislation. *Natura viam monstrat.* We have no more reason to fear association than we have to dread competition, for they are the necessary and inseparable factors of progress. They are the agencies which have transformed the face of society during the present century. They are only in the infancy of their power, and no man can measure their potency in overcoming the evils which survive or which have been incidentally occasioned in the application of the natural forces in new directions. If we are careful to secure the maintenance and the application of individual energy, we have nothing to fear from association and combination. Participation in the ownership of the instruments of production and the agencies of distribution, rendered possible through the subdivision of the shares of the great corporations which control the domain of industry, will give the workmen who are employed in their conduct full scope for individual energy and the development of special skill in every department. The general distribution of shares is, therefore, to be encouraged as the true solution of the conflict between capital and labor, and may be relied upon to bring peace out of contention without resorting to the exasperating fallacies of communism, or the dangerous tendencies of class legislation, or to governmental interference with industrial pursuits.

APPENDIX.

I.

Table of Production of Leading Metals and Minerals in the United States during the First Century of National Independence.

Prepared by R. W. RAYMOND.

Republished for Comparison, from the Appendix to the Presidential Address of 1876, "A Century of Mining and Metallurgy in the United States," *Trans.*, v, 164.

	Anthracite, in tons of 2240 lbs. avoir.	Pig-iron, in tons of 2240 lbs. avoir.	Lead, in tons of 2240 lbs. avoir.	Copper, in tons of 2240 lbs. av.	Quicksilver in flasks of 76½ lbs. avoir.	Gold, in dollars, U. S. coin.	Silver, in dollars, U. S. coin.	Petrol'm, in barrels of 42 gallons.
1819	18,000*	.....	.....	.....	.....	.....	.....	.....
1820	1,965	.....	.....	.....	.....	.....	.....	.....
1821	3,273	.....	.....	.....	.....	.....	.....	.....
1822	4,940	.....	.....	.....	.....	.....	.....	.....
1823	9,023	.....	.....	.....	.....	.....	.....	.....
1824	13,641	.....	4,432*	.....	.....	.....	.....	.....
1825	38,499	.....	1,781	.....	.....	.....	.....	.....
1826	54,815	.....	1,771	.....	.....	.....	.....	.....
1827	71,167	2,178,239*	3,927	.....	.....	.....	.....	.....
1828	91,914	130,000	7,815	.....	.....	.....	.....	.....
1829	133,203	142,000	7,824	.....	.....	.....	.....	.....
1830	209,634	165,000	7,163	.....	.....	.....	.....	.....
1831	230,320	191,000	6,646	.....	.....	.....	.....	.....
1832	448,171	200,000	8,888	.....	.....	.....	.....	.....
1833	592,210	218,000	9,767	.....	.....	.....	.....	.....
1834	456,859	236,000	10,552	.....	.....	.....	.....	.....
1835	678,517	254,000	11,696	.....	.....	.....	.....	.....
1836	825,729	272,000	14,216	.....	.....	.....	.....	.....
1837	1,039,241	290,000	11,994	.....	.....	.....	.....	.....
1838	873,013	308,000	13,512	.....	.....	.....	.....	.....
1839	957,436	326,000	15,539	.....	.....	.....	.....	.....
1840	1,008,220	347,000	15,000	.....	.....	.....	.....	.....
1841	1,115,045	200,000	18,171	.....	.....	.....	.....	.....
1842	1,286,618	230,000	21,586	.....	.....	.....	.....	.....
1843	1,478,926	312,000	21,000	.....	.....	.....	.....	.....
1844	1,899,805	394,000	22,000	2,680*	.....	.....	.....	.....
1845	2,352,984	486,000	26,500	100	.....	.....	.....	.....
1846	2,707,321	765,000	25,000	150	.....	.....	.....	.....
1847	3,327,155	800,000	25,000	300	.....	20,000,000*	.....	.....
1848	3,572,695	800,000	22,500	500	.....	10,000,000	.....	.....
1849	3,724,806	650,000	21,000	700	.....	40,000,000	.....	.....
1850	3,863,365	563,755	19,500	600	25,424*	50,000,000	.....	.....
1851	5,190,690	413,000	16,500	800	24,000	55,000,000	.....	.....
1852	5,725,148	540,755	14,000	1,000	20,000	60,000,000	.....	.....
1853	5,940,905	723,214	15,000	1,850	19,000	65,000,000	.....	.....
1854	6,846,556	662,216	14,000	2,250	27,000	60,000,000	.....	.....
1855	7,684,542	700,159	14,000	3,000	33,000	55,000,000	.....	.....
1856	7,999,767	788,515	14,000	4,000	30,000	55,000,000	.....	.....
1857	7,694,842	712,640	14,000	4,800	28,000	55,000,000	.....	.....
1858	7,864,230	629,552	14,000	5,500	31,000	50,000,000	1,000,000*	.....
1859	9,010,726	750,560	14,000	6,300	12,000	50,000,000	100,000	3,200
1860	9,807,118	821,223	14,000	7,200	10,000	46,000,000	150,000	650,000
1861	9,147,461	653,164	14,000	7,500	35,000	43,000,000	2,000,000	2,113,600
1862	9,026,211	702,912	14,000	9,000	42,000	39,200,000	4,500,000	3,056,606
1863	10,953,077	846,075	14,000	6,474	40,531	40,000,000	8,500,000	2,611,359
1864	11,631,400	1,013,837	14,000	6,518	47,489	46,100,000	11,000,000	2,116,182
1865	10,783,032	831,768	13,165	6,811	53,000	53,200,000	11,250,000	3,497,712
1866	14,233,919	1,200,199	14,342	6,978	46,550	53,500,000	10,000,000	3,597,527
1867	14,345,644	1,305,015	13,662	7,774	37,000	51,700,000	13,550,000	3,347,306
1868	15,810,466	1,431,250	14,636	9,467	37,000	48,000,000	12,000,000	3,715,741
1869	16,375,678	1,711,276	15,653	11,858	33,713	49,500,000	13,000,000	4,215,000
1870	17,819,700	1,696,429	15,922	12,650	29,546	50,000,000	16,000,000	5,659,000
1871	17,370,463	1,707,685	17,854	12,546	31,881	43,500,000	22,000,000	5,795,000
1872	22,032,265	2,539,783	23,106	11,948	30,306	36,000,000	25,750,000	6,539,103
1873	22,828,178	2,560,962	46,661	15,573	28,600	35,000,000	36,500,000	9,879,455
1874	21,667,386	2,401,261	53,219	17,548	34,254	39,600,000	32,800,000	10,910,303
1875	20,643,509	2,108,554	53,000	15,625	53,706	33,400,000	41,400,000	8,787,506
Total	341,521,423	40,000,000	855,000	200,000	840,000	1,332,700,000	261,450,000	76,594,600

* Including the whole previous period from 1776.

II.

Table of Production of Leading Metals and Minerals in the United States from 1876 *to* 1889 *inclusive.*

Prepared by W. B. KUNHARDT.

Year.	Anthracite in tons of 2240 lbs.	Pig-iron in tons of 2240 lbs.	Lead in tons of 2240 lbs.	Copper in tons of 2240 lbs.	Quicksilver in flasks of 76½ lbs.	Gold, in dollars, U. S. coin.	Silver, in dollars, U. S. coin.	Petroleum in barrels of 42 gals
1776 to 1875	341,521,423	40,000,000	855,000	200,000	840,000	1,332,700,000	261,450,000	76,594,600*
1876	19,611,070	1,868,960	57,210	18,000	75,074	39,900,000	38,800,000	9,175,906
1877	21,323,000	2,066,593	73,125	20,089	79,396	46,900,000	39,800,000	13,490,171
1878	18,275,000	2,126,785	81,303	20,759	63,880	51,200,000	45,200,000	15,163,462
1879	29,830,447	2,741,853	82,840	23,500	73,684	38,900,000	40,800,000	19,785,176
1880	27,115,000	3,835,191	86,450	27,000	59,926	36,000,000	39,200,000	26,032,421
1881	30,738,185	4,144,254	104,540	31,741	60,851	34,700,000	43,000,000	27,376,509
1882	31,358,264	4,623,323	118,652	40,913	52,732	32,500,000	46,800,000	30,053,500
1883	33,336,469	4,595,510	127,640	52,300	46,725	30,000,000	46,200,000	23,400,229
1884	33,175,756	4,097,868	124,908	64,831	31,913	30,800,000	48,800,000	24,089,758
1885	34,228,548	4,044,525	115,546	76,322	32,073	31,801,000	51,600,000	21,842,041
1886	34,853,077	5,683,329	121,097	71,980	30,068	35,000,000	51,000,000	28,110,115
1887	37,578,747	6,417,148	143,482	82,442	33,825	33,100,000	53,441,300	28,249,597
1888	41,624,611	6,489,738	*166,071	103,245	33,250	33,175,000	59,195,000	27,615,929
1889	37,532,172	7,603,642	†169,640	107,857	26,464	32,800,000	64,646,464	33,902,000
Total	772,101,769	100,338,719	2,427,594	940,979	1,539,861	1,839,476,000	929,932,764	404,881,414

* Including 28,636 tons imported in ores from Mexico.
† Including 19,000 tons imported in ores from Mexico.

III.

Bituminous-Coal Production.

Year.	Tons of 2240 lbs.
1875	26,031,726
1876	*27,569,081
1877	30,778,329
1878	31,525,322
1879	33,665,709
1880	38,768,000
1881	47,621,918
1882	60,860,000
1883	68,531,000
1884	73,730,000
1885	64,840,000
1886	65,810,000
1887	78,464,000
1888	87,535,000
1889	99,923,000

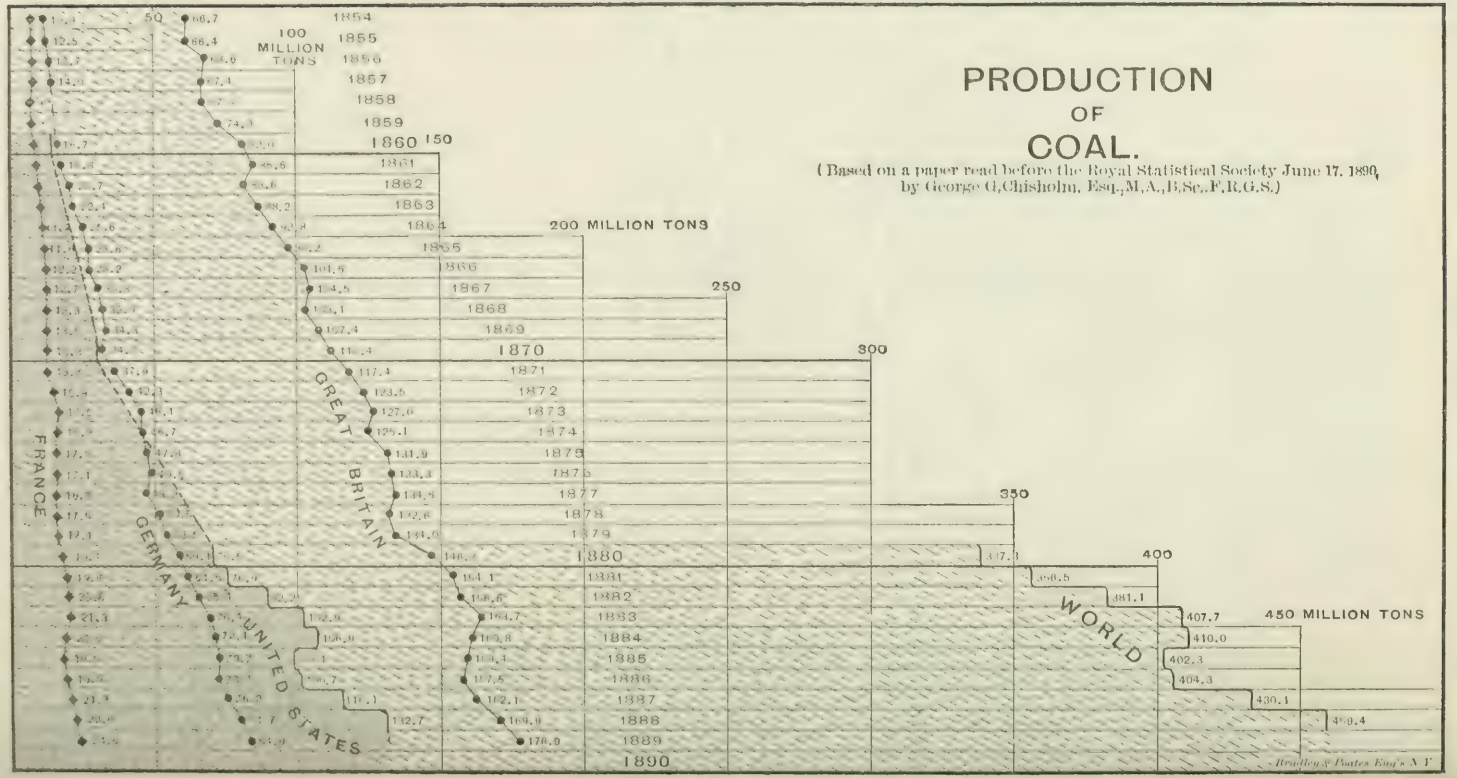
* The figures for 1876, 1877, 1878, 1879 and 1881 are estimates kindly furnished by Mr. Frederick Seward of the New York *Coal Trade Journal*.

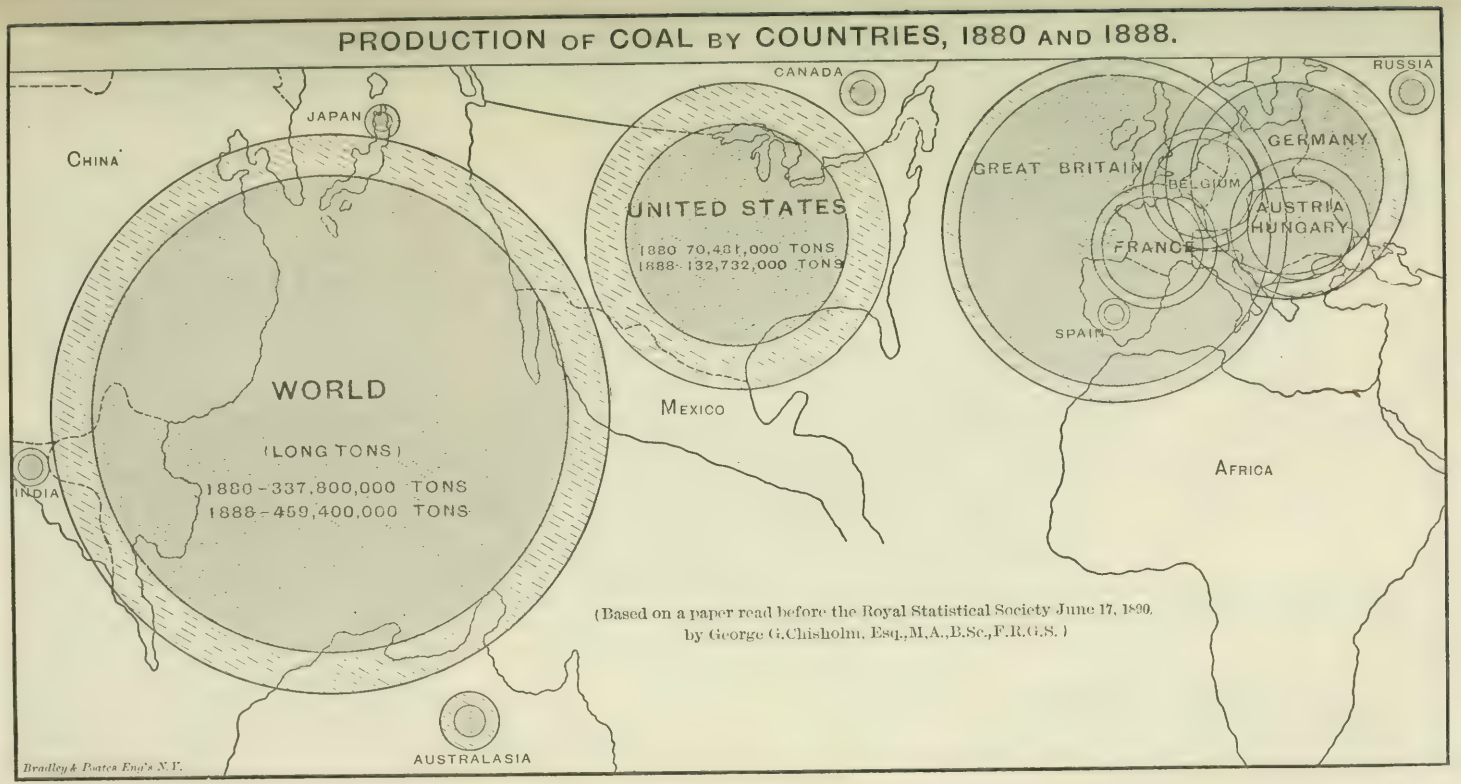
STATISTICAL CHARTS.

[Prepared by Mr. John Hughes, New York City, under the Direction of Mr. A. S. Hewitt, to Illustrate the Presidential Address on "Iron and Labor," New York Meeting, September, 1890.]

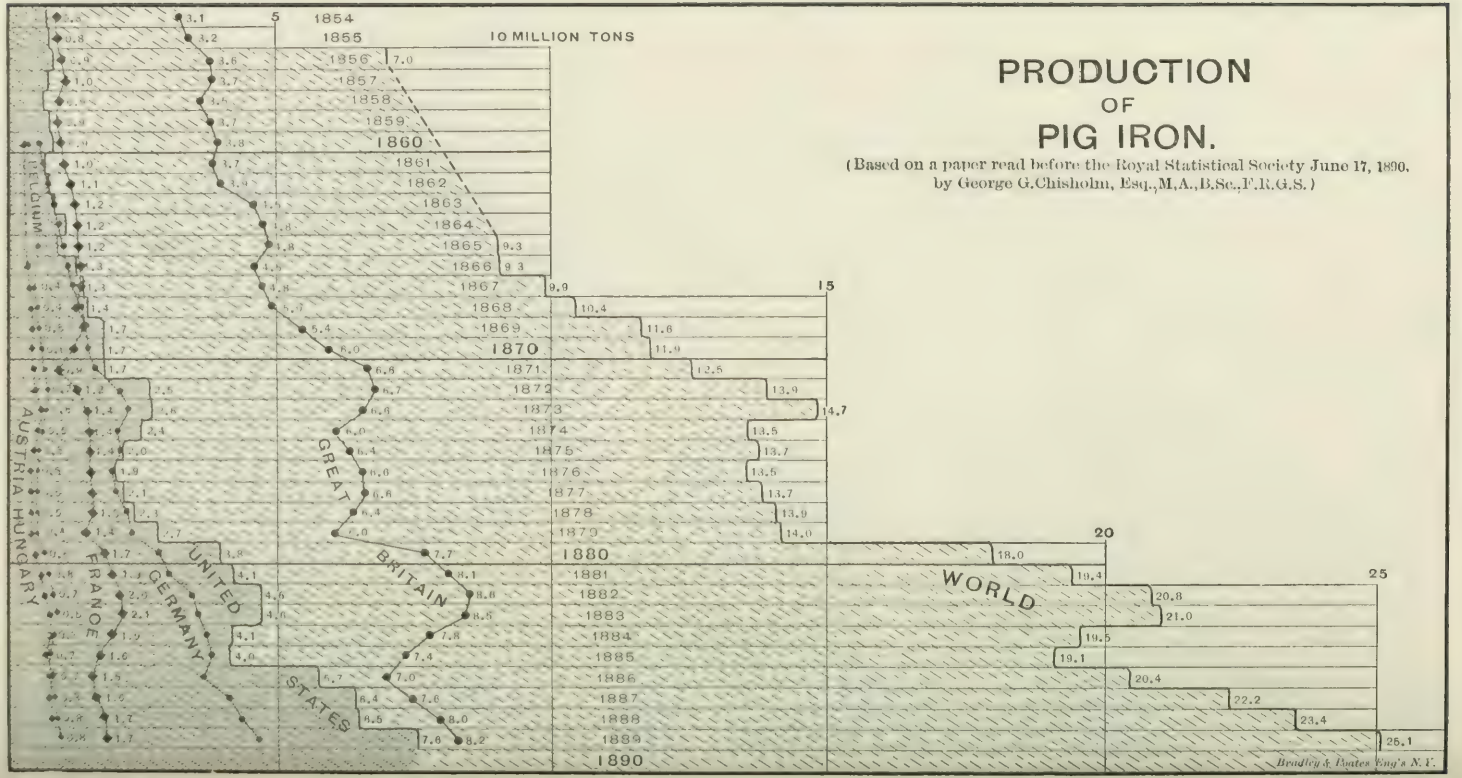
The following diagrams, reduced and engraved from a large chart prepared for use in Chickering Hall, on the occasion of the recent meeting, illustrate graphically the progress of the world and of the several leading countries in the production of coal, iron and steel, and the relation which this development bears to increase in wealth and population, and to the course of prices and the productivity of land.

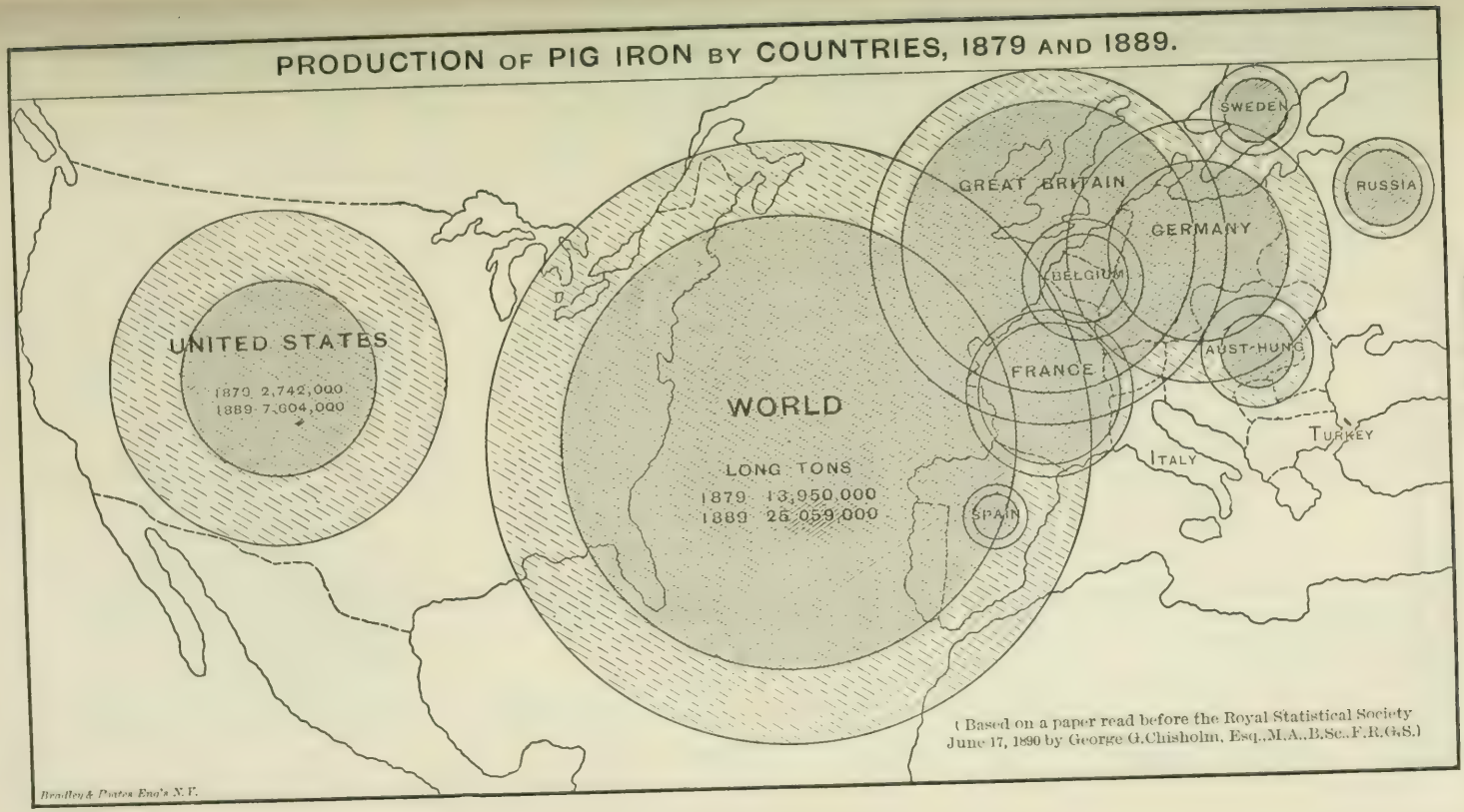
R. W. R.

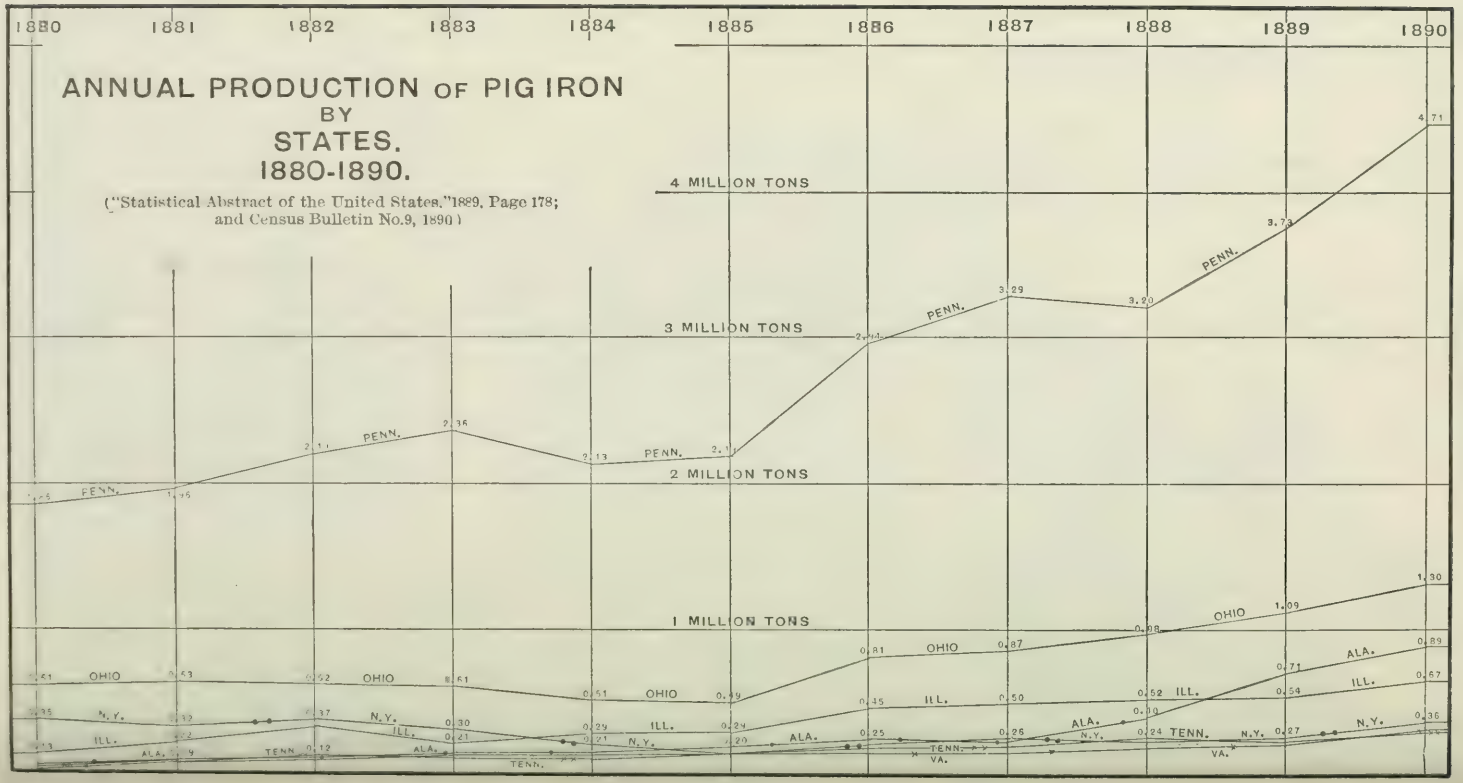

PRODUCTION
OF
COAL.
(Based on a paper read before the Royal Statistical Society June 17. 1890,
by George G.Chisholm, Esq.,M.A.,B.Sc.,F.R.G.S.)
50
100
MILLION
TONS
150
200 MILLION TONS
250
300
350
400
450 MILLION TONS
1854
1855
1856
1857
1858
1859
1860
1861
1862
1863
1864
1865
1866
1867
1868
1869
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
1888
1889
1890
FRANCE
GERMANY
UNITED STATES
GREAT BRITAIN
WORLD
Bradley & Bates Engrs N.Y.

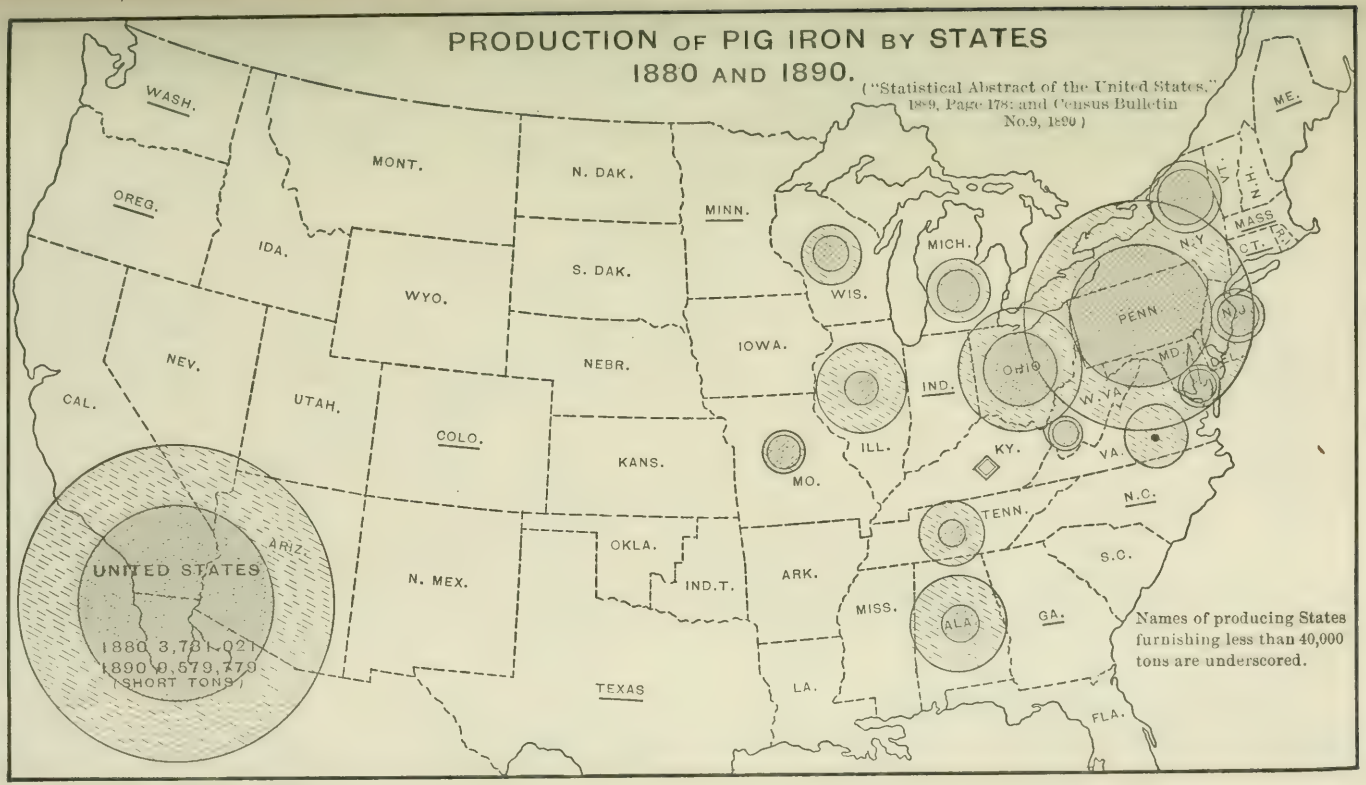


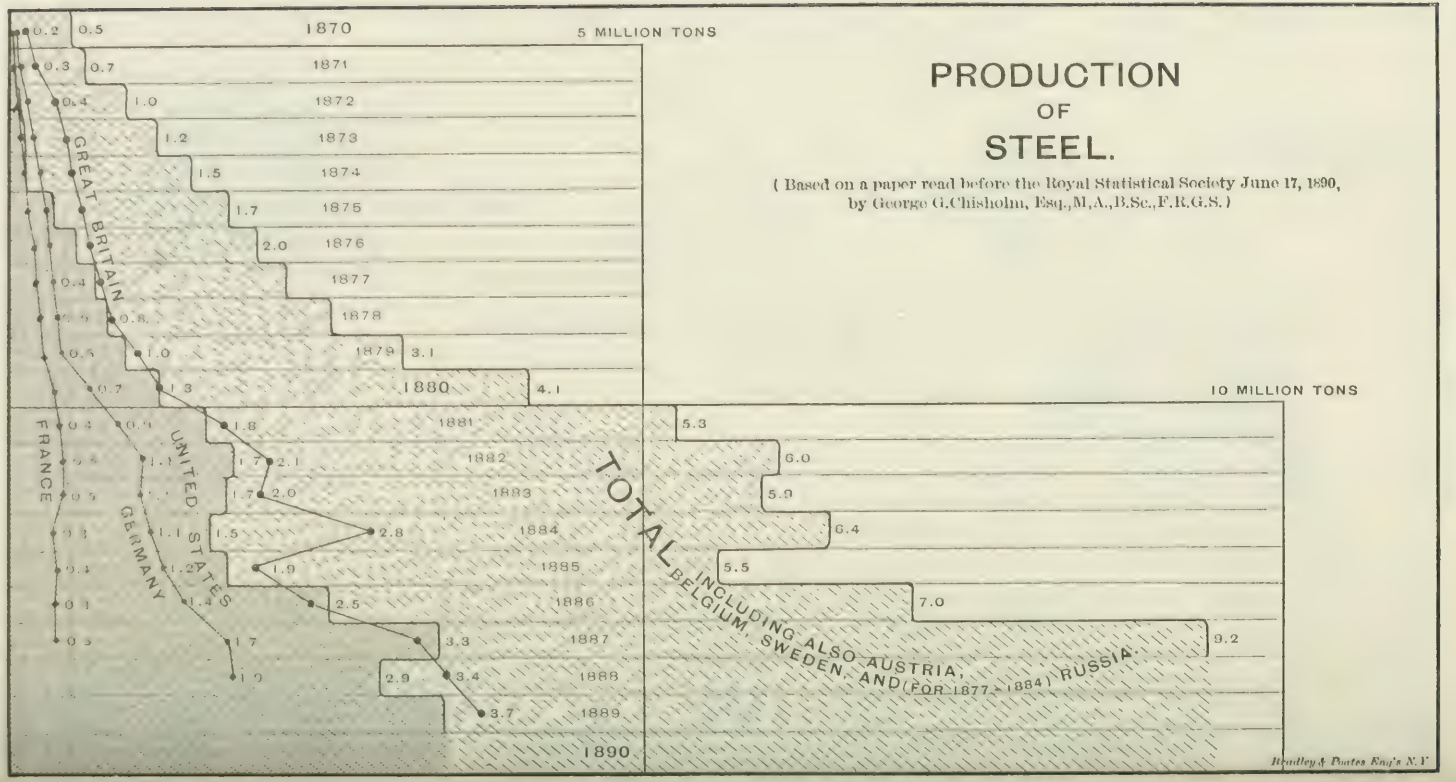
(Based on a paper read before the Royal Statistical Society June 17, 1890, by George G. Chisholm, Esq., M.A., B.Sc., F.R.G.S.)

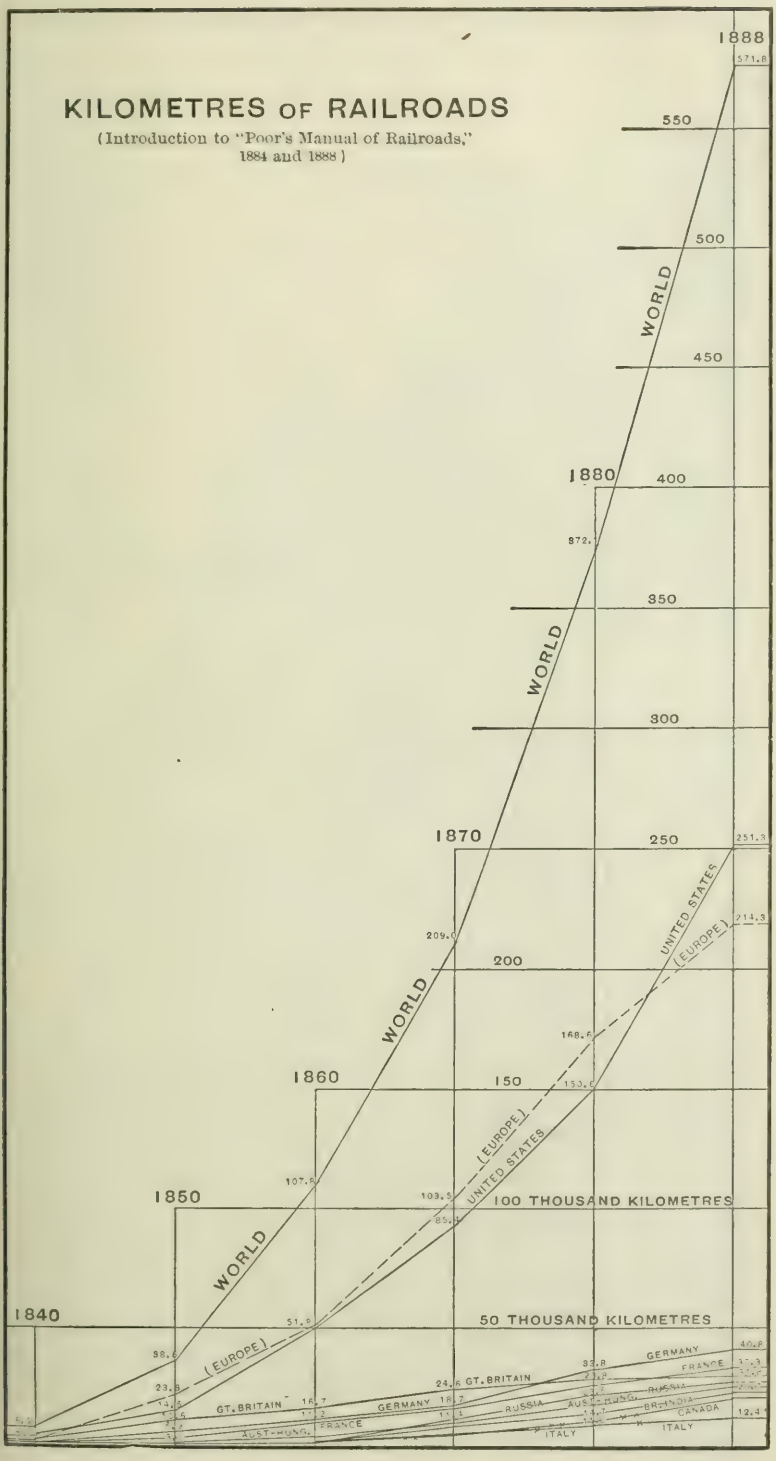

PRODUCTION
OF
PIG IRON.
(Based on a paper read before the Royal Statistical Society June 17, 1890, by George G.Chisholm, Esq.,M.A.,B.Sc.,F.R.G.S.)
10 MILLION TONS
5
15
20
25
1854
1855
1856
1857
1858
1859
1860
1861
1862
1863
1864
1865
1866
1867
1868
1869
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
1888
1889
1890
BELGIUM
AUSTRIA HUNGARY
FRANCE
UNITED STATES
GERMANY
GREAT BRITAIN
WORLD
Bradley & Coates Eng's N.Y.

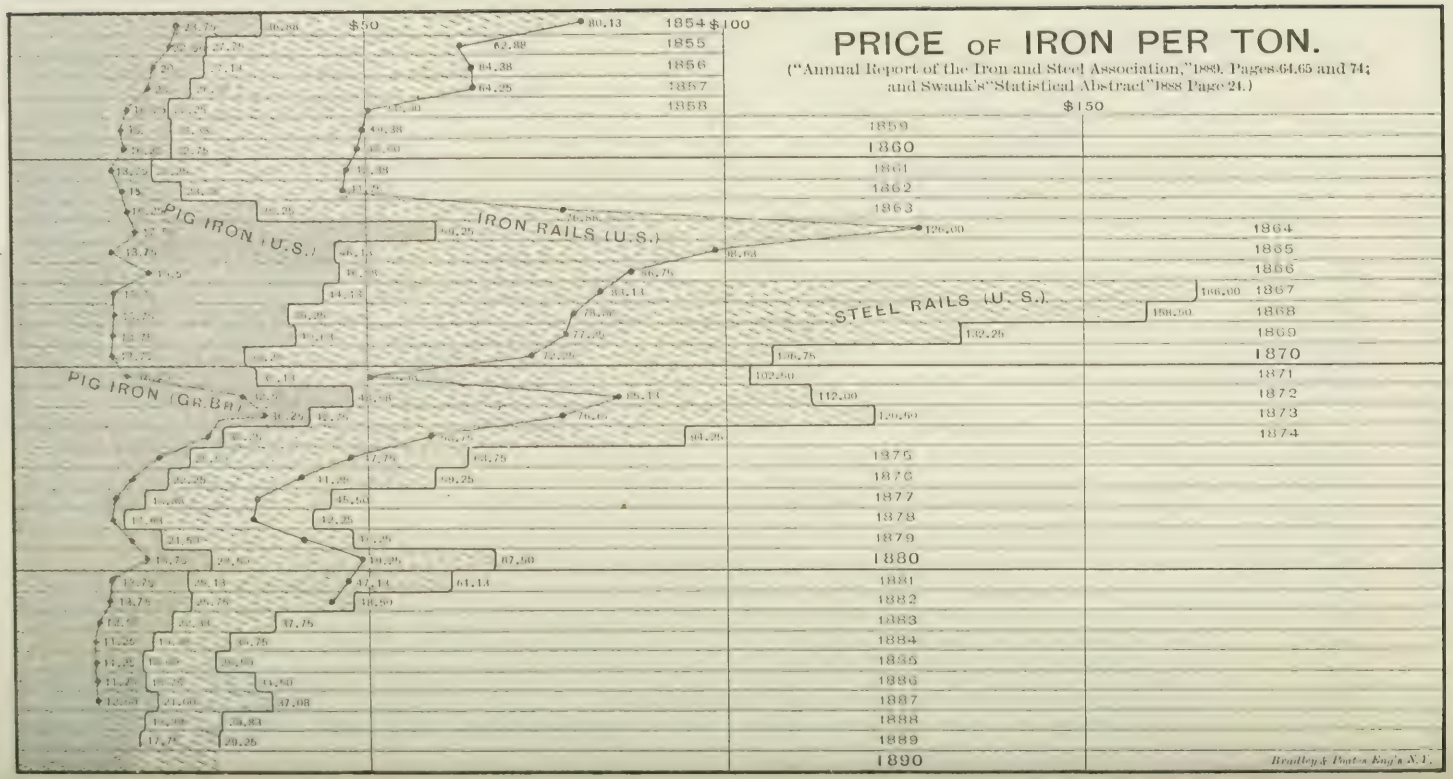

PRODUCTION OF PIG IRON BY COUNTRIES, 1879 AND 1889.
SWEDEN
RUSSIA
GREAT BRITAIN
GERMANY
BELGIUM
AUST-HUNG
FRANCE
TURKEY
ITALY
SPAIN
UNITED STATES
1879 2,742,000
1889 7,604,000
WORLD
LONG TONS
1879 13,950,000
1889 26,059,000
(Based on a paper read before the Royal Statistical Society June 17, 1890 by George G.Chisholm, Esq.,M.A.,B.Sc.,F.R.G.S.)
Bradley & Porter Eng's N.Y.

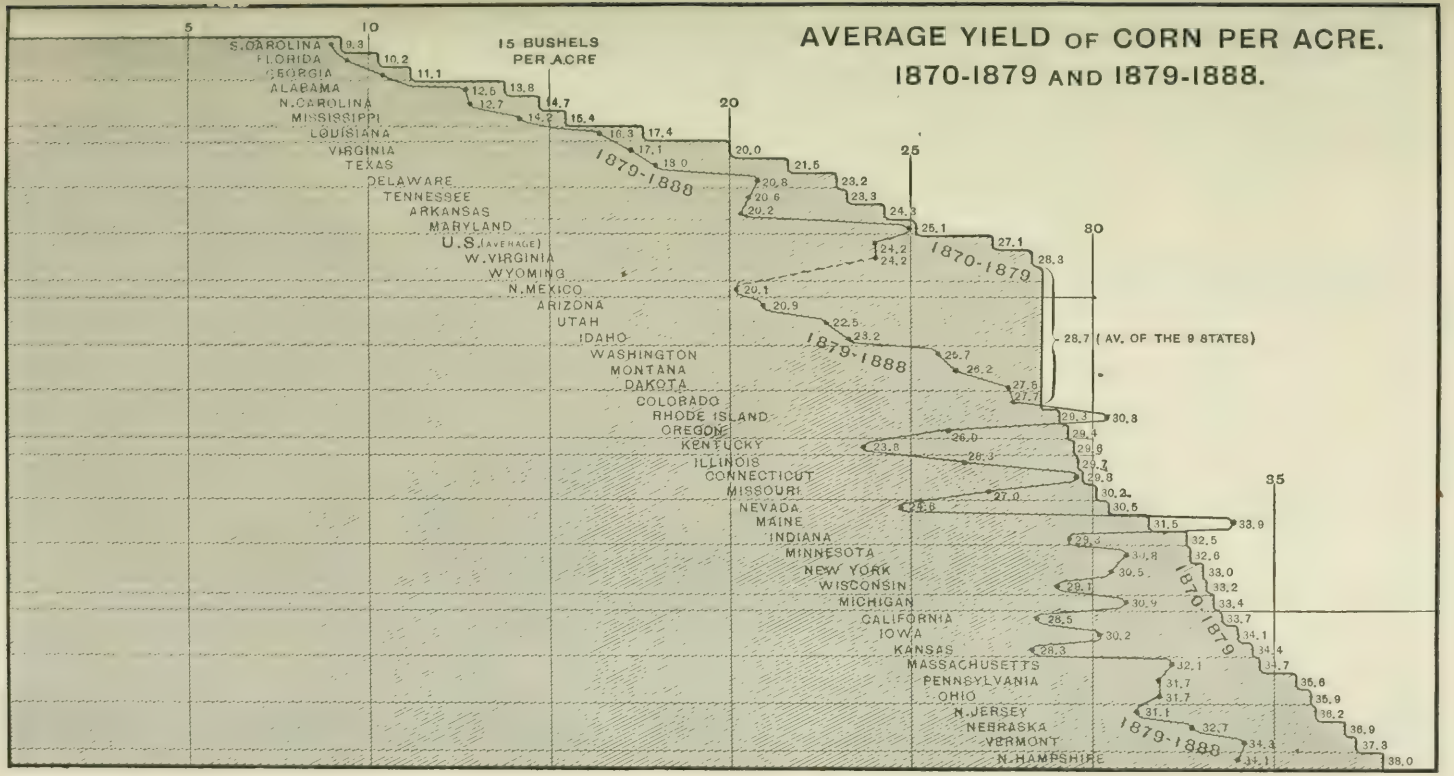

1880
1881
1882
1883
1884
1885
1886
1887
1888
1889
1890
ANNUAL PRODUCTION OF PIG IRON
BY
STATES.
1880-1890.
("Statistical Abstract of the United States," 1889, Page 178;
and Census Bulletin No.9, 1890.)
4 MILLION TONS
3 MILLION TONS
2 MILLION TONS
1 MILLION TONS
PENN.
OHIO
ILL.
N.Y.
ALA.
TENN.
VA.
4.71
3.73
3.29
3.20
2.13
2.36
1.90
1.09
0.89
0.87
0.81
0.71
0.67
0.63
0.61
0.54
0.52
0.51
0.50
0.49
0.45
0.39
0.37
0.36
0.35
0.30
0.27
0.25
0.24
0.21
0.20
0.12

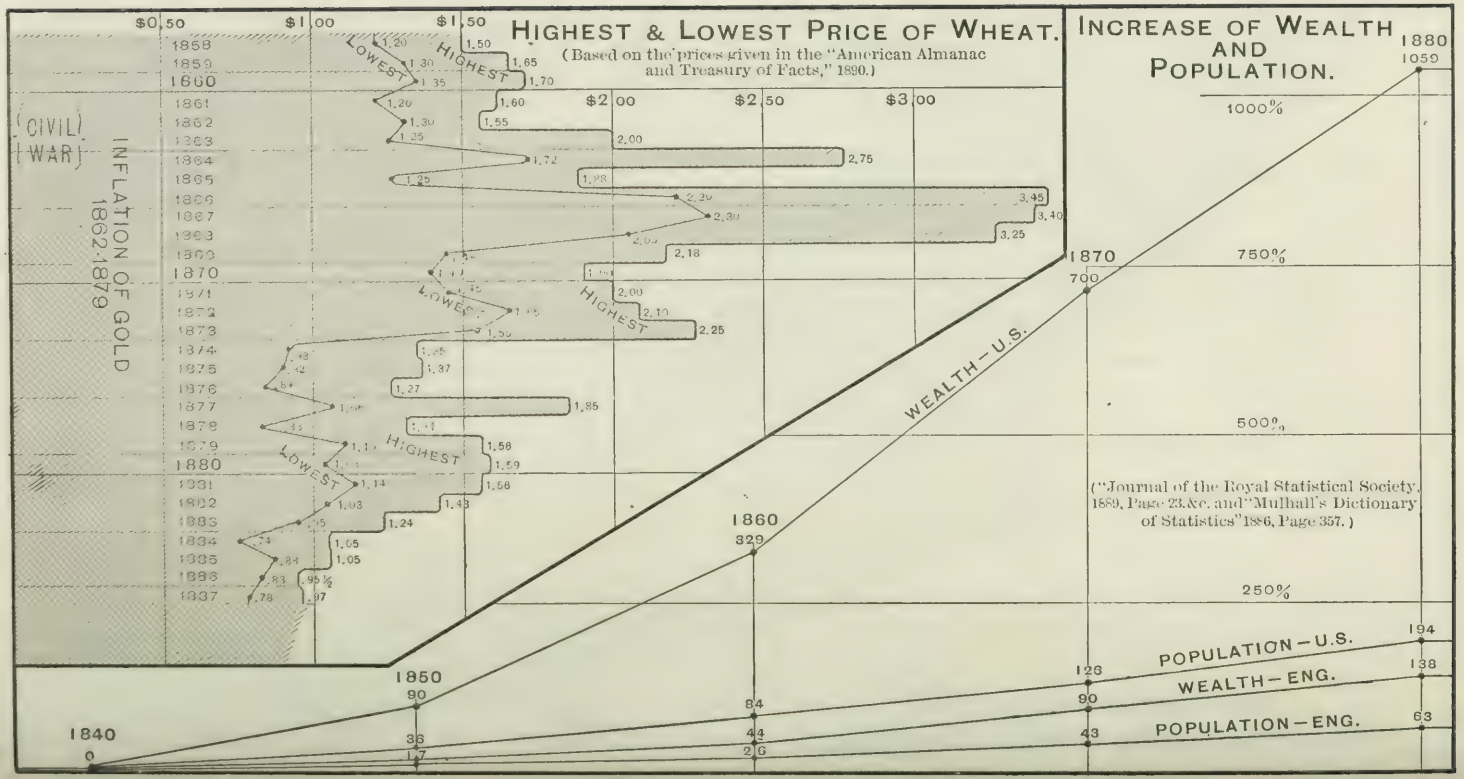

PRODUCTION OF PIG IRON BY STATES
1880 AND 1890.
("Statistical Abstract of the United States," 1899, Page 178; and Census Bulletin No.9, 1890)
WASH.
OREG.
CAL.
IDA.
NEV.
MONT.
WYO.
UTAH.
N. DAK.
S. DAK.
NEBR.
COLO.
KANS.
OKLA.
IND.T.
TEXAS
N. MEX.
ARIZ.
MINN.
IOWA.
MO.
ARK.
LA.
WIS.
ILL.
MICH.
IND.
OHIO
KY.
TENN.
MISS.
ALA.
GA.
FLA.
S.C.
N.C.
VA
W.VA.
PENN
N.Y
MD.
DEL.
N.J.
CT.
R.I.
MASS
VT.
N.H.
ME.
UNITED STATES
1880 3,781,021
1890 9,579,779
(SHORT TONS)
Names of producing States furnishing less than 40,000 tons are underscored.


PRODUCTION
OF
STEEL.
(Based on a paper read before the Royal Statistical Society June 17, 1890,
by George G.Chisholm, Esq.,M.A.,B.Sc.,F.R.G.S.)
5 MILLION TONS
10 MILLION TONS
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
1888
1889
1890
GREAT BRITAIN
FRANCE
GERMANY
UNITED STATES
TOTAL
INCLUDING ALSO AUSTRIA, BELGIUM, SWEDEN, AND (FOR 1877-1884) RUSSIA.
0.2
0.5
0.3
0.7
0.4
1.0
1.2
1.5
1.7
2.0
0.4
0.6
0.8
1.0
0.7
1.3
3.1
4.1
0.4
0.9
1.8
1.1
1.7
2.1
1.7
2.0
1.1
1.5
2.8
1.2
1.9
1.4
2.5
1.7
3.3
1.9
2.9
3.4
3.7
5.3
6.0
5.9
6.4
5.5
7.0
9.2
Bradley & Bates Eng's N.Y.


KILOMETRES OF RAILROADS
(Introduction to "Poor's Manual of Railroads," 1884 and 1888)
1888
571.8
550
500
450
WORLD
1880
400
872.
350
WORLD
300
1870
250
251.3
214.3
209.6
200
UNITED STATES
(EUROPE)
168.6
1860
150
153.6
(EUROPE)
UNITED STATES
1850
107.6
103.5
85.4
100 THOUSAND KILOMETRES
WORLD
1840
51.8
50 THOUSAND KILOMETRES
98.6
(EUROPE)
23.3
GT. BRITAIN
16.7
GERMANY
18.7
FRANCE
AUST-HUNG.
24.6 GT. BRITAIN
RUSSIA
33.8
GERMANY
FRANCE
AUST-HUNG.
RUSSIA
BRITANNIA
CANADA
ITALY
ITALY


PRICE OF IRON PER TON.
("Annual Report of the Iron and Steel Association," 1889, Pages 64,65 and 74; and Swank's "Statistical Abstract" 1888 Page 24.)
$50
$100
$150
PIG IRON (U.S.)
IRON RAILS (U.S.)
STEEL RAILS (U.S.)
PIG IRON (GR. BRI.)
1854
1855
1856
1857
1858
1859
1860
1861
1862
1863
1864
1865
1866
1867
1868
1869
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
1888
1889
1890
Bradley & Potter Eng's N.Y.


AVERAGE YIELD OF CORN PER ACRE.
1870-1879 AND 1879-1888.
5
10
15 BUSHELS PER ACRE
20
25
30
35
S. CAROLINA 9.8
FLORIDA 10.2
GEORGIA 11.1
ALABAMA 12.5
N. CAROLINA 13.8
MISSISSIPPI 12.7
LOUISIANA 14.7
VIRGINIA 14.2
TEXAS 15.4
DELAWARE 16.3
TENNESSEE 17.4
ARKANSAS 17.1
MARYLAND 18.0
U.S. (AVERAGE) 20.0
W. VIRGINIA 20.8
WYOMING 20.6
N. MEXICO 20.2
ARIZONA 21.5
UTAH 23.2
IDAHO 23.3
WASHINGTON 24.3
MONTANA 25.1
DAKOTA 27.1
COLORADO 24.2
RHODE ISLAND 24.2
OREGON 28.3
KENTUCKY 29.1
ILLINOIS 20.9
CONNECTICUT 22.5
MISSOURI 23.2
NEVADA 25.7
MAINE 26.2
INDIANA 27.5
MINNESOTA 27.7
NEW YORK 29.3
WISCONSIN 30.8
MICHIGAN 29.4
CALIFORNIA 26.0
IOWA 29.6
KANSAS 23.0
MASSACHUSETTS 28.3
PENNSYLVANIA 29.7
OHIO 29.8
N. JERSEY 30.2
NEBRASKA 27.0
VERMONT 30.5
N. HAMPSHIRE 25.8
31.5
33.9
29.3
32.5
34.8
32.6
30.5
33.0
28.1
33.2
30.9
33.4
28.5
33.7
30.2
34.1
28.3
34.4
32.1
34.7
31.7
35.6
31.7
35.9
31.1
36.2
32.7
36.9
34.3
37.3
34.1
38.0
1870-1879
1879-1888
28.7 (AV. OF THE 9 STATES)


HIGHEST & LOWEST PRICE OF WHEAT.
(Based on the prices given in the "American Almanac and Treasury of Facts," 1890.)
$0.50
$1.00
$1.50
$2.00
$2.50
$3.00
1858
1859
1860
1861
1862
1863
1864
1865
1866
1867
1868
1869
1870
1871
1872
1873
1874
1875
1876
1877
1878
1879
1880
1881
1882
1883
1884
1885
1886
1887
LOWEST
HIGHEST
(CIVIL WAR)
INFLATION OF GOLD 1862-1879
INCREASE OF WEALTH AND POPULATION.
1880
1059
1000%
1870
700
750%
500%
250%
1860
329
1850
90
1840
0
WEALTH—U.S.
POPULATION—U.S.
WEALTH—ENG.
POPULATION—ENG.
194
138
126
90
84
63
44
43
36
26
17
("Journal of the Royal Statistical Society," 1889, Page 23.&c. and "Mulhall's Dictionary of Statistics" 1886, Page 357.)

THE PRESENTATION OF THE BESSEMER MEDAL.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

Address of Sir James Kitson, President of the Iron and Steel Institute.

I have now the duty—the very pleasant duty—to perform, of presenting to the Hon. Abram S. Hewitt the Bessemer gold medal for distinguished services to the iron and steel trade. When this matter was brought under consideration in London it was well known that, as Mr. Hewitt would be the first to acknowledge, there is more than one man in America upon whom we would gladly have conferred this distinction, and to whom this merit might be justly attributed; but Mr. Hewitt has been long known to our distinguished members on the other side of the Atlantic as one of the most active minds in the investigation of new methods of manufacture, and one of the most enterprising metallurgists on this side of the Atlantic. Well, gentlemen, we proposed to Mr. Hewitt that he should receive this gold medal, but I think it is quite right, and is his due, that I should state that he at once declined to receive it; and he declined in a manner which rendered it impossible to any one but very persistent friends and very obstinate Englishmen to refuse. When we heard that our American friends had conferred once more upon Mr. Hewitt the distinction of President of the American Institute of Mining Engineers, and particularly with reference to our contemplated visit, and when we heard that he had been elected unanimously to that position, we felt that we had received the stamp of approval from this side.

Gentlemen, as I said before, Mr. Hewitt has long been known to many of us in England as an advocate of scientific education. A report which he made of the metallurgical products at the Paris Exposition in 1867 was one of the matters which drew very clearly and distinctly the attention of the English iron and steel trade to the necessity of further improvement in our technical education as to iron and steel; and it was one of those beginnings which led ultimately to the foundation of the Iron and Steel Institute. The development of the ideas which he there investigated undoubtedly led

our foremost minds to the foundation of this Institute; and therefore it is only a debt of gratitude which we are paying when we present to him this gold medal. But, gentlemen, his services to the iron and steel trade of America are very remarkable. I might take up again the point which Mr. Carnegie gave me yesterday, that this is a year and that this is the country of "firsts;" for, in 1856, very shortly after the announcement by Sir Henry Bessemer of his invention for the manufacture of steel at Cheltenham, Mr. Hewitt very quickly made inquiries; and the result of those inquiries was that the first experimental Bessemer converter in America was erected at the works of Messrs. Cooper & Hewitt in 1856. In fact, I believe that, with the usual rapidity of Americans, within sixty days from that announcement, a Bessemer converter was working in the States. Of course it is to be understood that that was an experimental converter. It was not the establishment of the Bessemer process in the States.

Well, then, I find further that Mr. Hewitt was the first to build an open-hearth furnace in the United States. He was concerned with the Martin patents, involving also the use of the Siemens' regenerative furnace, and the open-hearth furnace which he built for the manufacture of steel was the successful "first" of a multitude. To him is therefore due the initiative of the introduction of that process.

But not content with that, on the announcement of Mr. Snelus's invention for improved basic linings, Mr. Hewitt took an interest in that invention, which was the first step which led him to be associated with the Thomas-Gilchrist basic patents, and through his intervention that process also was first introduced into the United States.

I think, gentlemen, that I have given you a record which justifies our distinguishing him, and also ourselves, by enrolling him on the list of the recipients of the Bessemer medal. Of him as a man in New York it is not for me to speak. His integrity, his public spirit, his self-denial, are well known to you all. I think that I might rightly use toward him the words of one of your American writers, who says:

> You know him well; no need of praise,
> Or bonfire from the windy hill
> To light to loftier paths and ways
> The world-worn man we honor still.

Reply of Hon. Abram S. Hewitt.

Mr. President: No one can be more sensible than I am of the great honor conferred by the bestowal of the Bessemer medal, or more grateful for the felicitous and generous words in which you have given expression to the decision of the Council. Nevertheless, I am not able to accept this distinction, the great object of ambition in our profession, without a feeling of compunction, which justifies me in referring to the fact that when the decision was made known to me I asked the Council to reconsider its determination, and confer the honor upon some one who had contributed directly to the improvement of the processes which have brought the production of steel to its present state of perfection in this country. My own connection with the business, although life-long, has been rather in the direction of administration, and of the study of the economic conditions of the trade, and of legislation affecting its interests, than with technical details. I can make no pretensions to either the scientific or the practical knowledge which constituted the merit of the eminent men whom you have heretofore decorated with the Bessemer medal. Nevertheless, I am somewhat reconciled to your decision by the information that this particular medal is conferred out of the ordinary course, and has been provided by Sir Henry Bessemer in view of your visit to this country, in order to enable you to make a recognition of the wonderful development of the steel industry which has taken place during the last decade in the United States. Receiving it, as I do, with a profound sense of the honor which its possession implies, I feel that I hold it in trust, as it were, for the able engineers who have had to deal with the new processes for the manufacture of steel, and of whom you have seen fit to select me as the representative.

In one respect, however, I may have been sufficiently in advance of my associates to attract your favorable consideration. Mr. Bessemer read his celebrated paper describing the process of producing steel without fuel at the Cheltenham meeting of the British Association for the Advancement of Science, in the summer of 1856. An imperfect report of this paper was published in the journals of the day, and attracted my notice. The theory announced seemed to be entirely sound, and the apparatus simple and effective. I gave orders at once, without further information than that derived from the published report, to erect an experimental vessel for the purpose of testing the possibility of producing steel direct from the pig-iron. In the same year in which the paper was read, the experiment was

tried at the furnace of Cooper & Hewitt, at Phillipsburg, in New Jersey, and the result served to show beyond all doubt that the invention of Mr. Bessemer was one which could be successfully reduced to practice. The same difficulties, however, which confronted him showed themselves in this humble experiment, and the further prosecution of the matter was deferred to a more convenient season. We all know the obstacles which Mr. Bessemer succeeded in overcoming, and the marvellous ingenuity and wide range of knowledge which he brought to bear in perfecting his process. To-day, not yet thirty-five years from the time when the announcement was made, it may be said to have revolutionized the iron and steel trade of the world. The whole product of steel of all kinds made prior to his invention was insignificant. To-day, the production has reached 10,500,000 tons, being more than one-third of the whole consumption of iron in the world. It is still rapidly advancing upon the domain occupied by the ordinary iron of commerce, and it is quite evident that the day is not far distant when the use of that commodity will be restricted to special purposes, serving to remind us of processes which otherwise would have been consigned to history.

I do not propose to enlarge upon the practical application of the Bessemer process to the manufacture of steel; but, if you will bear with me, I think it would be well to direct attention to the effects of this invention upon the economic, social and political condition of the world. A very few considerations will serve to show that the Bessemer invention takes its rank with the great events which have changed the face of society since the time of the Middle Ages. The invention of printing, the construction of the magnetic compass, the discovery of America, and the introduction of the steam-engine are the only capital events in modern history which belong to the same category as the Bessemer process. They are all examples of the law of progress which evolves moral and social results from material development. The face of society has been transformed by these discoveries and inventions. It is inconceivable to us how the world even existed without these appliances of modern civilization, and it is quite certain that if we were deprived of the results of these inventions, the greater portion of the human race would perish by starvation, and the remainder would relapse into barbarism. I know it is very high praise to class the invention of Bessemer with these great achievements, but I think a candid survey of the situation will lead us to the conclusion that no one of them has been more potent in preparing the way for the higher civilization which awaits the coming century than the pneumatic process for the manufacture of steel. Its

influence can now be traced, although the future results are still beyond the reach of the imagination.

Its principal characteristic is to be found in its cheapness. Steel is now produced at a cost less than that of common iron. This has led to an enormous extension in its use and to a great reduction in the cost of the machinery which carries on the operations of society. The effect has been most marked in three particulars: First. The cost of constructing railways has been so greatly lessened as to permit of their extension into sparsely inhabited regions, and the consequent occupation of distant territory otherwise beyond the reach of settlement. Second. The cost of transportation has been reduced to so low a point as to bring into the markets of the world crude products which formerly would not bear removal, and were thus excluded from the exchanges of commerce. The practical result of these two causes has been to reduce the value of food-products throughout the civilized world; and inasmuch as cheap food is the basis of all industrial development and the necessary condition for the amelioration of humanity, the present generation has witnessed a general rise in the wages of labor, accompanied by a fall in the price of the food which it consumes. I think it would be a very modest estimate of the improvement in the condition of the working-classes as a whole to say, that, in the essential elements of comfort, the working-classes of our day are enabled to earn and to expend at least double the amount which was at their command in any previous age of the world. This result appears to me to be due very largely, if not altogether, to the economy in the agencies of production made by the cheap steel of the Bessemer process, and of the other inventions which have followed in its wake. These are material results, but they are accompanied with the slow but sure elevation of the great mass of society to a higher plane of intelligence and aspiration. No better evidence of this can be afforded than the association of workingmen together for the advancement of their moral and social condition. Troublesome as the trades-unions may have been, they indicate a step in advance which should be the subject of congratulation among all well-wishers of the race. I see nothing but good to come out of the modern tendency to association, and I hold it to be one of the chief glories of Sir Henry Bessemer that he has contributed more than any other living man to that condition of industry which compels all who are engaged in its conduct to combine on a scale unknown before his time in the work of economic production and equitable distribution.

The first striking result in the cheapening in the cost of the pro-

duction and transportation of food-products was felt in Great Britain, which is now compelled to import at least two-thirds of its consumption. The competition of our western wheat-regions with the products of India in the English market altered the whole condition of agriculture in the British Isles. The profitable raising of wheat practically became impossible, and the farmers who had depended upon it could no longer pay the rents stipulated in their leases. A general reduction of rent, therefore, became necessary, which of course reduced the income of the landlords. The aristocracy of Great Britain is a survival of previous conditions, depending for its existence upon the ownership of the land and the revenue derived from it. Hence a serious if not fatal blow at the domination of what may be termed the privileged class of Great Britain was struck, unintentionally, doubtless, by the invention of Bessemer. We have not yet seen the final result of the competition it has introduced, but enough is apparent to show that the structure of the British Government will necessarily undergo very serious changes, all tending to the transfer of power from those who own the land to the commercial, manufacturing and working-classes of the people. I think it is doubtful whether any event in modern times, of equal significance, has occurred. Sir Henry Bessemer has certainly been the great apostle of democracy, and although he may be inclined to disavow the claim, history will record the fact that he has been the most potent factor in the reconstruction of the British Constitution upon the basis, ultimately to be reached, of universal suffrage.

Turning from Great Britain to this country, the effects of the Bessemer invention have been even more pronounced and striking. The cheapening in the cost of transportation enabled us to increase enormously the sales of food-products in foreign markets. In accordance with the well-known law of commerce that a nation cannot sell without buying, our imports of foreign merchandise have been increased in a corresponding degree. Under our fiscal system, made necessary by the war for the Union, a revenue has been derived enabling us to reduce our national debt in twenty-five years from about four thousand millions of dollars to less than nine hundred millions of dollars at the present time, notwithstanding the payment of a pension-roll which now amounts to fully one hundred and twenty millions of dollars per annum. We can trace, therefore, directly to the Bessemer invention the ability to reduce our national debt, and finally to pay off the outstanding bonds at maturity. This proposition can easily be verified by examining the results of the operation of our railroads, by which it will appear that since 1870,

when Bessemer rails began to be largely used, the rate of transportation has been reduced about two-thirds, and an eminent authority has recently stated that the difference in a single year would now amount to one thousand millions of dollars, a very large portion of which is directly traceable to the greater durability of the track, due to steel rails and the capacity to haul increased loads, not only in the cars but in the train. I doubt whether it ever occurred to Sir Henry Bessemer to consider the effect of his invention in furnishing us the means of paying off our national debt, but it certainly ought to secure for him the gratitude of every American citizen; and I am glad to have the opportunity, on this occasion, to bring this obligation to the notice of my countrymen.

The third point to which I would call attention is the vast extension and new direction of commerce which has resulted from the construction of steel vessels. The size of these vessels has enormously increased, and the cost of operating them has been reduced in a corresponding degree, comparing very favorably with the reduction of cost upon land, which is about one-third of what it was ten years ago. The characteristic of modern commerce is the rapidity with which exchanges are made, and in the fact that all portions of the habitable globe are quickly reached. The commercial world has been converted into a vast clearing-house for the exchange of products. One country may sell to another more than it buys from it, or buy more than it sells, but the difference is counterbalanced by a corresponding sale and purchase from some other country. The balances are not paid in money, but are passed to the credit of each country in the general settlement which takes place in the banking centers of the commercial world. Thus the function of the precious metals is reduced simply to the payment of final balances which, in the course of any one year, are small in amount. The economy in exchange thus effected is largely due to the improvement in transportation, made possible by the general use of steel, aided by the telegraph and particularly by the submarine cables which now reach every part of the civilized world. The interdependence of the human race has thus been increased, and the possibilities of hostile action by war diminished in a corresponding degree. The name of Bessemer will, therefore, be added to the honorable roll of men who have succeeded in spreading the gospel of "Peace on earth and good will towards men," which our divine Master came on earth to teach and to encourage.

I have some hesitation, in conclusion, in referring to another point which seems to me to be required in order to complete this hasty

reference to the claims of Sir Henry Bessemer to the admiration and gratitude of mankind. It is to be hoped that out of the stupendous results of his genius he has acquired for himself an ample fortune. How large the amount may be I have no means of knowing, but if he should be in the receipt of even a small percentage of the annual saving to society, he would have the largest income of all men in the world. It is the fashion of the day to rail at the possessors of large fortunes, and there are people who imagine that they have been wronged by the existence of such aggregations of wealth. It is doubtful whether, in proportion to the total wealth of mankind, the portion which may be controlled by the few who are recognized as great capitalists is as large in our time as in other ages of the world. But even if the case were otherwise, no man can allege that he is not a positive gainer by the results of the Bessemer invention, or that he suffers loss or damage in any way from the fact that Bessemer has secured for himself a small portion of the benefits which his genius has conferred upon mankind. His example is of inestimable value, therefore, in showing that the existence of large fortunes, as a rule, is the evidence of benefactions vastly greater to the wealth of the world, by which all have gained and none have lost. Although there are undoubtedly striking exceptions to this general rule in the possession of fortunes due to accident or even to fraud, these exceptions only serve to call public attention to the real question which underlies the accumulation of wealth in private hands, namely, the mode in which it is used by the possessors. A great capital is a great blessing if it is employed in adding to the resources and advancing the civilization of the world. It is a curse only when it is used for demoralizing expenditures and vicious indulgence. The career of Bessemer, therefore, is admirable, not only in having added to the general wealth, but in the employment of his share of the proceeds of his invention for the spread of knowledge, the progress of industry, and the reward of efforts to promote the welfare of the race. He is now in the evening of his days, but he is also at the summit of his fame, for he has lived to see the marvelous fruits of his genius in the advancement of his fellow-men to a higher plane of comfort and intelligence than has been possible in any previous age.

I think, therefore, Mr. President, that I am justified in asking you to carry back to Sir Henry Bessemer the grateful regards of his kin beyond the sea, whose homes he has multiplied, whose country he has developed, whose burdens of debt he has lightened, and whose progress in all the arts of civilization he has placed upon a basis as durable as the material with which his name will ever be associated.

NOTES ON RECENT IMPROVEMENTS IN GERMAN STEEL-WORKS AND ROLLING-MILLS.

BY R. M. DAELEN, DUSSELDORF, GERMANY.

(New York Meeting, September, 1890.)

In recent years many new appliances have been introduced into German steel-works and rolling-mills for the purpose of improving working-methods, and of creating and maintaining increased production, and satisfying the ever-growing demand for metal of closely specified hardness, strength and form. One result of the improvements has been to aid very effectively in demonstrating the real superiority of ingot- over wrought-iron for almost every possible purpose, and in dispelling the last vestiges of prejudice unfavorable to the new metal.

If, nevertheless, no marked decrease in the number of puddling-furnaces has yet been noticed, the cause of their survival must be attributed to the greatly increased consumption of all kinds of iron-manufactures. Several large new works for the production of ingot-iron are now, however, in course of erection in the Rhenish-Westphalian district, and upon their completion a rapidly diminishing output of wrought-iron may be anticipated.

The present aim of all large works manufacturing ingot-iron is to work the molten blast-furnace pig into a finished product without any further fuel-consumption than is required to furnish steam for the rolling-mill. Though practice has established the feasibility of this principle, no works can yet point to its complete adoption, partly because of the limitations imposed by existing conditions and plant, and partly, also, on account of the highly-diversified character of the finished products.

The Bessemer-works were the first to feel the influence of the basic process; but of recent years a similar influence has affected the open-hearth, and the owners of works possessing both a converter- and an open-hearth plant are now seeking to arrange them for combined working, in charging as well as in casting. In small works the basic-lined reverberatory furnace is constantly gaining ground over the old puddling-furnace. The small basic converter, on the

other hand, has been erected at only a few places and is in actual operation at fewer still.

Opinions have long differed with regard to the best number and most suitable arrangement of basic converters for a given daily production. These varying views are partially explained by the rapidly improving efficiency of basic practice: the number of heats per day and the durability of the lining were, in the early days of the process, far inferior to the corresponding results of acid practice; and the improvements which experience has gradually developed are such as could scarcely have been anticipated.

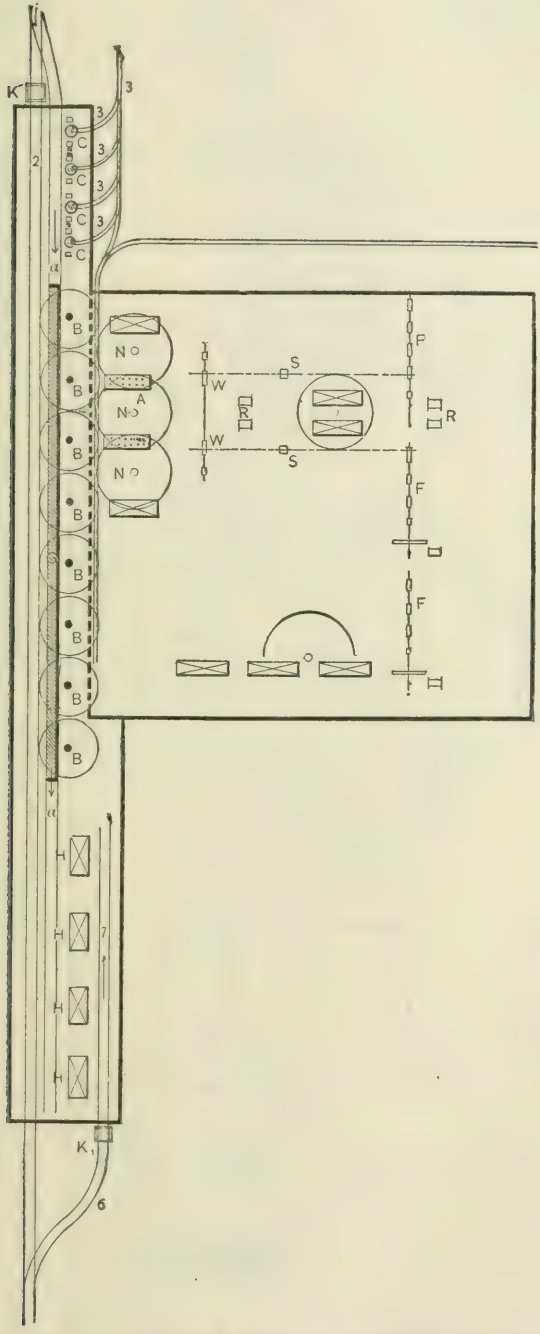
The system of removing and replacing the entire vessel has been adopted in isolated cases only. As soon as the fact seemed fairly well established that three 10- to 12-ton basic vessels would suffice for a daily production of 400 to 500 metric tons,* the circular arrangement of the vessels around a stationary, revolving ladle-crane was recognized as simplest in general plan, and was carried out in a number of works. Wherever this crane handles the charging-ladle, and also has to carry the steel-ladle while the metal is being poured from the converter, separate provision must be made for handling the steel-ladle during the cast. This may be done, either by a second stationary crane, or by a ladle-car which runs over a long casting-pit. The latter plan promotes to best advantage the separation of the converter-house from the casting-house, and commends itself particularly for a combined open-hearth and converter-practice.

Fig. 1 shows the general arrangement of a plant embodying these ideas. The four vessels, *C*, are set in line; adjoining them are the casting-pit, *G*, and the ingot-cranes, *B*, and beyond these, in the same direction, the open-hearth furnaces, *H*. The ladle of direct-metal is brought by track No. 1 on the general floor-level to the hydraulic lift, *K*. There it is raised to the charging-level of the converters. The track at this level may be laid either on the crane- or the chimney-side of the vessels. In the former position the supports of the track partially intercept the view of the vessel's mouth from the pulpit during the operation of pouring; in the latter position long runners for the pig-metal are unavoidable.

As, however, the advantage of emptying the charging-ladle by tapping, rather than by tipping, has come to be recognized, another, even more favorable arrangement would consist in providing each pair of converters with a hydraulic lift, placed between

* A metric ton has 2204.6 pounds, and equals 0.984 long ton.

FIG. 1.



General Plan of a Combined Converter and Open-Hearth Works,
Designed by R. M. Daelen.

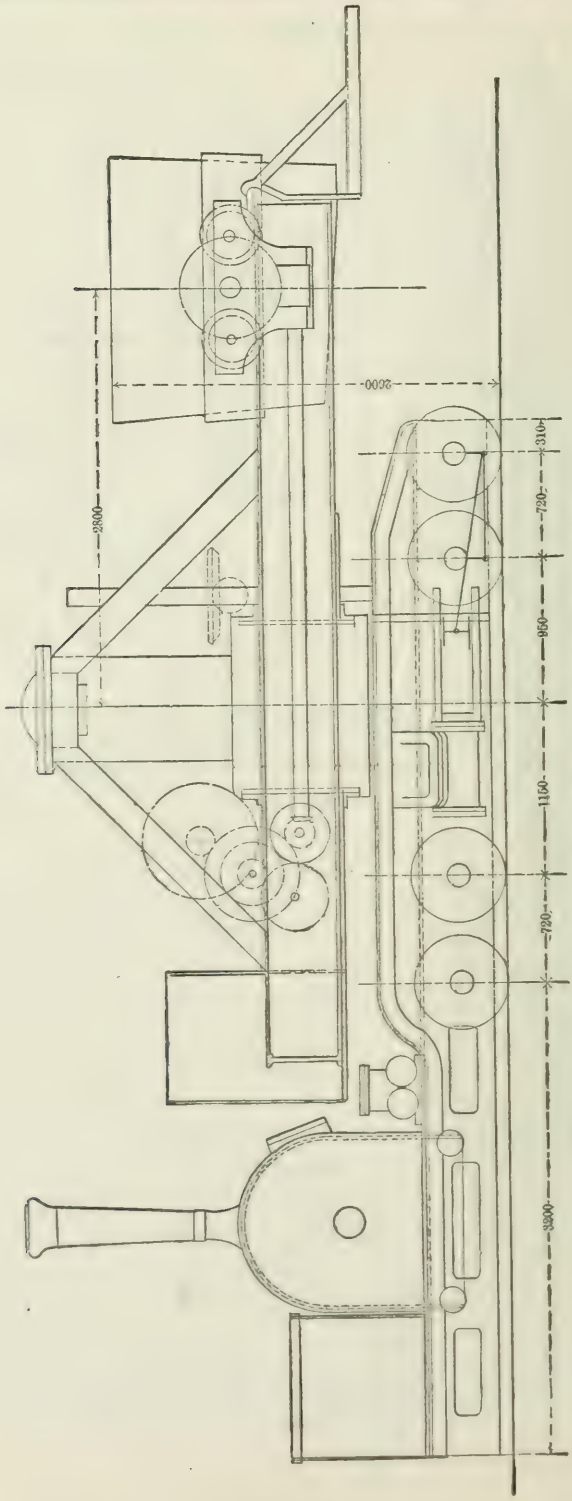


Fig. 2.

Side-Elevation of 12-Ton Locomotive Ladle-Crane.

Designed and Built by the Krauss Locomotive Works, of Munich, Bavaria.

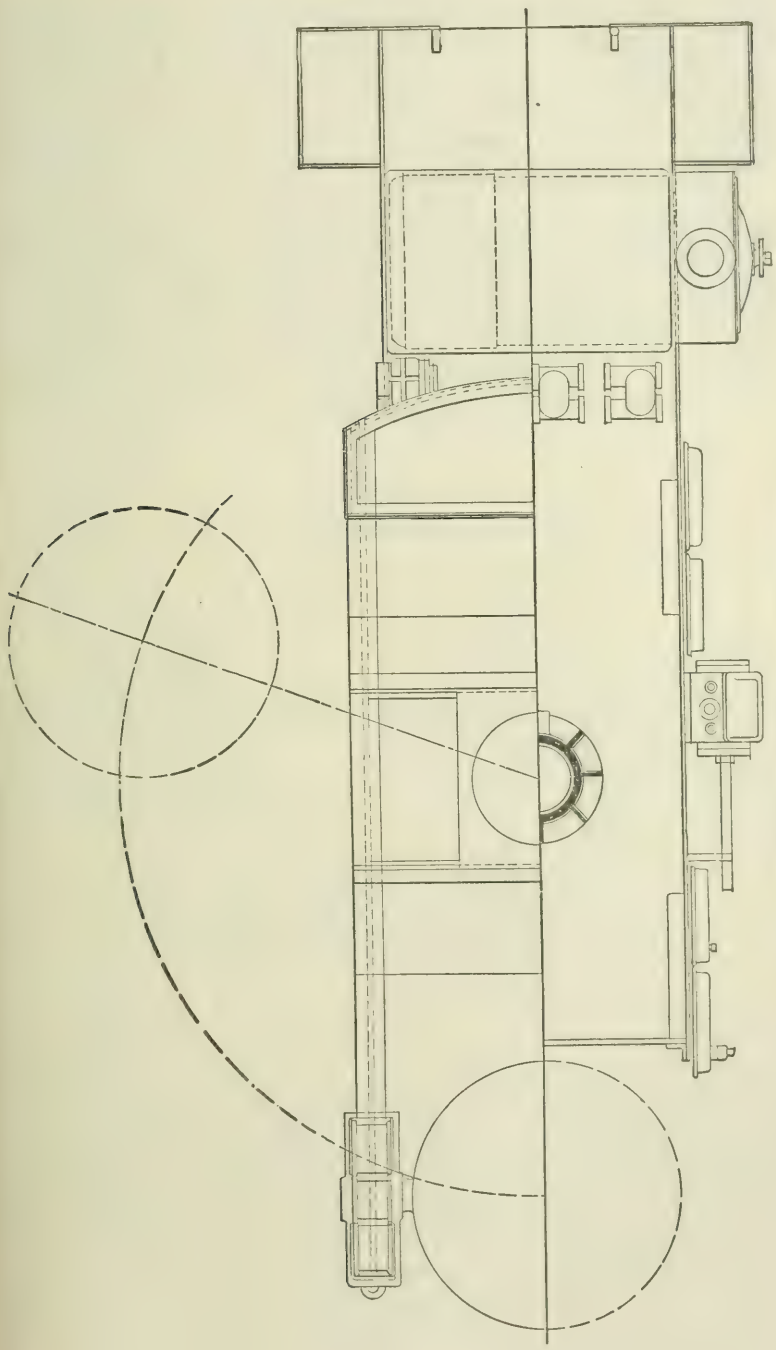


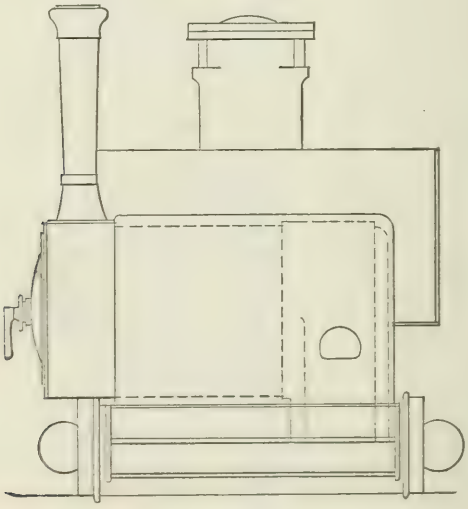
Fig. 3.

Plan of 12-Ton Locomotive Ladle-Crane.

them. This arrangement exists in several works outside of Germany. It permits the use of short runners (which would have to be provided for tapping in any case) and does away with the upper charging-track altogether. Track No. 1, on the general level, then serves with the necessary switches for the movement of both the pig- and the steel-ladles.

The design of the steel-ladle car has usually been determined by local conditions. A locomotive-crane (Figs. 2 to 5) is the customary type; the boiler, engine and pump are all mounted on the car, and the ladle is arranged to be swung and also lifted. A somewhat simpler, and on this account preferable, form of car is illustrated in Figs. 6 to 10. It dispenses with the vertical movement of the ladle, which experience has shown to be superfluous; and, instead of being

FIG. 4.



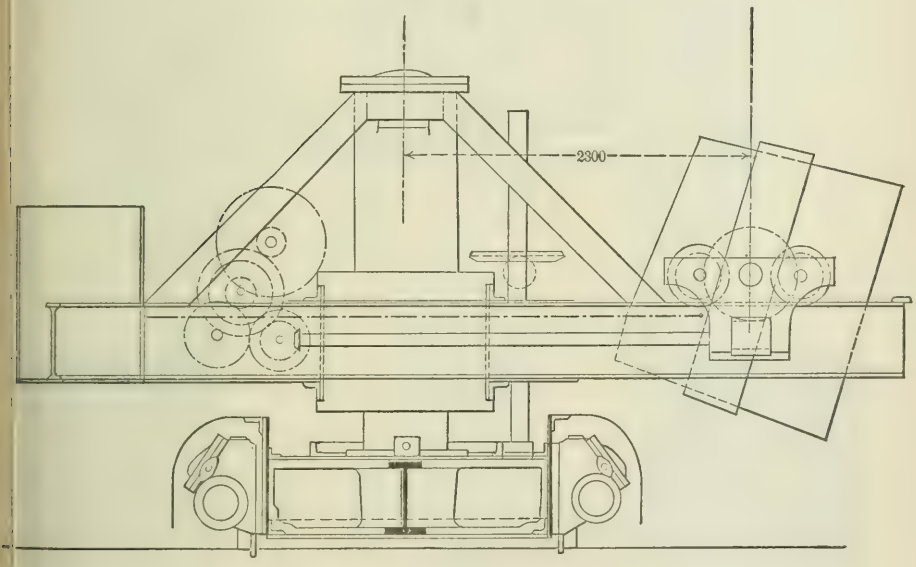
Front-Elevation of 12-Ton Locomotive Ladle-Crane.—Locomotive-End.

propelled by steam, it is moved by means of a hand-gear, *u*, or by an independent locomotive (which runs on the adjoining track No. 2, shown in the general plan, Fig. 1). The car is of wrought-iron. The ladle can be tipped, as shown at *n*, and swung in the arc *o*, to serve two lines of moulds. During the casting the car is always moved in the direction *m*, so as to clear the filled ingot-moulds. Care is taken to bring the tops of all the moulds to the same level.

A hydraulic cylinder is placed in the pit, out of the way of the moulds, and is fitted with a hooked plug which is used to force inward any obstruction that may form in the tapping-hole of the ladle.

Track No. 3 (Fig. 1) is used to convey the slag-cars from beneath the converters to the slag-grinding mill. Refractory materials are brought to the vessels over track No. 4, located on the first platform (Figs. 12 and 13). Lime and fuel are supplied by track No. 5 in the roof-truss. Both tracks, Nos. 4 and 5, are served by a hydraulic hoist, *A* (Fig. 12). The same service might be more cheaply performed by a wire-rope tramway, the first cost of which is certainly

Fig. 5.



Front-Elevation of 12-Ton Locomotive Ladle-Crane.—Crane-End.

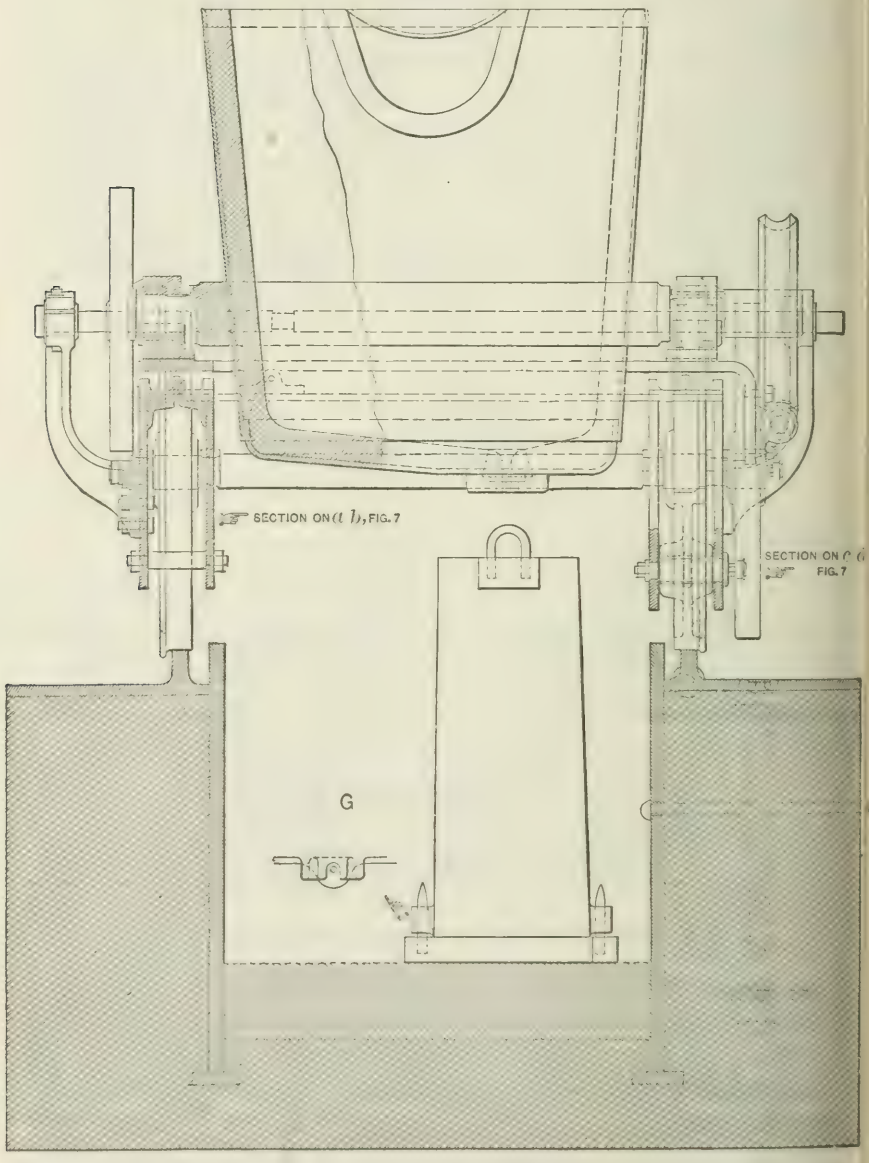
less than that of the track and hoist, while it offers the decided advantage of an automatic lime- and coke-supply to the converters.

For a plant embodying the above-described general features, a building of moderate height will answer. The roof can be set so low that the greater part of the converter-flame will be carried off on the outside, reducing interior smoke and heat to a minimum. This makes it practicable and advantageous to adopt the same form of roof for the converter- and the casting-house.

In the construction of the converter itself the use of a middle ring and journal, made in a single piece, has been quite generally

abandoned; for when this piece is made of cast-iron, even though it

Fig. 6.



Elevation and Vertical Half-Section of 12-Ton Casting-Ladle and Car,
Designed by R. M. Daelen.

be very heavy, it is not perfectly safe, and when made of wrought-

iron it is very costly. The ring is now commonly constructed of heavy plate, to which cast-steel journals are screwed.

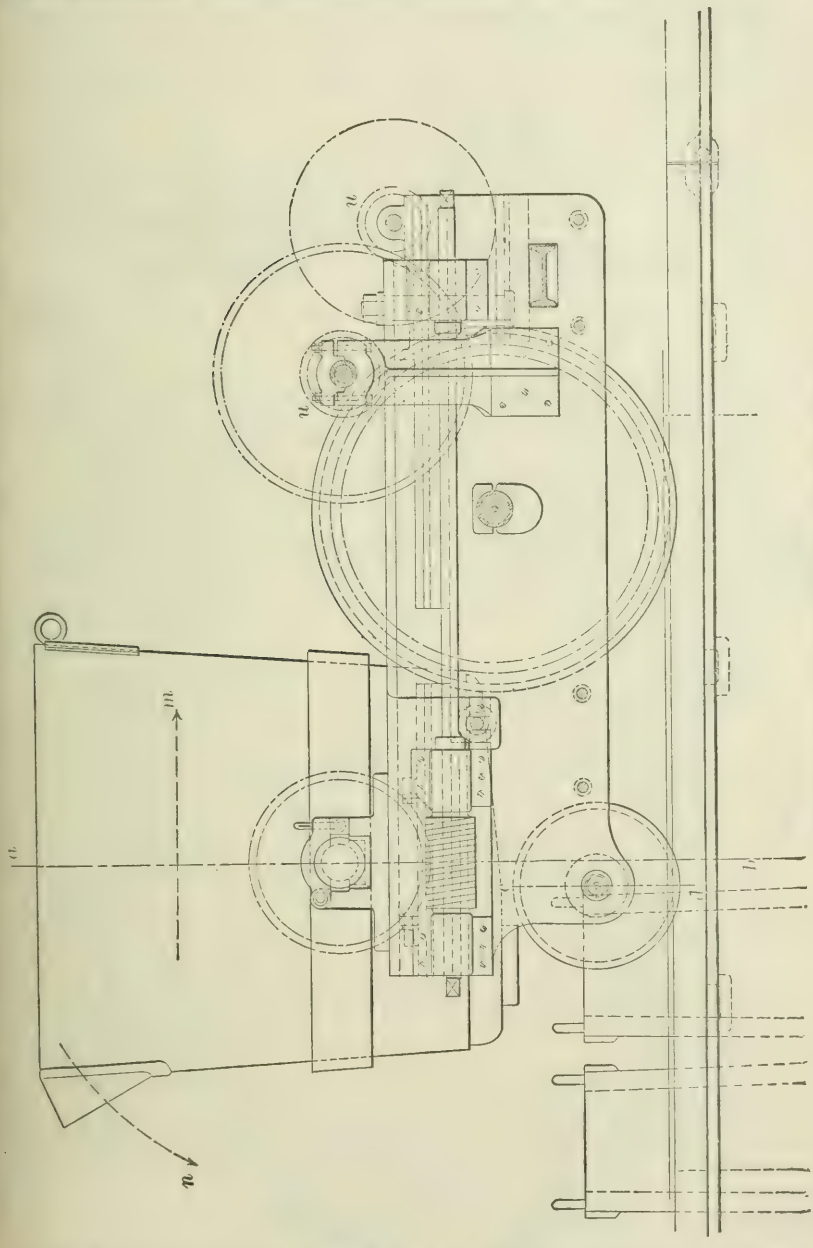


Fig. 7.

Side-Elevation of 12-Ton Casting-Ladle and Car.

Tar-dolomite bricks, moulded in a hydraulic press, have proved an excellent material for the basic lining. Walling-up with them

occupies much less time than ramming, while their durability is not inferior. Mechanical appliances have also been successfully employed in making the bottoms.

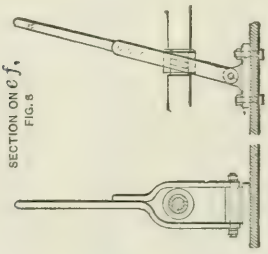


Fig. 10
SECTION ON *e f*, FIG. 8

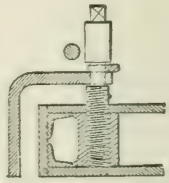


Fig. 9
SECTION ON *g h*, FIG. 8

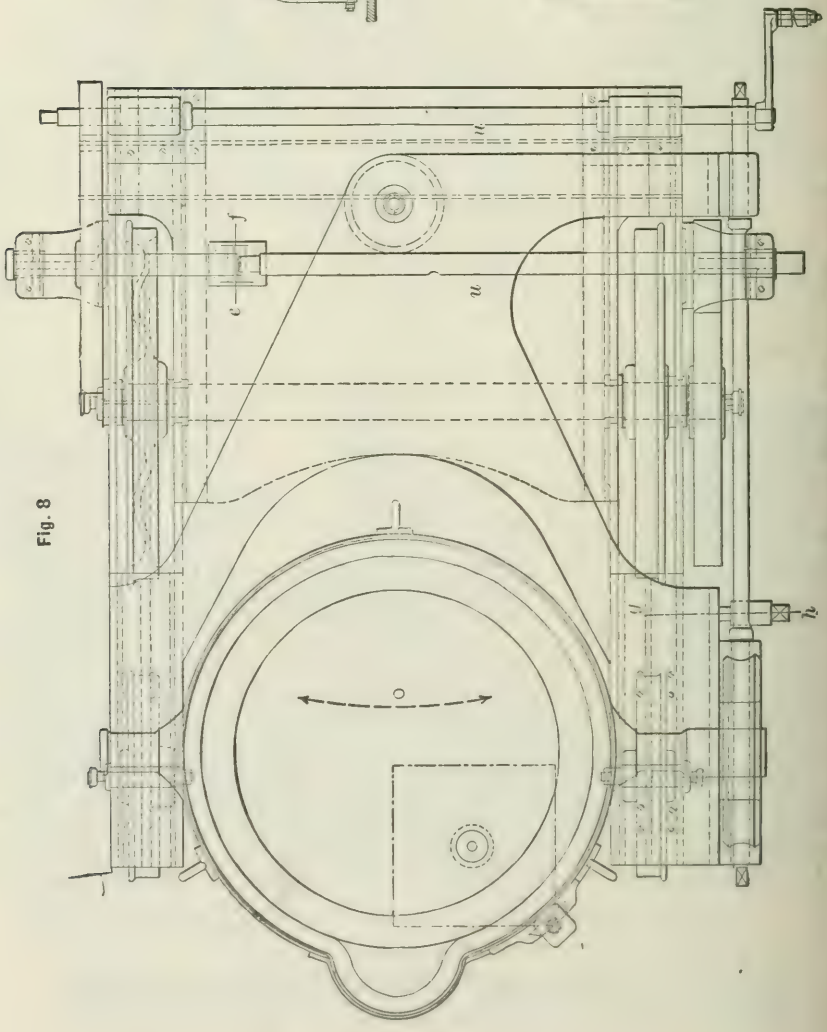


Fig. 8

Plan of 12-Ton Casting-Ladle and Car.

The utilization of converter-slag as a fertilizer is an important business for every basic works. The substitution of a ball-pulverizer for the Chili-mill marks a decided improvement. The new machine costs less and consumes less power, besides easily allowing a far more thorough removal of the dust which works through the casing.

The cause of the great progress which has been made in increasing the output of the basic converter lies first and foremost in the gradually reduced period of the blow. The former duration of twenty minutes for 10-ton converters has been brought down to twelve minutes. The excess of blast over the theoretical requirement, formerly amounting to 150 per cent., is now only about 30 per cent. Great improvement of design is noticeable in the air-compressors, the effect of which has been to reduce the lost space at one end of a single air-cylinder from 5 per cent. to 2 per cent.

Communication between the converter-plant and the open-hearth plant is established by the side-tracks Nos. 2 and 6 (Fig. 1). Over these the ladle of partially-blown metal (*i.e.*, metal taken from the converter before the completion of the after-blow) is brought to the lift, K_2, by means of which it is raised to track No. 7, on the charging-floor, where it is tapped through a short runner into one of the open-hearths.

The requirements of the basic lining have led to material improvements of the Siemens regenerative furnace. Hearth, flues and regenerators are now made independent, in order to secure the greatest possible accessibility for rapid repairs.* These changes have made it practicable to enlarge the regenerators and design them in a more rational form. As a result, higher temperatures have been obtained with exceedingly economical fuel-consumption, and the duration of each heat has been reduced with a corresponding increase in daily output. Six 12-ton charges, each consisting of 8 tons of pig and 4 tons of scrap, are now run in twenty-four hours, with a fuel-consumption of 300 kgs. of coal per metric ton of ingot-iron of the softest quality.

In the instance here illustrated, the casting-house is common to both converter- and open-hearth plant, and forms the connecting link for both with the soaking-pits, *A*, the blooming-mill, *W*, and the finishing-mill, *F*. Soaking is carried on upon a large scale, es-

* See, for example, *Some Recent Improvements in Open-Hearth Steel Practice*, by A. E. Hunt, *Transactions*, xvi., pp. 704 to 707; or, the patents of the Batho Company, of London, England.

pecially in plain, unfired pits. The adoption of this system is continually extending. It is particularly adapted to the working of the blooming-mill, because it permits the use of very heavy ingots of uniform size, and thus simplifies the work of the casting-department. Moreover, since the further reduction of the blooms without wash-heating to $\frac{1}{10}$ or $\frac{1}{15}$ of the original cross-section has proved to be practicable, the soaking-pits effect important savings in that direction also, and it is, therefore, to be expected that they will come into general use in preparing or heating blooms and billets for various finishing-rolls.* The latest improved practice in this direction is reported from the Phœnix Steel Works, near Ruhrort, where ingots, taken from the soaking-pit are now rolled out to rails with a wash-heat of only a few minutes' duration. Three-fourths of the heating-furnaces formerly used are now reported to be idle.

The plain pits require for their successful operation a regular running of the steel-works with a large output. To maintain these conditions, not only a close and careful control of the various processes is necessary—which is in itself an advantage to any establishment—but the construction, arrangement and proportioning of all departments must be exactly adapted to the needs of continuous running. Since this is seldom the case in existing works, which have been more or less remodelled in single departments and retain many antiquated features, the introduction of soaking-pits may be delayed in these until complete reconstruction has become imperative. The following figures show the rapidly extending use of soaking-pits in Europe:

Countries.	1887. Long tons.	1888. Long tons.	1889. Long tons.
Great Britain	237,546	257,335	320,358
Germany	153,454	227,020	397,649
Austria	90,000	95,000	120,000
Belgium	68,162	67,377	71,858
France	8,223	22,558	20,724
Sweden	4,348	4,578	7,624
United States	.....	38,660	.....
Total	561,733	712,528	938,213

According to this table the increase in the treatment of steel by the soaking-pit process in 1888 over 1887 was 26.8 per cent., and

* In rolling rails from 15 by 15-inch ingots, the cross-section is reduced to about $\frac{1}{10}$.

in 1889 over 1888, 31.6 per cent. The comparison of 1889 with 1887 shows a gain of 67 per cent. The indications for the current year are that the tonnage of ingots passed through the Gjers soaking-pits will be considerably in excess of 1,000,000 long tons.

The number of the plain, unfired pits of a Bessemer steel-works is so adjusted that ingots of 1000 to 1500 kg. may remain in them, if necessary, as long as 40 minutes. The loss of metal by oxidation during soaking and rolling, taken together, amounts to 1 per cent. The fired pits are provided with gas-firing and regenerators, or with grates and air-heating. Their consumption of coal is $2\frac{1}{2}$ to $3\frac{1}{2}$ per cent., and the loss by oxidation $1\frac{1}{2}$ to 2 per cent.; while a heating-furnace consumes 5 to 6 per cent. of coal and causes $2\frac{1}{2}$ to 3 per cent. of loss by oxidation. At the same time, the service and the maintenance of the pits are less expensive. They are served by the cranes, *N*, Fig. 1, all the motions of which are controlled by hydraulic pressure. Of these hydraulic cranes, which have many advantages over the simple lever-cranes, there are various designs in use.

The blooming-trains are mostly of the two-high type, driven by reversing-engines. The diameter of the rolls has increased from 900 to 1100 mm.; those of the last-named size being capable of reducing the ingot-diameter $\frac{1}{8}$ at each pass, when the driving-machinery is adequately heavy. For this work the engine, using 75 lbs. steam-pressure, has a cylinder-diameter of 1300 mm.; stroke, 1500 mm.; and a reduction by gearing of $1 : 2\frac{1}{2}$ or $1 : 3$, so that the number of revolutions per minute is 120 to 130.

The feed-rollers for moving the ingot before and behind the blooming-rolls demand special care in construction, because they are subject to breakage of their motive parts, due chiefly to the frequent and sudden reversals of motion to which they are subjected. It is therefore advantageous to operate these feed-rollers by belt-transmission, as this gives a lower initial velocity at every reversal than does a directly-applied reversing-engine.

Of the reversing-engines, *R*, Fig. 1, which drive the rolls, various designs are in use. That which combines a vertical and horizontal cylinder has the advantage of conveying all the power through a single straight shaft;* but engines with three horizontal cylinders have lately been adopted in many quarters, because calculations show that they can be run with a higher degree of steam-expansion in the cylinders.

* For an illustrated description of this type see *Stahl und Eisen*, vol. viii., March, 1888, p. 180.

Blooming-mills are employed chiefly in the production of blooms for rails, shapes and billets. For the plate-mills it is more customary to use bottom-cast ingots from narrow plate-moulds.

The bloom-shears, *S*, Fig. 1, are represented by many different designs, and are provided with horizontally- or vertically-acting blades, according to the needs of their special work. Most of them are operated with hydraulic pressure (up to 500 atmospheres), which is furnished, for the sake of simplicity, by a steam-intensifier close by the shears. This consists of a large vertical steam-cylinder with single-acting piston, on which the steam presses from below, the upward prolongation of the piston-rod constituting a water-plunger in a much smaller hydraulic cylinder, where the pressure is, of course, in the inverse ratio of the two piston-areas. The high water-pressure works upon the pressure-plunger of the shears, and, after the cut is ended, the water returns into the smaller cylinder. Each stroke of the intensifier is produced by a motion of the operator, as would be that of a steam-hammer. After many experiments, this construction was at last made practicable by regulating the speed of the steam-piston by means of the out-flow of water to the hydraulic cylinder. A variety of valves has been designed to carry out this principle. A considerably higher number of strokes, ranging up to twenty-five per minute, has been attained than is practicable with shears driven by rotary motion. As the beginning of the cut is determined by the operator, the danger of waste by cutting too short is avoided.

It may be remarked, in passing, that this system is much used for hydraulic presses and lifts in metallurgical works, because it dispenses with the accumulators, long conducting-pipes and valves under heavy hydraulic pressure, which are so difficult and costly to maintain when hydraulic power is derived from steam-pumps in the old-fashioned way.

For the further reduction of the blooms to rails, ties and shapes, three-high rolls are very generally used. The diameter of the rolls ranges up to 800 mm., which is sufficient for the rolling of ingot-iron I-beams 400 mm. high and 16 meters long.

In these operations the most approved form is the three-high mill with fixed middle roll, and upper and lower rolls adjusted both horizontally and vertically. The upper and lower pressure-screws receive each a cogged-gearing, by means of which they can be operated by a workman standing on the floor at the side of the housings. Such a three-high train requires but three sets of rolls, while a reversing

two-high train must have five sets in order to roll, from blooms, I-beams up to 400 mm. in height. In view of the large product of 300 to 400 metric tons per day, the larger number of workmen required for the three-high mill is less important than the greater consumption of steam for running the two-high. The movement of the pieces to be rolled takes place wholly upon hand-levers, which are connected with a steam-lift, so that the trains of feed-rollers before and behind the mill, which are so expensive to install and to maintain, are dispensed with. It is, therefore, easy to understand that the three-high form is decidedly preferred for rolling-mills, and that the two-high is resorted to only when the dimensions of the product require it.

In the manufacture of plates, the three-high mill, with loose middle-roll (Lauth's system), is largely used, especially in preliminary rolling for the sheet-mills. For the production of heavy plate (boiler-plate, etc.), the following dimensions are employed in one establishment: diameter of rolls, 900 mm.; length, 3400 mm. The reversing-rolls for armor-plate, however, run as high as 1200 mm. in diameter and 4000 mm. in length. In one establishment, such rolls are made of forged cast-steel. The three-high train with loose middle-roll has also been successfully used for universal mills; in which case, however, the vertical rolls come into play during the pass in one direction only.

I would add, in conclusion, that several German manufacturers of machinery have furnished, at my request, drawings and photographs of their improved designs, and that I shall be happy to give to those who may feel an interest in the matter more detailed explanations than have been possible in this brief review.

Discussion.

H. M. Howe, Boston, Mass.: I would ask Mr. Daelen to inform us of the loss by oxidation in the open-hearth furnace in making basic charges of equal parts of scrap and pig. I would also ask, what is the real advantage of the plain soaking-pit? I think that most Americans have been puzzled to discover the advantage of the plain pit over the fired pit. We generally use fired pits in this country. Of course, when the ingots are coming so hot, and the pits themselves are so hot that firing is unnecessary, no gas need be burned, and almost no gas is burned, for the air is shut off and the condition of the plain Gjers pit is practically realized. In fact, I

think our loss by oxidation in the American gas-fired pit is no higher than in the Gjers pit.

What, then, is the advantage of the plain pit over the fired pit other than a slight saving in the first cost of construction? If that is the only advantage, many of us would think it an insufficient one. The gas-fired pit need under no circumstances be too cold, whereas, it will often happen that the unfired pit is too cold on a Monday morning, especially in our extremely severe winter weather. With a fired pit, gas is turned on only at such times and in such quantity as it may be required; when the pit has come up to a proper heat the gas can be shut off, and then the exact condition of the unfired Gjers pit is obtained. That being the case, I think that the reasons for advocating the use of the Gjers pit have been rather obscure to most of us, and I would be glad to hear the explanation.

R. M. Daelen, Dusseldorf, Germany: There is no question in my mind but that the advantages are on the side of the unfired pits. They effect a saving in coal, of which the fired pits consume up to three per cent. in weight of the ingots. Moreover, they diminish the waste by oxidation; in the fired pits there is always some excess of air, because the chimney-draft cannot be so accurately controlled as to avoid it. We invariably find the waste greater in the fired pits. As for the trouble which Mr. Howe fears from cold pits, we experience none of it where the tonnage-output is sufficiently large to keep the unfired pits warm. If, as occasionally happens, the pits are idle for two or three days, then the first new charge of ingots cannot, perhaps, be rolled directly from the soaking-heat, but the second charge causes no trouble. In no case should we employ fired pits where the output of a steel-works is sufficiently large to ensure warm brickwork in the plain pits.

W. F. Durfee, Birdsboro, Pa.: The portion of this very able paper that is particularly interesting to me, relates to this very matter of soaking-pits. It is well known to the profession that for several years I have been endeavoring to introduce the Gjers pit into American works, and that almost wherever it has been tried it has been pronounced a failure. One conspicuous cause to which in part I attribute this result, lies in the attempt made at two or three of the works to run soaking-pits and heating-furnaces at the same time; it has frequently happened that cold ingots, picked up in the yard or from stock, have been put into the furnaces, and whenever these have been ready to "come on," the furnace-ingots have had the preference over those in the pits; thus, the latter have been

allowed to get too cold, and, as a consequence, there has been a slow service of the pits, with a corresponding falling off in tonnage, and, incidentally, some breakages of rolls.

Strange figures have been shown me by parties who have tried to use the soaking-pits. In one case all of the expenses of the two systems of heating were pooled, and the sum was pro-rated between the pits and furnaces on the basis of the actual tonnage passing through each. The people who figured in that way very naturally failed to see any pecuniary advantage in one system over the other. Up to the present time there has been a lamentable want of persistent effort in the United States to make the soaking-pits a success. Nevertheless, I adhere as firmly as ever to the belief that they are destined to be an important adjunct of all steel-works, and that American engineers will in time admit their obligation to Mr. Gjers.

One of the great advantages of the pits is the absence of oxidation, whereas, when an ingot is heated in the ordinary furnace, oxidation commences almost immediately, and in the majority of cases develops to a very wasteful degree. Another important consideration is that the ingot, as taken from the Bessemer casting-pit, is kept vertical all the time, and any bubbles of gas in it naturally rise to the top; whereas, if the ingot, as taken from the Bessemer casting-pit, is put horizontally into an ordinary furnace, those bubbles diffuse themselves along its upper side, and the rail or bloom rolled from it is more or less imperfect throughout its length.

S. T. Wellman, Thurlow, Pa.: I would inquire of Mr. Daelen whether trouble was experienced in Germany from cold tops, and if so, how it was overcome? As I understand it, that has been the trouble in this country from the use of soaking-pits. It has been difficult to get a uniform heat on the whole ingot—the lower end would be too hot, or perhaps of the right temperature, while the top would be too cold.

R. M. Daelen, Dusseldorf, Germany: We met with trouble of this kind when soaking-pits were first tried experimentally. Only four pits used to be put down in a group then, and the troubles to which they always gave rise retarded the adoption of the soaking-process. But, as I have already said, when the number of pits is large, and the output sufficient to keep them very hot, all difficulties vanish. The early trouble with cold tops was due to shallow pits; now that the pits are built very deep, the heat of the ingot is quite uniform, on top as well as all through. The only case in which we should build fired pits to-day is where the output is not large enough

to keep the plain pits hot. In every other case the advantage lies with the plain pits; they effect a saving of fuel and labor, decrease the loss by oxidation, and reduce the cost of maintenance.

H. M. Howe, Boston, Mass.: I do not understand Mr. Daelen's statement that a gas-fired pit cannot be closed practically tight. I fail to see why the covers cannot be as tight on the fired pit as on the unfired pit. It may not be possible to shut the air admission-valves perfectly tight, but they should certainly close so tight that the amount of air which leaks through them should be trifling, and that the quantity of gas needed to take up that air and maintain a non-oxidizing atmosphere should be insignificant in point of cost. Then you have at your command in a fired pit an extra source of heat when you want heat, and when you do not want it, you do not use it. I think that in our country the loss by oxidation does not exceed 1 per cent., with the vertical gas-fired pits of our later mills.

R. M. Daelen, Dusseldorf, Germany: If the loss by oxidation is only 1 per cent., it is no greater than our loss in plain pits in Germany. But our experience in working the fired pits has always given more loss than that, as I have said in my paper. I believe that it is rather difficult to shut off the valves so tight that there is not at least a slight draft through the chimney. Mr. Howe has also told us elsewhere* that records taken directly by him show a loss of 1.5 per cent. In this case, as well as in those on which I have based my statement that the loss in the unfired pit did not exceed 1 per cent., the loss by oxidation during the rolling of the ingot is included. The latter amounts to about 0.5 per cent., leaving a net loss of 0.5 per cent. for the unfired pit, and of 1 per cent. for the fired pit, or just twice as much.

R. W. Hunt, Chicago, Ill.: I am sure that the Institute and the metallurgical world are greatly indebted to Mr. Daelen for the thorough and concise presentation of German practice in his able paper. In this country I would state that there have been several, I may say persistent, certainly expensive, efforts to introduce the non-fired soaking-pits. The Edgar Thomson works, under the administration of the late Captain Jones, spent $35,000 or $40,000 in plant alone, to say nothing of what the experiment may have cost in practice; but their troubles from cold tops and unequal working were so great that the plant was dismantled. Again, the Joliet Steel Company before becoming a part of, or perhaps since their consolidation

* *Notes on the Bessemer Process* in the Proceedings of the Iron and Steel Institute reprinted in this volume of the *Transactions*.

as a part of, the Illinois Steel Company, made a very extensive and persistent effort in the same direction, which was also a failure, or at least it was abandoned. The only case where the pits seemed to work successfully, as far as I know, was at their first introduction in this country at the works of the Scranton Steel Company under the management of President W. W. Scranton. He at first pronounced them a great success, and yet afterwards for reasons of his own, abandoned them. I think that I can explain why the Edgar Thomson and the Joliet attempts were failures, and the failing casts no stigma upon Mr. Gjers's plan. German practice as described and illustrated in Mr. Daelen's paper, requires that the soaking-pit should be close to the ingot-pit. In both the cases to which I refer, the ingots had to be transported several hundred yards. As a result they were naturally and by force, chilled before entering the pits. In the Scranton works the soaking-pits were placed within a few feet of the casting-pit; and they came nearer to successful working, as I have stated. I believe Mr. Scranton's reason for giving them up was his inability to control the working of his pits. In other words, he said that he had demonstrated the possibility of making a piece of steel burn itself. That by too fast working he would get the ingots from the pits in such a condition that they would go to pieces on the blooming-table; just as would have been the case had the steel been overheated in a fired furnace.

Behind every American Bessemer manager there is an urging, goading something which a horseman would picture by a pair of great big spurs; he must get out so many tons of steel, and when he accomplishes that, the puissant "powers that be" still ask for more. A proposition to introduce a machine of any kind that threatens to control the product will not be entertained for a moment by the higher authorities. Therefore the manager must have a furnace which he can absolutely control. He cannot start up on Monday morning and waste two or three rounds of steel, or precious minutes, in bringing up the heat of his pits. He should have several tons of finished product by that time. I believe this is the reason why the Gjers pit has not taken the position here that it has abroad. At the same time it is claimed by our engineers, as Mr. Howe has stated, that they obtain as good results from our fired-pits of newer design. The latest ones built in this country are at the South Chicago Works of the Illinois Steel Company. Our pits were extravagant with fuel in their early history, but as they are now constructed it is stated that the loss is brought down to and

maintained persistently at 1 per cent., hot ingots being used. Of course cold ingots require longer time in the pits and use more gas, and the loss naturally increases.

Another subject I would refer to is Mr. Daelen's statement that in Germany the two-high blooming-mills require less labor than the three-high. It is quite the reverse in the United States—our best three-high mills require practically no labor beyond that of the lever-man, who controls the table-rollers. The ingots are not turned over by hand; we depend upon the Fritz device, which our young men about the mills call the "go-devil," to turn the ingot and move it back and forth. Labor and economy, in our experience, are decidedly in favor of the three-high mill; production, also, is very much in its favor. I am referring now to the rolling of rail-blooms; and as for the rolling of billets in a three-high mill, I may cite the practice at Joliet, Ill., where four-inch blooms are rolled in a three-high combined blooming- and rail-mill—billets, in other words, taking the place of rails as a product—and their output of four-inch billets is almost equal to that of rails; that certainly is beyond precedent, at least so far as I am informed.

J. F. Holloway, New York City: I would like to ask Mr. Hunt if Captain Jones, at the Edgar Thomson Works, did not undertake to supplement the lack of heat by the use of natural gas in his soaking-pits?

R. W. Hunt, Chicago, Ill.: Yes; he did; but we must remember that his pits were not made for gas-burning. He introduced natural gas for the purpose of bringing up the heat of his cold-topped ingots and of those which were too cold all through. He never succeeded in making the pits work. I do not know exactly why.

W. F. Durfee, Birdsboro, Pa.: The practice of the Scranton Steel Works has been mentioned in this discussion. I had several conversations with Col. Scranton and with his manager regarding the soaking-pits there. Col. Scranton assured me of his utmost confidence in the pits, and said that the only reason why he abandoned them was that for certain reasons he increased the size of his ingots to such an extent that the pits as originally constructed would not contain them, and that in order to carry on the soaking-practice he would have been obliged to reconstruct his whole pit-plant, which it was not convenient to do at that time, nor has it been since. He assured me further, that on one or two occasions he had revised descriptions of his works written for technical papers, and had

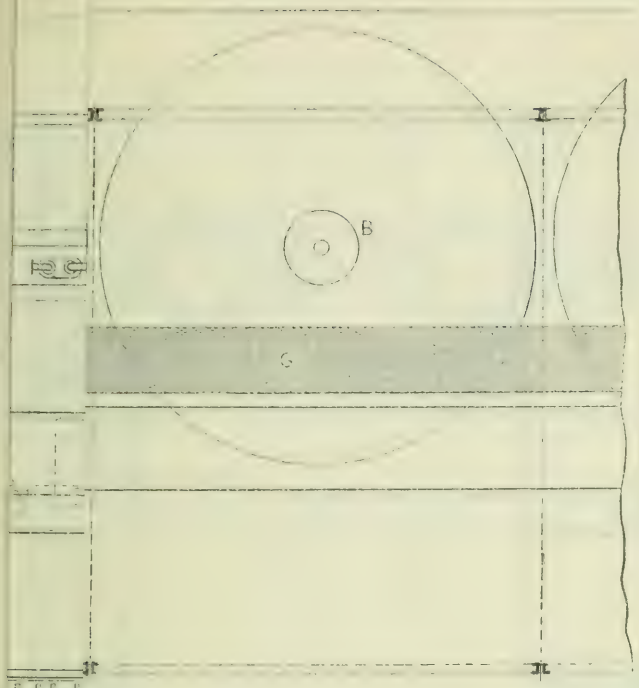


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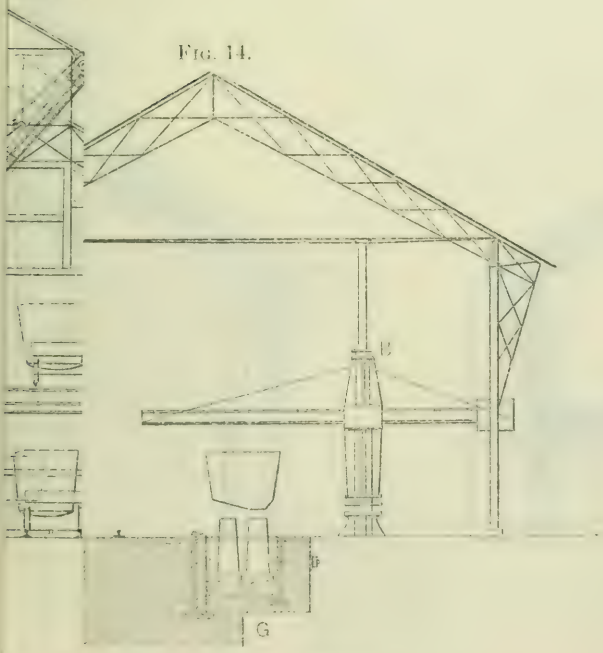
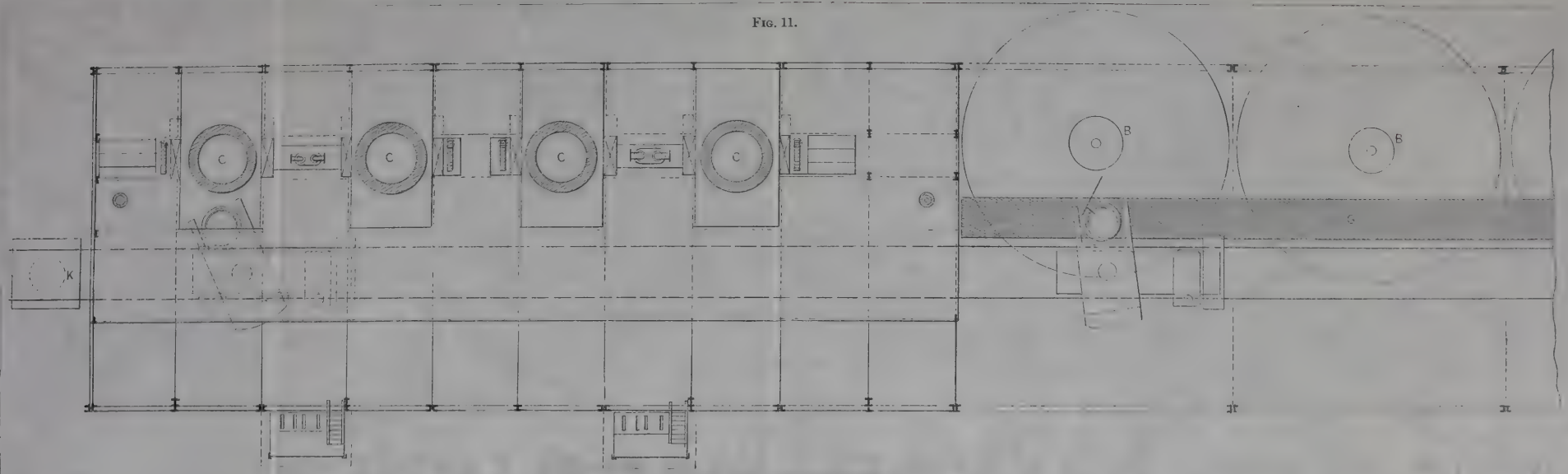
Fig. 14.



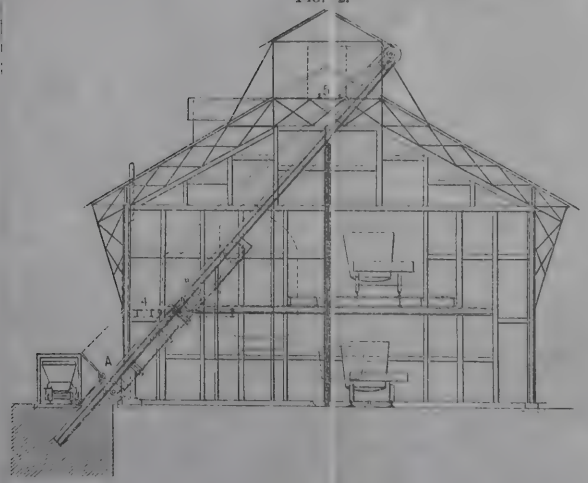
er-House. Section through Casting-House.

Fig. 11.



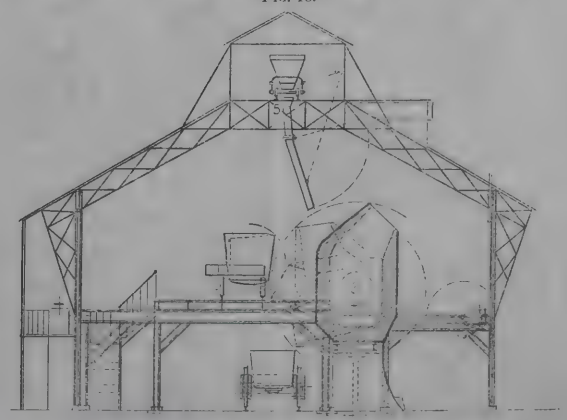
Plan of Converter- and Casting-Houses.

Fig. 12.



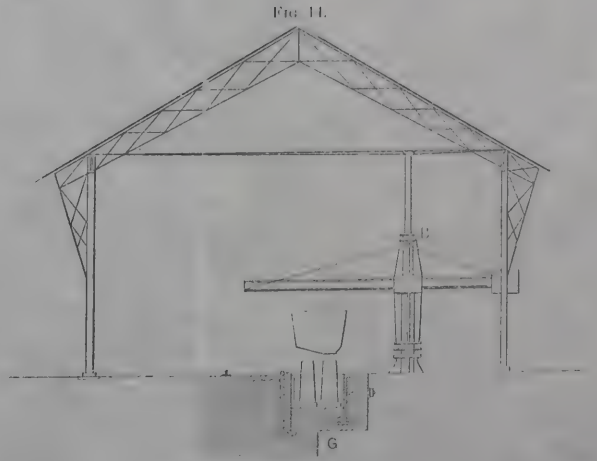
End-Section of Converter-House.

Fig. 13.



Cross-Section through Converter-House.

Fig. 14.



Cross-Section through Casting-House.

stricken out adverse strictures on the pit-practice. In fact, I think that he stated, in a letter published in the *Engineering and Mining Journal* at the time, that he was regularly running more than 80 per cent. of the product of his works through the pits.

The question of the proximity of the soaking-pits to the Bessemer-pit is, I admit, a very important one. The shorter that distance, the hotter will be the ingots when they reach the pits. Yet I think it is possible to carry on the pit-practice successfully even in works where the pits are necessarily some distance away; it has been accomplished abroad and can certainly be accomplished here.

During last year over 900,000 tons of ingots passed through plain soaking-pits in England and on the continent of Europe; this output exceeds the tonnage of the preceding year by 30 per cent., and the tonnage of 1887 by more than 60 per cent. Such figures ought to encourage American metallurgists to introduce pit-practice on the Gjers system in American works. An attempt is about to be made by the Pennsylvania Steel Company at their Sparrows Point works. They are laying down their new plant with the express intention of using the non-fired Gjers pit, and they are devising a very complete apparatus for stripping the ingot-moulds and for conveying the ingots rapidly to the pits. I feel confident that their efforts will meet with entire success.

H. M. Howe, Boston, Mass.: A few months ago I visited the new mill at Dowlais, South Wales, where the Gjers pit is placed immediately around the circumference of the casting-pit, in close proximity to it. So far as I was able to learn from representatives of the Company, the Gjers pit had never been used, and the ingots were heated in furnaces only. Perhaps Mr. Durfee's special facilities for obtaining information will enable him to explain the reason for this.

W. F. Durfee, Birdsboro, Pa.: I learn now for the first time the alleged disuse of the soaking-pits at Dowlais, and therefore I am unable to account for it.

In discussing the introduction of Gjers pits in this country, I omitted to mention labor as one of the factors to be contended with. Men who are accustomed to work gas-heating furnaces will naturally antagonize a system which threatens the existence of their craft. I feel confident that this unfriendliness has had an important bearing upon the introduction of pits in some of the places where they have been tried.

R. M. Daelen, Dusseldorf, Germany: I shall look forward with

great interest to seeing the fired pits in America, and hope to return to the subject-matter of this discussion after having profited by the opportunities for further observation.

With regard to the blooming-mill I am quite of Mr. Hunt's opinion; my personal preference is for the three-high blooming-mill. I have seen one at Darlington which worked well, and I shall hope to see others in the United States. It is my belief that in Germany, up to the present time, the mechanical facilities for quick and convenient handling and lifting of the ingot in front of the three-high mill are inadequate. It is certainly a fact that the preference throughout Germany for the two-high blooming-mill lies in the simpler contrivances that suffice to handle the ingots before a mill of that type.

At the conclusion of the discussion, Mr. Daelen exhibited a collection of photographs and drawings in further illustration of his paper. Of special interest were designs of hydraulic bloom-shears and presses for steel railroad-ties, all shown to be actuated by steam-intensifiers, as described in the author's paper.

ON SULPHUR IN BESSEMER STEEL.

BY JOHN W. CABOT, BELLAIRE, O.

(New York Meeting, September, 1890.)

In the manufacture by the Bessemer process of soft steel suitable for rolling into fine sheets, tubes and so forth, a difficulty is sometimes met with, in the tendency of this kind of metal to rise violently and boil in the moulds while being cast; this results in hollow-topped ingots, and consequently in an undue amount of scrap in the subsequent rolling. It has been noticed that this peculiarity does not pertain to all pig-irons, some grades producing a blown metal which pours quietly, while others, in whatever way the blow may have been conducted, give a metal which rises in the moulds, accompanied by the evolution of large quantities of hydrogen and carbonic oxide gases, and requiring a long time to solidify. This difference in behavior has been some-

what difficult to understand. We were at first inclined to look to the element manganese for the explanation, because of the well-known part which it plays in the reactions between the oxygen of the blast and the other elements, particularly carbon. The higher manganese-irons generally gave a characteristic rising metal, but the manganese theory was abandoned when it was found that pig-irons of lower manganese-content sometimes worked in the same way.

While blowing soft steel at Bellaire from metal taken directly from the blast-furnace, we made some observations which seemed to offer an explanation of this difference in the behavior of different grades of iron.

As the direct-iron has less sulphur than the same iron when remelted in the cupola, owing to the absorption of that element from the coke in the remelting, we were able to produce from it steel very low in sulphur. Now this unusually low-sulphur metal showed the rising tendency in a most marked degree; when, however, the metal from the furnace contained an increased percentage of sulphur, this excessive evolution of gases did not take place, and the casting of the ingots was accomplished in the normal manner.

Some analyses are here given of a number of blows varying in their carbon- and manganese content, which were of this character of rising steel; and, for comparison, a second series which worked quietly. The rising steel analyzed as follows:

C.	Mn.	S.	Si.	C.	Mn.	S.	Si.
0 08	0.32	0.033	0.009	0.09	0.38	0.032	.....
0.08	0.33	0.034	.....	0.08	0.35	0.035	.....
0 10	0.28	0.033	.....	0.09	0 38	0.032	.....
0.10	0.34	0.035	.....	0.08	0.35	0.035	.....
0.09	0.38	0.031	.....	0.10	0.44	0.030	.....
0.09	0.30	0.035	.....	0.10	0.42	0.030	.....
0.09	0.35	0.028	0.009	0.08	0.20	0.030	.....
0.09	0.40	0.036	.....	0.08	0.34	0.036	.....
0.09	0.40	0.036	.....	0.09	0.37	0.036	.....
0.08	0.30	0.035	.....	0.08	0.33	0.035	.....
0.08	0.40	0.038	.....	0.10	0.42	0.035	.....
0.08	0.25	0.035	.....	0.10	0.44	0.032	.....
0.10	0.44	0.039	.....	0 08	0.22	0.035	.....
0.10	0.40	0.025	.....	0.08	0.40	0.030	.....
0.09	0.38	0.025	.....	0.08	0.33	0·035	.....
0.10	0.41	0.020	.....	0.10	0.41	0.020	.....
0.08	0.35	0.035	.....	0 08	0.35	0.030	.....
0.11	0.48	0.028	.....	0.08	0.33	0.028	.....
0.10	0.45	0.025	.....	0.08	0.33	0.028	.....
0.08	0.35	0.035	.....	0.09	0.28	0.032	.....

The normal-working steel is represented by the following:

C.	Mn.	S.	Si.	C.	Mn.	S.	Si.
0.08	0.45	0.069	.....	0.08	0.44	0.050	.....
0.08	0.32	0.057	.....	0.10	0 44	0.050	.....
0.08	0.30	0.049	.....	0.09	0.42	0.048	.....
0.08	0.28	0.048	.....	0.11	0.46	0.054	.....
0.08	0.28	0.084	0.009	0.09	0.38	0.060	.....
0.08	0.33	0.053	.....	0.08	0.38	0.075	.....
0.08	0.25	0.065	.....	0.09	0.42	0.072	.....
0.08	0.38	0.050	.....	0.08	0.28	0.057	.....
0.08	0.35	0.068	.....	0.08	0.30	0.058	.....
0.08	0.27	0.064	.....	0.09	0.40	0.078	.....
0.08	0.35	0.064	0.009	0.08	0.40	0.057	.....
0.08	0.32	0.046	.....	0.08	0.35	0.055	.....
0.08	0.32	0.068	.....	0.08	0.35	0.055	.....
0.10	0.44	0.050	.....	0.08	0.33	0.060	.....
0.09	0 36	0.048	.....	0.08	0.35	0.055	.....
0.08	0.38	0.050	.....	0.09	0.42	0.045	.....
0.08	0.35	0.072	.....	0.08	0.33	0.046	.....
0.08	0.42	0.068	.....	0.09	0.45	0.065	.....
0.09	0.52	0·070	.....	0.09	0.43	0.070	.....
0.08	0.42	0.080	.....	0.09	0.42	0.045	.....

It will be noticed that the only essential difference in the two series is in the amounts of sulphur present. It is unusually low in the rising steel, and of about the amount common in cupola-metal in the case of the tests which worked normally, the average analysis being for the rising steel,

$$C = 0.09. \qquad Mn = 0.36. \qquad Si = 0.009. \qquad S = 0.03.$$

and for the quiet metal,

$$C = 0.084. \qquad Mn = 0.39. \qquad Si = 0.009. \qquad S = 0.058.$$

It is well known that sulphur exerts a potent influence upon the condition in which carbon exists in combination with iron in pig-metal. A few hundredths of one per cent. are sufficient to throw a large portion of it into the combined form in an iron which would otherwise be graphitic. Now, in the case of the heats blown from the high-sulphur metal, the sulphur probably acted in the same way in forcing the carbon of the metallic bath into combination (in which condition only is it readily attacked by the oxygen of the blast), the line of separation between complete carbon-oxidation and the beginning of iron-oxidation being thus more sharply defined, and the carbon thoroughly washed out before the iron was attacked.

This supposition was borne out by the character of the change in the flame, which was quick and decided.

In order to determine the correctness of this view, we added to the blows of the low-sulphur iron enough more sulphur to increase the .01 and .02 per cent. originally present to .05 and .06, and in every case the troublesome evolution of gas from the metal was prevented and the steel poured quietly. In whatever way the extra hundredths were added, whether by adding coal, or shale, to the heats, or by mixing with them a high-sulphur pig-iron, the effect was the same.

No doubt this peculiarity of low-sulphur steel can be overcome by mechanical means, such as stirring or by the method of repouring. The presence, however, of .04 or .05 per cent. of sulphur seems to have a favorable influence on the reactions taking place between the oxygen of the blast and the elements in the pig-metal, in producing this description of material.

INTERESTING VEIN-PHENOMENA IN BOULDER COUNTY, COLORADO.

BY JOHN B. FARISH, DENVER, COL.

(New York Meeting, September, 1890.)

THIS brief paper is designed to place on record some interesting occurrences, recently observed during an examination of the Golden Age Mine, in Boulder county, Colorado.

Leaving the little village of Jamestown—or Jimtown, as it is more generally called—located in the cañon of James creek, the road winds up a steep mountain composed of the coarse gray granite, with occasional belts of gneiss, common to all the mineral-bearing sections of Boulder county, to the summit of the front range of the Continental Divide. Here are located the mines known as the Golden Age and the Sentinel, the relative locations of which are shown in Fig. 1.

The Golden Age location covers the outcrop of a quartz-porphyry dike, which cuts through the granite country-rock with a strike of about N. 70° E. This dike varies in width from a few feet, as at a point between the main shaft and the Hill shaft, to about 50 feet,

and dips about 45°, as shown in Fig. 2. The outcrop of the main

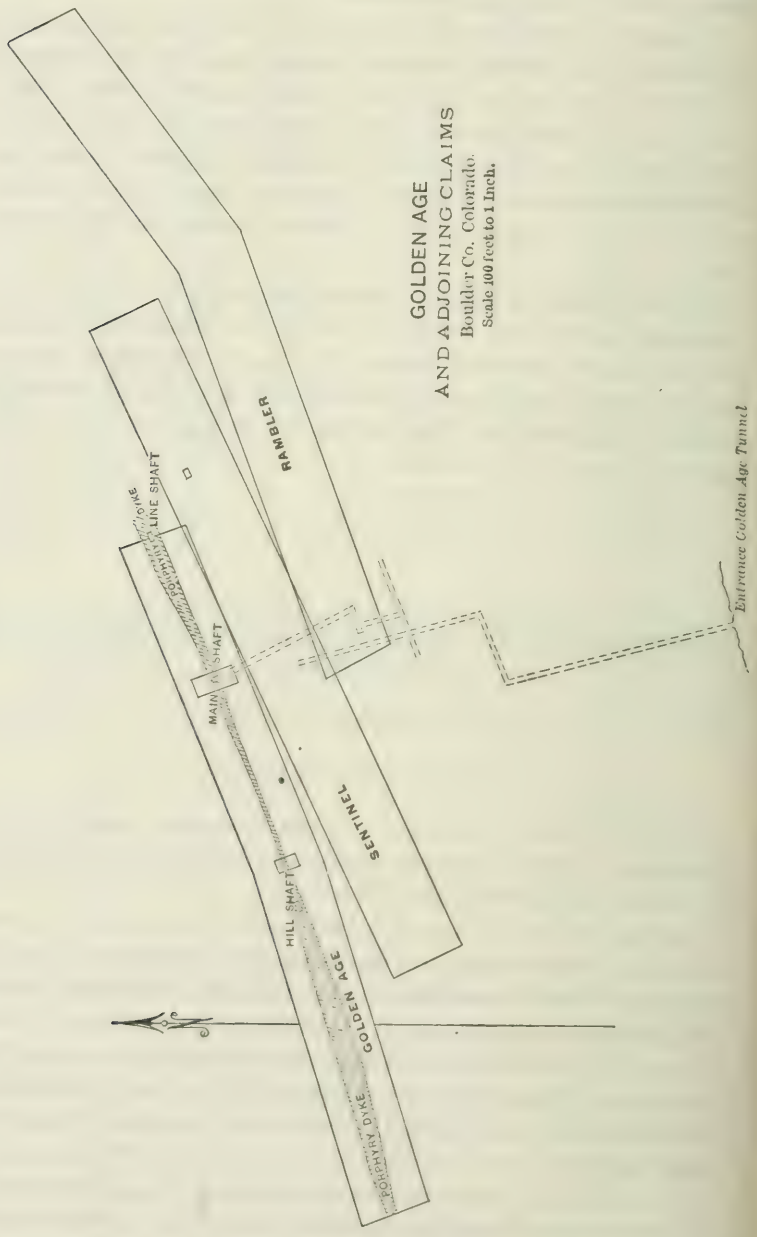


FIG. 1.

ore-chute thus far explored on the Golden Age vein is marked by a line of surface-works, extending along the contact on the lower side

of the porphyry dike, from the Line shaft on the east to the main shaft on the west. At the depth of about 100 feet the main shaft

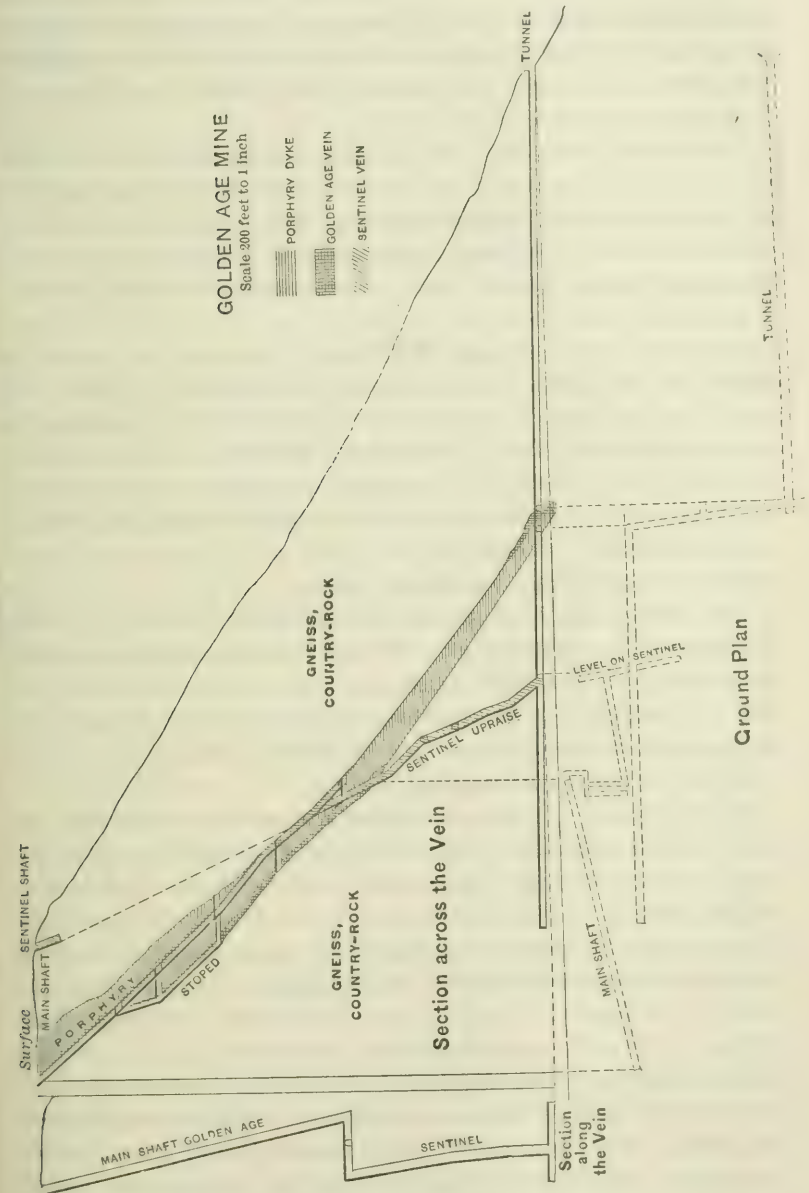


FIG. 2.

discloses a split in the vein. The upper or hanging-wall streak

continues into the dike on approximately the same dip, but with porphyry hanging- and foot-walls, until a depth of 370 feet is reached, where it enters the upper contact between the porphyry and granite, and remains in it to the bottom of the main shaft, 470 feet on the incline below the apex. The adit which cuts the porphyry dike about 250 feet below the bottom of the shaft, shows this streak in the same contact.

Beyond the split, the lower or foot-wall streak, stands a little steeper; and at the second level, 180 feet below the surface, it is found in the contact between the under-side of the porphyry dike and the granite foot-wall. It remains approximately in this contact, though occasionally found wholly in the granite, to the lowest explorations in the shaft.

The porphyry dike varies in width on its dip as well as on its strike. On the third level a cross-cut, 47 feet long, reaches the granite on both sides of the dike, while in the adit the porphyry is passed through in 8 feet. It has been considerably acted upon by the vein-forming agencies in the upper workings; and in none of these openings was I able to secure a specimen that was not more or less decomposed. Where it is cut by the adit it is less acted upon, and shows considerable amounts of pyrites.

The Golden Age veins are well-defined, often presenting a banded structure. They are inclosed in distinct walls, with selvages, which at times exhibit slickensides. So far as my observation went—and this was confirmed by Mr. Amsden, who has been foreman on the mine for ten years—the seams and feeders that have enriched both veins come in from the porphyry dike.

The ore from the Golden Age veins is somewhat remarkable, and the rich and magnificent gold specimens obtained from them are familiar to all the miners of Boulder county. It is a typical free-milling gold-ore. A good percentage is saved by simple amalgamation on copper-plates, the resulting bullion running over 900 fine; the tailings yield pyritic concentrates of fair grade. As a rule, the vein-rock, especially when rich, is a hard, often flinty or vitreous, white quartz. The gold, especially in the hanging-wall vein, is seldom accompanied by pyrite, chalcopyrite, or any of the baser minerals. It is generally imbedded in the white quartz as bright yellow gold, ranging in size from coarse grains to nuggets, occasionally several ounces in weight. One specimen, found by a former owner, contained 70 ounces of gold, nearly all in one piece. The foot-wall vein contains more of the base minerals than that of the hanging-wall. After it

reaches the lower contact between the porphyry and granite, there is a marked increase in the quantity of these associated minerals, and this is still further increased when the vein leaves the contact and enters the granite. In such places, blende and galena appear in small quantities, with pyrite and considerable chalcopyrite; but the gold in the ore remains as free as elsewhere. In none of the openings were any tellurium-minerals found by me; and Mr. Amsden says they are unknown in the Golden Age.

Returning to the surface, the Sentinel location covers the apex of a vein which appears on the surface enclosed in a belt of schistose or gneissic rock. It lies nearly parallel to, and about 100 feet south of the apex of the Golden Age vein. Only two shallow openings have been made on this vein on the surface; the deeper being a shaft sunk about 30 feet. In driving the adit, the first vein encountered was the Golden Age, on the hanging-wall contact of the dike, and about 175 feet further north the Sentinel vein was reached in granite. An upraise was made on this vein to the lowest (No. 5) level from the main shaft, demonstrating that this vein dips south about 70°, and passes through the Golden Age vein on its course.

The Sentinel produces an ore entirely distinct from that of the Golden Age. It is the familiar bluish quartz of the tellurium veins of Boulder county, with the characteristic chalcedony, quartz crystals and finely disseminated pyrites. The value is in metallic gold, petzite and sylvanite. While most of the metallic gold was deposited as native gold, a certain portion has evidently been rendered free by the partial decomposition of the tellurium-minerals. Some of this ore is very rich. One specimen recently found, weighing a little over two pounds, is valued at $228.00.

The richest ore usually occurs in two narrow seams or streaks, often only a foot, but at times as much as 10 feet apart; the intervening space is more or less mineralized country-rock. The miners working as lessees on this vein, consider it richest in the schistose rock, and poorest when it is in the porphyry, on its course through the dike. Though the openings on the vein, in the dike, are limited, this opinion appears to be correct.

So distinct are the characteristics of these veins, that the crossing of the Sentinel through the Golden Age is plainly marked, being exposed in the main shaft and workings connected with it. The dip of the former, as stated, is about 70° south, and is quite regular as far as it has been explored, though at the points where it comes in contact with the quartz streaks of the Golden Age veins, it often

follows along, without a break in its continuity, either above or below them for short distances, before finally passing through and assuming its regular dip. Nor does the dip of the Golden Age veins appear to be much disturbed, the greatest vertical displacement noticed being only about 30 inches, at a point where the vein is broken by the passage of the Sentinel vein through it.

It is possible that the facts observed here confirm the opinion that the gold-mines of Boulder county belong to at least two distinct periods of vein-formation. To the first and earlier may be assigned the Golden Age, and the mines of Ward and other districts producing similar gold-ores, free from tellurium-minerals; to the second and later, the tellurium gold-veins for which Boulder county is particularly noted.

That the ores from the Sentinel, or tellurium-vein, are of lower grade where the vein passes through the porphyry dike than elsewhere, may be due to the formation of the Golden Age vein first. This vein probably drained the dike of much of its disseminated mineral-values. The Sentinel doubtless received its mineral through the schistose or gneissic rocks, and is consequently richer where enclosed in those rocks than in the dike.

Prospectors, as a rule, look for richer or larger bodies of ore where veins unite or cross each other. In this property we have two interesting occurrences of this kind. The Golden Age veins unite at a point 100 feet below the surface. These are similar veins of the same age. The result was seen in larger and richer ore-bodies mined in the stopes near the main shaft and adjacent to the junction of the veins. In the other case, the crossing of the Sentinel tellurium-vein through the Golden Age veins, which was the passing of a later through earlier veins, produced no local enlargement or enrichment of the ore-bodies. It may be inferred with probability as a general rule, subject of course to local exceptions, that for the production, by the junction of two veins, of ore-bodies larger or richer than are characteristic of either vein separately, the two veins should be of contemporaneous origin.

Discussion.

R. W. Raymond, New York City: Perhaps the concluding proposition of Mr. Farish's paper may be both confirmed and explained by restricting its application to such cases of enrichment as result from the simple enlargement of the space available for deposition, and the consequent diminution of the speed of percolating

currents—both of which conditions are observed to produce similar effects in veins which vary in dip or width but are not intersected by other veins. Practical miners look for the larger ore-bodies in the flat places and the wide places of the vein, and the pinches, though they do now and then compensate by special richness for their contracted size, are usually as "lean" as they are thin.

LATEST DEVELOPMENTS IN COMPRESSED-AIR MOTORS FOR TRAMWAYS.

BY D. S. JACOBUS, HOBOKEN, N. J.

(New York Meeting, September, 1890.)

COMPRESSED-air motors are specially desirable, if economically practicable, for underground haulage, because they require no fuel, involve no danger of fire, and not only avoid heating and fouling the atmosphere with smoke and combustion-gases, but by the emission of expanding fresh air, cool and improve it and aid ventilation.

The highest developments in the use of compressed-air motors for locomotion are to be found, in the opinion of the writer, in the principles of the Mekarski car-motor as applied to street tramways, and it is thought that notes taken regarding this system, during a recent trip to Europe, where two plants are in successful operation, at Nantes and Vincennes respectively, will be interesting to mining engineers.

The plant at Nantes operates a street-line 5.2 miles long, requiring about twenty motors; the one at Vincennes about 7 miles of road. Each car at Nantes carries its own motor. At Vincennes, however, the motor-car is arranged to tow a second car containing no motor.

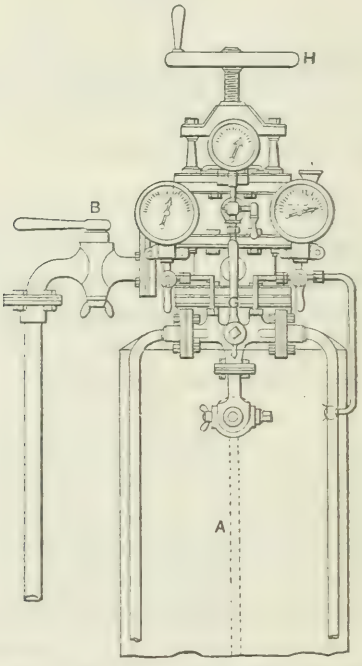
The road at Nantes is the older, having been in successful operation for ten years, during which time neither the American consul, Mr. Shackelford, who has resided at Nantes for about five years, nor the vice-consul, Mr. Bennett, who has been at Nantes ever since the road started, has ever seen anything go wrong with the motors.

DESCRIPTION OF THE RAILROAD AT NANTES.

Motor-Car.—Two small engines are connected so as to rotate the front axle of the car, a reversing lever being used to alter the cut-

off and run the motor in either direction. The compressed air is stored in ten tanks, fastened under the car, eight of these tanks being connected together and constituting what is called the battery, and the other two forming the reserve. Air may be admitted from either the battery or the reserve to the tank, A, Fig. 1, where it

Fig. 1.



Regulating Mechanism of the Mekarski Motor.

passes through a mass of hot water before it reaches the regulating-valve, the construction of which can best be seen in Fig. 2.

The valve is opened and closed by a difference of the air-pressure acting on the two sides of the diaphragm, *a*, *a*. The pressure in the space, E, will therefore be governed by the air-pressure that is brought to bear on top of the diaphragm, which pressure can be varied at will by means of the piston, D. From the regulating-valve the air passes to the three-way cock, B, Fig. 1, which causes it to pass either to the engines or to the air-brakes. The dimensions are as follows:

Diameter of cylinders of engines, $13\frac{1}{2}$ centimeters or $5\frac{5}{16}$ inches.

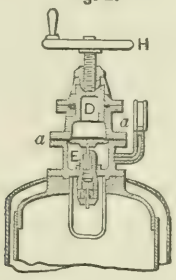
Stroke of engines, 26 centimeters or $10\frac{1}{4}$ inches.

Compressed-air cylinders, about 18 inches diameter and 5 feet long.

The cylinders containing the compressed air are not all of the same size. The total capacity of a set of connected cylinders, as given in Mekarski's circular, is such that each kilogramme of pressure per square centimeter corresponds to 2.5 kilogrammes weight of air in the battery and 0.8 kilogramme of air in the reserve. The pressure of air employed at Nantes when the motors are fully charged is 30 kilogrammes per square centimeter in both the battery and the reserve.*

The motors run very steadily, there being no appreciable teetering or rocking when the car is at full speed. In starting, there is no

Fig. 2.



Regulating-Valve.

jerking motion experienced, and as air-brakes are used for stopping, the quickness with which this is done may be varied at will. The air exhausted from the motors makes a slight noise, but not enough to cause annoyance to persons residing near the car-lines.

Air-Compressors and Storage-Tanks.—The air-compressors are of the compound type, in which air is first compressed into an intermediate receiver, and from this enters the high-pressure cylinder, where it is compressed to the final pressure. The air-cylinder pistons are both fastened to the same rod. The cylinders are single-acting, the pressure in the receiver being made to act constantly on one side of the high-pressure piston. The steam-cylinder is set parallel to the air-cylinders, and the pistons of all the cylinders are connected to the same fly-wheel shaft. The dimensions are about as follows:

* A pressure of 1 kilogramme per square centimeter is equivalent to 14.22 pounds per square inch.

	Four Compressors at Main Station. Inches.	Two Compressors at Supplementary Station. Inches.
Length of stroke of air-cylinders,	23.6	17
Length of stroke of steam-cylinder,	39.4	28
Diameter of high-pressure air-cylinder,	9.4	7
Diameter of low-pressure air-cylinder,	19.7	15
Diameter of steam-cylinder,	19.7	14

The compressors are made to force air into a receiving-tank until the arrival of a motor that is to be charged. The motor is first connected to the receiving-tank so that its cylinders become charged to the pressure of the tank; they are then connected directly to the compressor until the required pressure is reached, at which point an automatic valve acts to prevent any excess if the person charging the motor does his work in a careless manner. The receiver is always at a lower pressure than that required for the cylinders of the car, so that all the work of storing the air is not done against the maximum pressure. At the same time that the air-tanks are being charged, the water in the tank, A, Fig. 1, is heated by passing steam into it. The loss of water by vaporizing during the trip is just about the same as the amount of steam that is required to be condensed in the water in order to heat it, so that there is no escape of steam or hot water at the charging-station. No opportunity was offered to measure the temperature of this water, but, according to Mr. Mekarski, it is 160° Cent. when the motor starts from the station, and about 100° Cent. when it returns.

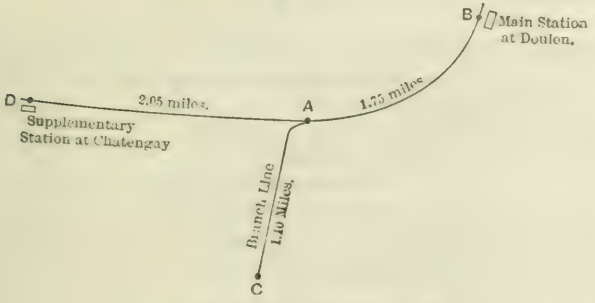
Method of Running the Motors.—The reversing-lever is first thrown forward to the last notch from the center; the hand-wheel, H, is then slowly turned so that it forces the piston, D, Fig. 2, downward, and opens the valve admitting air to the chamber, E, from which it passes through the cock, B, Fig. 1, to the engine. The lever is held in the forward position until the motor begins to start, and then it is brought to the notch nearest to the center. When the lever is forward in the last notch, the cut-off occurs at $\frac{3}{4}$-stroke; in the notch nearest the center the cut-off is $\frac{1}{4}$.

About 6 to 10 kg. per square centimeter are used on the engines at starting, and this pressure is preserved until the motor attains a speed a trifle faster than the average. The pressure is then shut off from the engines, provided the track is quite level, and the reversing lever is thrown forward; the car then runs a short time by its inertia. When the speed diminishes, the lever is brought back to the first notch, and 6 to 10 kg. again brought on the engines. By

working the air in this way, it can be used more efficiently than if a constant low-throttled pressure were used. In going up grades the pressure is constantly on the engines and, if possible, the lever is in the first notch. There was no grade at Nantes so steep that this could not be done. To stop the car, the handle of the three-way cock, B, Fig. 1, is turned, and the required amount of air-pressure brought to bear on the brakes by screwing down the hand-wheel, H. When the pressure in the battery, which is the first set of cylinders drawn from, becomes too low to start quickly, or to drive the motor up a grade, the three-way cock shown at G, Fig. 1, is turned so as to allow the pressure of air from the reserve to be communicated to the interior of the tank, A. The cock is then turned back to its former position, so that there will still be a high pressure of air in the reserve to be used again in case of need.

Stationary Plant.—Fig. 3 will give a general idea of the plan of the tracks at Nantes, although there are many more curves in the lines than are here represented.

Fig. 3.



Plan of Route at Nantes.

The motors start from the main station, B, charged to about 30 kg. per square centimeter in both the battery and reserve, go to A, then to C, and back to A, and, finally, to D. At the supplementary station, D, the tanks are re-charged to about 20 kg. per square centimeter, and the motor returns to A and then to B.

There are several stations along each route at which inspectors enter the cars to see if all the passengers have tickets, these tickets having been purchased from a conductor who rides on the car.

The track is quite level, the heaviest grade being where the road crosses bridges the road-beds of which are higher in the middle than at

the two ends. The maximum grade is 1 to .045, and all the grades are short.

There are four compressors at the main station at Doulon and two at Chatengay, the dimensions of which have been already given. Two of the compressors at Doulon are run on Sundays and *fête* days, at which time there are twenty motors in use, whereas on week-days, one, and only for part of the time two, compressors are run and sixteen motors are in use. At Chatengay both of the compressors run on Sundays and only one during the week. The motor-cars have seats for sixteen persons without crowding, and eighteen with slight crowding. The back platforms are quite large, however, so that sometimes thirty or more passengers will be carried at one time. There are in all twenty-two motors belonging to the company.

RECORD AT NANTES FOR THE YEAR ENDING JULY, 1889, AS GIVEN BY MR. MEKARSKI.

Fuel, 1300 tons at $5 per ton, delivered to station,	$6,500.00
Oil and waste for rolling-stock at $80 per ton,	862.00
Oil and waste for stationary plant at $80 per ton,	300.00
Total number of men employed at Doulon Station,	7
" " " " " Chatengay Station,	5
Number of trips made by each motor per day,	6
Length of each trip (one full charge of air and one two-thirds charge for each trip),	10.4 Miles.
Number of miles made by each motor per day,	62.4
" " motors in use on week days,	16
" " " " " Sundays and *Fête* days,	20
Time of starting the first motor from the station,	7 A.M.
Time of starting the last motor from the station,	7.08 P.M.
Total number of miles run during the year,	346,600

The cost of repairs for this particular year could not be obtained, but in a statement of expenses for the year 1888, Mr. Mekarski places the maintenance of rolling-stock at 26,011 f., or about $5200, and of the stationary plant at 15,803 f., or $3200.

Tests of Speed.—Table I. shows the trip-records taken at Nantes, while Table II. gives the totals and averages of all the results. They clearly exhibit the following points:

1st. The motors started with about the same rapidity as an ordinary street car, *i.e.*, they attained full speed in 12 to 16 seconds, and could be started quicker than this, but at a greater expenditure of compressed air.

2d. The total distance run over in order to measure the speed was 22.55 miles, and the number of stops made was 61. The rate of

speed including stops was 7.58 miles per hour, and excluding stops 8.52 miles per hour.

3d. The greatest distance run with a single charge of air was 6.6 miles; this required 30 kg. per sq. cm. In returning from the supplementary to the main station the motors ran a distance of 3.8 miles, being charged at the start to about 18 kg. per sq. cm.

The recorded number of passengers in the tables gives the number on the car at the time of starting after a stop; this number does not include the driver and conductor. The average number of passengers is obtained by taking the average of the number on the car at each minute of the time required to make the trip.

Horse-Power and Coal Consumption of Compressors.—The mean effective pressure for 60 revolutions per minute, as shown by indicator cards taken by Mr. Mekarski, is 32 lbs. per square inch. We also have:

Diameter of cylinder = .5 m. = 19.7 inches. Stroke = 1 meter = 3.28 ft.

$$\text{Hence H. P.} = \frac{305 \times 32 \times 3.28 \times 60 \times 2}{33000} = \text{about } 115.$$

Sixty revolutions per minute is, according to the judgment of the writer, about the average speed that would be obtained by adding together the revolutions made by the two compressors during the day and dividing the sum by the number of minutes, for the compressors run ordinarily at about 45 turns, and one runs all day, and the other only occasionally.

The compressor in the supplementary station compresses only to about two-thirds of the pressure obtained at the main station, so that to fill a given volume its horse-power would be about two-thirds of 115, or say 75, provided the indicator-cards of the air-cylinder have the same mean effective pressure. As, however, the indicator-cards will be smaller, we may say that about 60 horse-power is required at this station, making the total requisite horse-power at both stations 175.

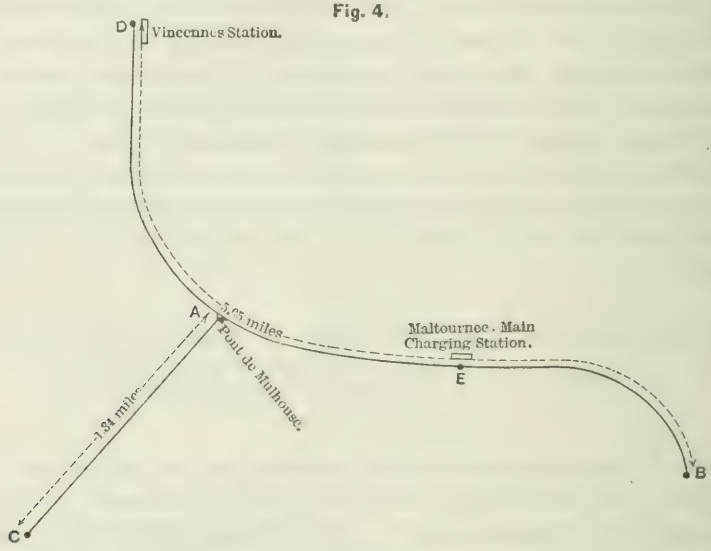
The engines have a condenser; if we therefore assume 3 lbs. of coal per horse-power per hour we obtain:

Coal per hour = $175 \times 3 = 525$ lbs.
" " day of 12 hours = 6300 lbs.
Figure given by Mr. Mekarski = 1300 tons per year.
" " " " = 7100 lbs. per day.

Description of the Railroad from Vincennes to Nogent.

Plant.—Figure 4, from which many curves have been omitted, will give an idea of the route, the distances being taken from Mr. Mekarski's pamphlet describing his system.

Fig. 4.



Plan of the Route at Vincennes.

The motors on this line are larger than at Nantes, and are charged to a higher air-pressure. The motors on the main line D, A, E, B, draw a supplementary car; they are charged at D and run to E, where they are replaced by another motor that has been charged at E; from E the second motor and car either proceed to B, and return to E, without a further charge at B, or they return to D without going to B.

The motors that run on the branch road do not draw a supplementary car; they are charged at E and run to A, and to C, and then return to E, making the round trip with a single charge of compressed air. There are seats on the top of the motors and the cars, as well as inside of both. The number of passengers given in the record is only approximate, as it was not possible for one person to keep an exact tally for both motor and car. At some portions of the trip the motor and car had all the passengers that could be accommodated.

The principal results obtained are:

Tests of Speed.—The details of a number of trip-records, taken on this line, are given in Table III., and the totals and averages are shown in Table IV.

1st. The motor with a car in tow started in from 20 to 40 seconds, which is about twice the time required by an ordinary street-car.

2d. The total distance over which a record was kept, in order to determine the speed of a motor when drawing a car was 11.9 miles. The average speed including stops was 8.58 miles per hour, and excluding stops 9.26 miles per hour.

3d. The motor alone ran on the branch line at 8.81 miles per hour including stops, and at 10.65 miles per hour excluding stops.

4th. The average distance run by a motor with a single charge of air and drawing a supplementary car, was 3.97 miles, and this required the air in the tanks to be compressed to about 40 kg. per square centimeter.

Time Required to Change Motors.—The only case in which this has to be done quickly is at Vincennes Station. The motor is disconnected from the car, run to the charging station, about 75 yards distant, charged there and then run back to be connected to the same car. In timing these operations the motor was disconnected from the car, turned at right angles to the track on a turn-table, and run to the charging station in three minutes. The air and steam were turned on three-quarters of a minute after the motor came to rest on the platform, and exactly three minutes afterward the motor was disconnected and started back toward the turn-table. At Nantes the motors are left on the charging platforms about 15 minutes, during which time the driver oils up the engines.

Estimated Cost of a Road in the United States.

As a basis for estimating the cost of equipping and operating a line in this country, we will take a double-track road $3\frac{1}{2}$ miles long, *i.e.*, 7 miles of track, two terminal charging-stations, and cars running at four-minute intervals for twelve hours a day. One gang of men is to be employed at each station.

Two charging-stations are required, one at each end of the line, unless the track is quite level and is kept *perfectly clean*. In the case of a city where wagons are continually crossing the tracks, thereby causing the motors to be slowed down and started up very often, and where the streets are not kept clean, it would require much more power to operate the motors than at Nantes, where there was little interference with the continuous running of the motors,

except for taking on or letting off passengers, and where the rails were kept as clean as possible by a couple of track-walkers employed for the purpose. Besides this, in many cities snow has to be contended with. There would also be many more stops for passengers in one of our larger cities than at Nantes. For these reasons the length of the assumed line, which has no intermediate charging-stations, has been limited by the practice developed at Vincennes, near Paris, where the motor and tow-car travel with one unrenewed charge of air an average distance of 3.97 miles.

The cost of coal per mile per motor at Nantes, at $5 per ton, is 1.875 cents; or at $3.50 per ton, 1.31 cents. Assuming that motors running in this country would require one and one-half times the power that they do at Nantes, the coal per mile per motor will be 1.96 cents.

The total number of miles made by the motors per day on the proposed plant is $\frac{60}{4} \times 12 \times (3\frac{1}{2} \times 2) = 1260$, and the cost of coal per day 1260×1.96 cents = about $24.50.

The waste and oil for rolling-stock at $80 per ton at Nantes, or about 30 cents per gallon, is 0.25 cent per mile per motor, or $3.15 per day. If oil is 50 cents per gallon, this comes to about $5.25 per day.* The oil for the stationary plant at this same price is about $1.75 per day.

The cost of maintenance of the rolling-stock as given in the printed statement by Mr. Mekarski is .052 f. per kilometer, or 1.67 cents per mile per motor. Assuming that the greater price paid for labor in the United States will add 50 per cent. to this figure, we have for the repairs per mile per motor $1.67 \times \frac{3}{2} = 2.5$ cents or $31.50 for the entire number of motors per day.†

The cost of maintenance of the stationary plant obtained in a similar way is $19.50 per day.

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* This assumes that the price of waste in the United States is sixty-six per cent. higher than at Nantes.

† Although it is fully appreciated that this figure is one of the most important in an estimate of this kind, it has been impossible to check it by directly observing the number of men in the repair-shop, because all repairs are made in Mr. Mekarski's shop at Doulon, and this shop is usually employed on other commercial work. There are 22 motors in all belonging to the company, and it is said that 20 of these have been run every Sunday and *fête* day since the road was started, so that it appears to be exceptional to have more than two motors undergoing repairs at the same time.

The number of men, exclusive of those making repairs, that would be required at each station are: 1 engineer, 1 oiler, 2 firemen, 2 workmen, and 2 charging-men, or 8 men in all, and an extra engineer who would take the place of either of the others in case of sickness. These figures correspond with what Mr. Mekarski gave me for the main station at Nantes.

The items for supplies and repairs for both stations are therefore:

Cost of coal per day at $3.50 per ton,	$24.50
Oil and waste for rolling-stock,	5.25
Oil for stationary plant,	1.75
Maintenance of rolling-stock,	31.50
Maintenance of stationary plant,	19.50
Cost per day—total supplies and maintenance,	$82.50

The labor items are:

3 engineers at $3.00 per day,	$9.00
2 oilers at $2.00 per day,	4.00
4 firemen at $2.50 per day,	10.00
4 workmen at $1.75 per day,	7.00
4 charging-men at $2.00 per day,	8.00
Total expended for labor in two stations per day,	$38.00

If we assume the cost of the two buildings required for this plant to be the same as that of a single larger one which would be required if horses were used, then, so far as plant is concerned, we have only the charge for depreciation of the machinery and motors in Mr. Mekarski's system to balance against the replacement and acclimating of horses in a system in which horses are the motive-power. The approximate cost of the machinery will be:

4 compressors,	$15,000
4 boilers,	5,000
22 motors,	25,000
Total cost of machinery,	$45,000

Taking interest and depreciation at 10 per cent., we have for this item $4,500 per year, or about $12.50 per day.

The total cost per day of the plant, neglecting the wages of conductors and drivers and also repairs and interest on buildings and such portions of the motor-cars as have their equivalent in an ordinary street-car, will be:

Fuel, oil and repairs,	$82.50
Labor,	38.00
Interest and depreciation of machinery,	12.50
Total,	$133.00

In the case here considered the men work on very long shifts —say 13 hours per day. Ordinarily, if there is sufficient traffic to require the cars to be run at four-minute intervals during the busy part of the day, they will have to be kept running in the evening, and the number of working-hours per day will be increased to eighteen.

If the trips are so arranged that the same number of miles is made by the motors in a day of 18 hours as in one of 12—the motors running more frequently during the busy part of the day than in the evening and very early morning—the cost per day with two gangs of men at each station will be $\$38.00 \times 2 = \$76.00$ for labor; adding to this $95.00 for fuel, etc., gives $171.00 as the total expense, apart from other items which would be common to roads operated either by compressed air or by horses.

Comparison with Horse-Traction.—The writer has received an estimate for operating by horse-power a road of the above description, namely, a $3\frac{1}{2}$-mile, double-track line, running cars on a four-minute headway for 12 hours a day. It is as follows:

Cost of feeding 350 horses per day,	$105.00
20 stablemen at $1.75 per day,	35.00
1 stable-foreman,	2.50
1 stable-doctor,	3.00
4 hitchers at $1.50 per day,	6.00
Medicines,	2.00
Replacement of horses and acclimating,	37.50
Shoeing,	14.00
Total,	$205.00

In order to compare these figures with the results obtained for the Mekarski motors, we must add to the total sum the interest on the capital invested in horses. Assuming each horse to be worth $100, we have for the value of the horses $350 \times \$100 = \$35{,}000$; five per cent. on this is $1,750, or about $5.00 per day, making the total per day $205 + 5 = \$210$.

The following, therefore, is the comparison obtained:

Mekarski compressed air-motor cars for a day of 12 hours,	$133.00
" " " " " 18 "	171.00
Horse-traction for a day of 12 hours,	210.00

(Neglecting in all cases wages of conductors and drivers, and interest and repairs on buildings and cars.)

It must, however, be borne in mind that the figures for repairs of the Mekarski motors, given in the estimate, are for a plant that has been in operation a long time, and in all probability brought to its best condition in regard to attendance. If a new plant were started in the United States, inexperienced men would have to be put in charge of the motors, and in all probability the repairs for the first year would far exceed the figures above given. The extra wages that men could earn for running the motors over those of an ordinary driver are also left out of consideration. Again, men not used to running the motors would use the air with less economy than they now do at Nantes ; and probably the addition of 50 per cent. to the fuel-cost per mile, on account of dirty tracks, etc., will not be a sufficient margin. It is also probable that the pressure of air stored in the tanks will have to be made very much higher than is actually necessary for the trip, so that an extra amount of air can be drawn on in case of an emergency, and as this was not done at Nantes, at which place the reserve had to be thrown in toward the end of nearly every trip, an extra allowance should be made on this account. Taking all this into consideration, it may be that the coal consumed per motor per mile in the United States will be twice, or even three times, that at Nantes.

The greatest drawback to the Mekarski motor is to be anticipated when snow can not be kept from the rails; for as each motor has only a definite charge of air which may become exhausted in contending with the snow, the car may come to a standstill on the track. This condition would not, of course, occur on underground-lines, so that the objection does not require to be considered for cases of mine-haulage.

TABLE I.

Record of Mekarski Air-Motors at Nantes.

Station at which the motor was charged with compressed air.	Section of the line run over.	Time: Of starting from and arriving at main station.	Time: Of making intermediate stops, and of noting other data.	Duration of stop, in seconds.	Time required to start, in seconds.	Condition of track where motor is started.	Number of passengers.	Pressure of air in tanks. Kilos per sq. cent.: Battery.	Pressure of air in tanks. Kilos per sq. cent.: Reserve.	Remarks.
		A. M.	A. M.							
D	A to B	10.17					25	10	19	Started from Central Station A.
			10.17½	30	20	Level	26			
			10.20	15	15		27			
			10.21½	10		Level	28			
			10.23	15	15	Slight d'n gr'de	27			
			10.24	75			20	6	19	
			10.27	20	13		14			
			10.28	10			14			
			10.29½	5			13			
			10.30				12			
			10.31	10			10			
		10.32					10	3¼	19	Arriv'd at Main Ch'rging Sta. B.
B	B to A	10.37¼			12	Level	2	29⅔	29½	Started on another motor, fully charged with compressed air.
			10.39¼				2		26½	
			10.41				2	28	26½	Used air from the reserve until 10.39¼, and then started to use from the battery.
			10.44¼	50			14			
			10.44½	5			15			
			10.49	10			16			
		10.49½					16	24¼	26½	Arrived at Station A.
B	A to C	10.52½					10	24¼	26½	Continued run on same car.
			10.57½	30	12	Level	10	20¼	26½	
			11.00	60			10			On switch.
			11.02	10			9			
		11.04					9	17¼	26½	Arrived at Station C.
B	C to A	11.14½						17⅔	26½	Continued run on same car.
			11.16¼	90	17	Up slight grade		14½	26½	On switch waiting for other car.
										Pressure on engines at starting 4½ kilos per sq. cent.
			11.21½	20	15	On sharp curve	8			Pressure on engines at starting 11 kilos per sq. cent.
		11.26¼					8	12	26½	Arrived at Station A.
B	A to D	11.27½					8	12	26½	Continued run on same car.
			11.29	15	13	Level	10			Pressure on engines at starting 8 kilos per sq. cent.
			11.31½	5	17	Up slight grade	7			
			11.33½	20	15	Level	7	8		
			11.35¼	10			4			
			11.36½	5			5			
			11.38	10	20	Up slight grade and on curve	4	6½	26½	
			11.41				4			Slowed up on switch.
			11.41¼				4			Reserve thrown in for a short time on a level track.
		11.43					4	5	22	Arrived at Station D. Motor remains at this station to be recharged.
D	D to A	11.47½			15		5	17½	21	Started on another motor, charged at D to two-thirds of its capacity.
			11.51¼	5	15	Level curve	5			
			11.54¼	5			6			
			11.56	15			12			
			11.57	15			12	14	21	
			12.00	10	12	Level	13			
			12.02				12	12½		
			12.03	10			12			
		P. M.								
		12.04¼					12	11¾	21	Arrived at station A.

(CONTINUED.)

TABLE I—(CONTINUED).

Record of Mekarski Air-Motors at Nantes.

Station at which the motor was charged with compressed air.	Section of the line run over.	TIME: Of starting from and arriving at main station.	TIME: Of making intermediate stops, and of noting other data.	Duration of stops, in seconds.	Time required to start, in seconds.	Condition of track where motor is started.	Number of passengers.	Pressure of air in tanks. Kilos per sq. cent.: Battery.	Pressure of air in tanks. Kilos per sq. cent.: Reserve.	REMARKS.
		P. M.	P. M.							
D	A to B	12.07	.....	...	...	.....	12	11¾	21	Continued run on same motor.
			12.08½	...	...	.....	13	.....	.....	
			12.10	5	...	.....	13	.....	.....	
			12.13	30	...	.....	9	.....	.....	
			12.16½	...	...	.....	9	7	21	
			12.18½	10	...	.....	5	.....	.....	
		12.19	.....	...	...	.....	5	6	21	Arrived at the main station, B, where motor is re-charged.
			.....	.....	.....	.....	...	.....	.....	
	B to A	12.28½	.....	.....	17	Level.....	9	30	27	Started on another motor fully charged at main charging-station B.
			12.31	10	...	.....	10	.....	.....	
			11.31¼	5	...	.....	11	26½	27	
			12.35	40	...	.....	5	.....	.....	
			12.37½	5	...	.....	6	.....	.....	
			12.38¼	10	...	.....	7	.....	.....	
			12.39	.....	.....	.....	6	.....	.....	
			12.40	*240	10	Level.....	5	.....	.....	Delayed on switch.
		12.45½	.....	.....	.....	.....	5	22½	27	Arrived at Station A.
B	A to C	12.47	.....	.....	.....	.....	28	22½	.....	Started from Station A.
			12.48¼	5	...	.....	29	.....	27	
			12.51¼	.....	...	.....	29	20¼	.....	
			12.55½	15	...	.....	27	.....	27	
			12.57	10	...	.....	26	.....	.....	
		12.59½	.....	.....	.....	.....	26	15½	27	Arrived at Station C.
B	C to A	1.07½	.....	...	...	.....	5	15½	27	Started from Station C.
			1.13	30	...	.....	8	.....	.....	
			1.16	5	...	.....	8	11	27	
		1.18½	.....	...	...	.....	8	10	27	Arrived at Station A.
B	A to D	1.20¼	.....	...	...	.....	9	10	27	Started from Station A.
			1.20½	10	...	.....	12	.....	.....	
			1.24	...	...	.....	15	.....	.....	Slowed down for a short time to wait for cart to be removed from the track.
			.....	...	...	.....	...	.....	.....	
			1.25¼	...	...	.....	16	.....	.....	
			1.27	30	15	Level.....	14	7	24½	Reserve used at starting.
			1.30	20	...	.....	10	.....	22½	" " "
			1.32	45	18	Level.....	3	6	18½	" " "
			1.35	...	...	.....	3	.....	.....	Reserve used on level track.
		1.38	.....	...	...	.....	3	4½	16½	" " " "
D	D to A	1.39½	.....	...	...	.....	3	18	20	Started on another motor ch'gd to two-thirds of its capacity at supplementary station D.
			1.42½	10	20	Level.....	7	.....	.....	
			1.46	25	...	.....	16	.....	.....	
			1.47½	30	20	Level.....	18	.....	.....	Only 4 kilos per sq. cent. on engines at time of starting.
			1.49	25	...	.....	21	15	20	
			1.51	45	20	Level.....	30	.....	.....	
			1.54	10	22	Level.....	29	.....	.....	Only 6 kilos per sq. cent. on engines at starting.
			1.56	10	18	Level.....	27	.....	.....	
		1.56½	.....	...	...	.....	27	11¾	20	Arrived at A. Nearly all of the passengers left the car, therefore record was not continued.
			.....	...	...	.....	...	.....	.....	
			.....	...	...	.....	...	.....	.....	
D	A to B	4.53	.....	...	...	.....	18	12¼	21½	Took another car at A.
			4.53½	20	...	.....	21	.....	.....	
			4.56	5	12	Level.....	23	.....	.....	
			4.56¾	5	...	.....	22	.....	.....	
			4.59½	10	...	.....	22	.....	.....	
			5.03	50	10	Level.....	16	9¼	21½	9 kilos per sq. cent. on engines at starting.
			.....	10	15	Level.....	14	.....	.....	
		5.06	.....	...	...	.....	14	8	21½	5 kilos per sq. cent. on engines at starting.

* This delay is not included in the time expended in intermediate stops.

TABLE II.—*Totals and Averages of the Records Given in Table I.*

Station at which the motor was charged with compressed air.	Section of the line run over.	Per Single Trip. Average number of passengers.	Total number of intermediate stops.	Distance traveled in miles.	Time required to make trip. Including intermediate stops.	Time required to make trip. Excluding intermediate stops.	Miles per hour. Including intermediate stops.	Miles per hour. Excluding intermediate stops.	Per Single Charge of Air. Average number of passengers per trip.	Total number of intermediate stops.	Distance traveled, in miles.	Miles per hour. Including intermediate stops.	Miles per hour. Excluding intermediate stops.
D	A to B	20	10	1.75	15 minutes.	11 m. 50 sec.	7.0	8.9					
B	B to A	8	3	1.75	12 m. 15 sec.	11 m. 10 sec.	8.6	9.4					
B	A to C	9.8	3	1.40	11 m. 30 sec.	9 m. 30 sec.	7.3	8.8					
B	C to A	8	2	1.40	11 m. 45 sec.	9 m. 55 sec.	7.1	8.5					
B	A to D	6.1	6	2.05	15 m. 30 sec.	14 m. 25 sec.	7.9	8.5	8.0	14	6.6	7.76	8.80
D	D to A	9.8	6	2.05	16 m. 45 sec.	15 m. 45 sec.	7.3	7.8					
D	A to B	10.4	3	1.75	12 m.	11 m. 15 sec.	8.7	9.3	10.1	9	3.8	7.93	8.44
B	B to A	7.8	6	1.75	15 m.	11 m. 50 sec.	8.1	8.9					
B	A to C	28	3	1.40	12 m. 30 sec.	12 m.	6.7	7.0					
B	C to A	6.5	2	1.40	11 m.	10 m. 25 sec.	7.6	8.1					
B	A to D	9.2	4	2.05	17 m. 45 sec.	16 m.	6.9	7.7	12.9	15	6.6	7.30	7.88
D	D to A	16.7	7	2.05	17 m.	14 m. 25 sec.	7.2	8.5					
D	A to B	18.1	6	1.75	13 m.	11 m. 20 sec.	8.1	9.3	17.4	13	3.8	7.60	8.85
Totals for all runs		158.4	*61	22.55	179 minutes.	159 m. 50 sec.	98.5	110.7					
Average		12.2					7.58	8.52					

* Number of stops, including those at main stations, 74.

Table III.

Record of the Mekarski Air-Motors at Vincennes and Nogent, near Paris.

The Motors draw a car on the main line, and run alone on the branch line.

Station at which the motor was charged.	Section run over.	Time: Of starting from and arriving at main stations.	Time: Of making intermediate stops, and of noting other data.	Duration of stop, in seconds.	Time required to attain the maximum speed, in seconds.	Condition of the track.	No. of passengers, approximate.	Pressure of air in tanks. Kilos per sq. cent.: Reserve.	Pressure of air in tanks. Kilos per sq. cent.: Battery.	Remarks.
		P. M.	P. M.							
D	D to A	12.03¼	.....	.....	.....	.....	80	44	36½	Motor drawing a car, starting from Station D.
			12.09	30	30	Up slight grade	80	.....	.....	
			12.12	10	40	" " "	80	43	21	8 kilos per sq. ct. on engines at starting. Reversing lever last notch from center for 20 sec., and on first notch afterwards.
			12.13	*240	.....	Heavy grade...	100	.....	.....	12.15. Heavy grade. Reserve used. Went up the hill slowly with 16 kilos per sq. cent. on engines.
			12.22	20	25	Slight d'n gr'de	100	.....	.....	Reversing lever in 2d notch from center.
			12.25	10	20	" " "	100	.....	.....	
			12.26	20	25	Down grade.....	100	.....	.....	Only 3 kilos per sq. cent. on engines at starting.
			12.28	.....	.....	Heavy d'ngr'de	100	.....	.....	Longest hill. 5 kilos per sq. cent. constantly on brakes.
		12.29½	.....	.....	.....	.....	100	29	13½	Arrived at Station A.
D	A to E	12.31½	.....	.....	.....	.....	45	29	13½	Contin'd run on same motor.
			12.35	25	30	Slight up grade	44	.....	.....	6 kilos per sq. cent. on engines at starting.
			12.37	.....	.....	.....	44	.....	.....	
			12.38	05	30	Slight up grade	40	.....	.....	
			12.41	10	45	" " "	40	.....	.....	
		12.41½	.....	.....	.....	.....	40	.....	.....	Arrived at E. Motor disconnected from car and run into station to be charged.
			.....	.....	.....	.....	.....	25½	7¼	
E	E to B	1.12½	.....	.....	.....	.....	75	.....	.....	Started on another motor, drawing a car.
			1.17	55	20	Level.....	75	.....	.....	11 kilos per sq. ct. on engines at starting. Revers'g lever in last notch fr'm cent'r for 10 sec. and in first 10 sec.
			1.18	5	20	Down grade.....	70	.....	.....	10 kilos per sq. cent. on engines. Reversing lever in 1st notch from center.
			1.21	.....	.....	Up grade and on a curve.....	70	.....	.....	
		1.23	.....	.....	.....	.....	70	39	14	Arrived at B.

(Continued.)

* This delay is not included in the time expended in intermediate stops.

TABLE III.—(CONTINUED.)

Record of Mekarski Air-Motors at Vincennes.

Station at which the motor was charged.	Section run over.	TIME: Of starting from and arriving at main stations.	TIME: Of making intermediate stops, and of noting other data.	Duration of stop, in seconds.	Time required to attain the maximum speed, in seconds.	Condition of the track.	No. of passengers, approximate.	Pressure of air in tanks. Kilos per sq. cent.: Reserve.	Pressure of air in tanks. Kilos per sq. cent.: Battery.	REMARKS.
		P. M.	P. M.							
E	B to E	1.28½	.....	.....	.....	.....	10	39	14	Returned from B.
			1.30	10	.....	.....	12	.....	.....	
			1.32	10	25	Slight up gr'de.	15	.....	.....	
			1.33	10	.....	.....	20	.....	.....	
			1.33¼	5	20	Slight up gr'de.	20	.....	.....	Reserve thrown in.
			1.36	25	.....	.....	25	.....	.....	
			1.37	.....	.....	.....	25	33	6	
		1.39½	.....	.....	.....	.....	25	29¼	5	Arrived at E. Motor disconnected from car and run into the station to be charged.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
E	E to A	1.44½	.....	.....	.....	.....	40	43	41½	Took another motor, drawing a car. Started by using air from reserve, and afterwards changed over to battery.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			1.47	5	20	.....	42	.....	.....	
			1.50	.....	.....	.....	42	39	32	1.48 P. M. Going up hill. 10 k. on engine lever in 2d notch from center.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			1.51	35	.....	.....	50	.....	.....	This motor went on to Vincennes.
		1.55	.....	.....	.....	.....	50	39	27	
			.....	.....	.....	.....	...	.....	.....	
E	A to C	2.37½	.....	.....	.....	Down hill.....	45	41	32	Started on a motor that came from E without draw'g a car. Did not draw car on branch line.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			2.40	10	.....	Down hill.....	44	.....	.....	
			2.42	40	.....	Slight d'n gr'de	42	.....	.....	
			2.43	15	.....	Level	41	.....	.....	
			2.45	45	.....	.....	40	.....	.....	
		2.46¼	.....	.....	.....	.....	40	40¾	26½	Arrived at C.
E	C to A	3.15¾	.....	.....	15	Level.....	21	40¾	26½	Returned on same motor to A.
			3.17	35	.....	.....	36	.....	.....	
			3.18	15	.....	.....	36	.....	.....	
			3.20	10	.....	.....	37	.....	.....	
			3.21	20	.....	.....	38	.....	.....	
		3.25	.....	.....	.....	.....	38	36	15	Arrived at A. Motor ret'd to E.
E	A to D	3.37½	.....	.....	.....	Up longest hill.	70	41	25	Motor drawing car. When gr'de was reached, air-pressure of 16 k. per sq. ct. was used, and the lever was at 2d notch.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			3.39½	.....	.....	Up longest hill.	70	.....	.....	14 k. on engine, rev. lever in 2d notch; motor going about as fast as a horse in a slow walk.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			3.40	.....	.....	Up longest hill.	70	.....	.....	
			3.41	20	25	Slight up grade	70	.....	.....	Nearly at the top of the grade when this stop was made. It required 8 k. per sq. cent. on the brakes to hold the train from starting backw'd down the hill.
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			.....	.....	.....	.....	...	.....	.....	
			3.42	10	.....	Down grade.....	75	.....	.....	
			3.45½	.....	.....	Short up grade	75	37	8	
			3.46	...	...	Down grade...	75	.....	.....	
			3.48	150*	.....	.....	70	38	8	
			3.53	40	30	Level.....	65	.....	.....	
		3.59	.....	.....	.....	.....	65	33¾	6¼	Arrived at D.

* This delay is not included in the time expended in intermediate stops.

Table IV.

Totals and Averages of the Records given in Table III.

	Motor drawing a car on the main line D A E B.	Motor alone on the branch line A B.
Total distance run in the round trip	11.9 miles.	2.68* miles.
Total number of stops made	22	8
Number of times that the motor is charged to make a round trip	3	1
Average number of passengers, approximate	59	38
Time to make run, including all stops except at the main stations and two delays mentioned in Tables I. and III.	83 m. 15 s.	18 m. 15 s
Time, excluding all stops	77 m. 5 s.	15 m. 5 s.
Miles traveled per hour, including stops	8.58	8.81
" " " " excluding stops	9.26	10.65

NOTES ON THE PROGRESS OF MINING IN CHINA.

BY ELLIS CLARK, PHILADELPHIA, PA.

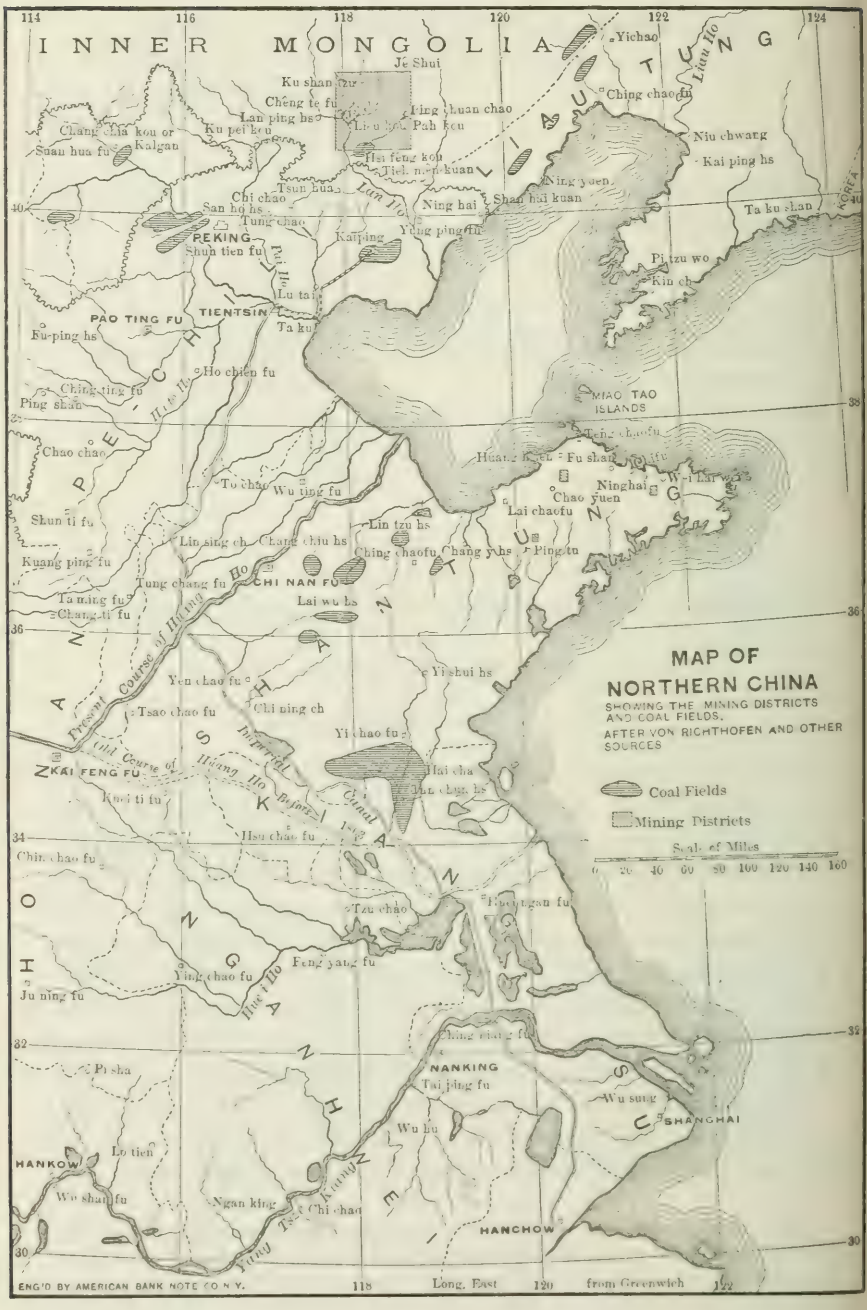
(New York Meeting, September, 1890.)

WITHIN the last ten years the progressive party of China, headed by Li Hung Chang, the Viceroy of Chi-Li, has been making great efforts to develop the mining resources of that country, and particularly those that are within reasonable distances of the sea-board. Attention was first paid to the coal-deposits, especially those at Kai-Ping, which have been placed on a profitable basis with an output of 1500 tons per day, under the energetic management of Mr. Claude Kinder, M.I.C.E., and the successful financiering of Tong King Sing. These coal-mines were described in a report by Kwong Yung Kwang, edited as a paper for the Institute (*Trans.*, xvi., 95.)

The attention of the leaders of the progressive party has for the past five years been turned also toward the development of the precious and related metals—gold, silver, copper and lead. De-

* This single motor runs, in addition to this length of 2.68 miles, the double distance of A from E, which distance I do not know, but should judge it to be about $1\frac{1}{2}$ miles—so that the total distance run by the motor without re-charging is about $2.68 + 2 \times 1\frac{1}{2} = 5.68$ miles.

FIG. 1.


INNER MONGOLIA
SHANTUNG
PECHILI
HONAN
KIANGSU
NGANHWEI
HUPEH
KIANGSI
PEKING
Shun tien fu
TIENTSIN
PAO TING FU
Fu-ping hs
Ho chien fu
Ching ting fu
Ping shan
Chao chao
Shun ti fu
Kuang ping fu
Ta ning fu
Chang ti fu
Tung chang fu
CHI NAN FU
Lai wu hs
Yen chao fu
Tsao chao fu
Chi ning ch
KAI FENG FU
Kwei ti fu
Present Course of Hwang Ho
Old Course of Hwang Ho
Imperial Canal
Grand Canal
Hsu chao fu
Chin chao fu
Ju ning fu
Ying chao fu
Feng yang fu
Tzu chao
Pi sha
NANKING
Tai ping fu
Wu hu
Wu sung
SHANGHAI
HANKOW
Lo tien
Wu shan fu
Ngan king
Tai Chi chao
HANCHOW
Yang Tze Kiang
Hwai Ho
Yellow Sea
Je Shui
Ku shan tsu
Cheng te fu
Lan ping hs
Kalgan
Chang chia kou or
Suan hua fu
Ku pei kou
Tsun hua
Chi chao
San ho hs
Tung chao
Kai ying
Ling huan chao
Pah kou
Hsi feng kou
Tieh mei kuan
Ning hai
Shan hai kuan
Ying ping fu
Yichao
Ching chao fu
Niu chwang
Kai ping hs
Ning yuen
Ta ku shan
Pi tzu wo
Kiu ch
MIAO TAO ISLANDS
Teng chao fu
Fu shan
Ninghai
Chao yuen
Lai chao fu
Lin tzu hs
Ching chao fu
Chang yuh s
Ping tu
Yi shui hs
Yi chao fu
Lai cha
Tai chang hs
Tsao chao fu
Liu Ho
Liao Ho
KOREA
MAP OF NORTHERN CHINA
SHOWING THE MINING DISTRICTS AND COAL FIELDS.
AFTER VON RICHTHOFEN AND OTHER SOURCES
Coal Fields
Mining Districts
Scale of Miles
0 20 40 60 80 100 120 140 160
ENG'D BY AMERICAN BANK NOTE CO N.Y.
Long. East from Greenwich
114 116 118 120 122 124
40 38 36 34 32 30

posits of the ores of these metals are found at Chi-Chao, on the Yang-Tsze-Kiang, shown on the accompanying map, Fig. 1; in the Shantung promontory, at Ning-Hai, Ping-Tu and Chao-Yuen; in the Ping-Chuan-Chao district, the Jê-Shui district, and the Jê-Hol district, the latter including the silver-lead mines of Yen-Tung-Shan and Ku-Shan-Tzu. The writer has visited all of these places professionally.

Chi-Chao District on the Yang-Tsze-Kiang.

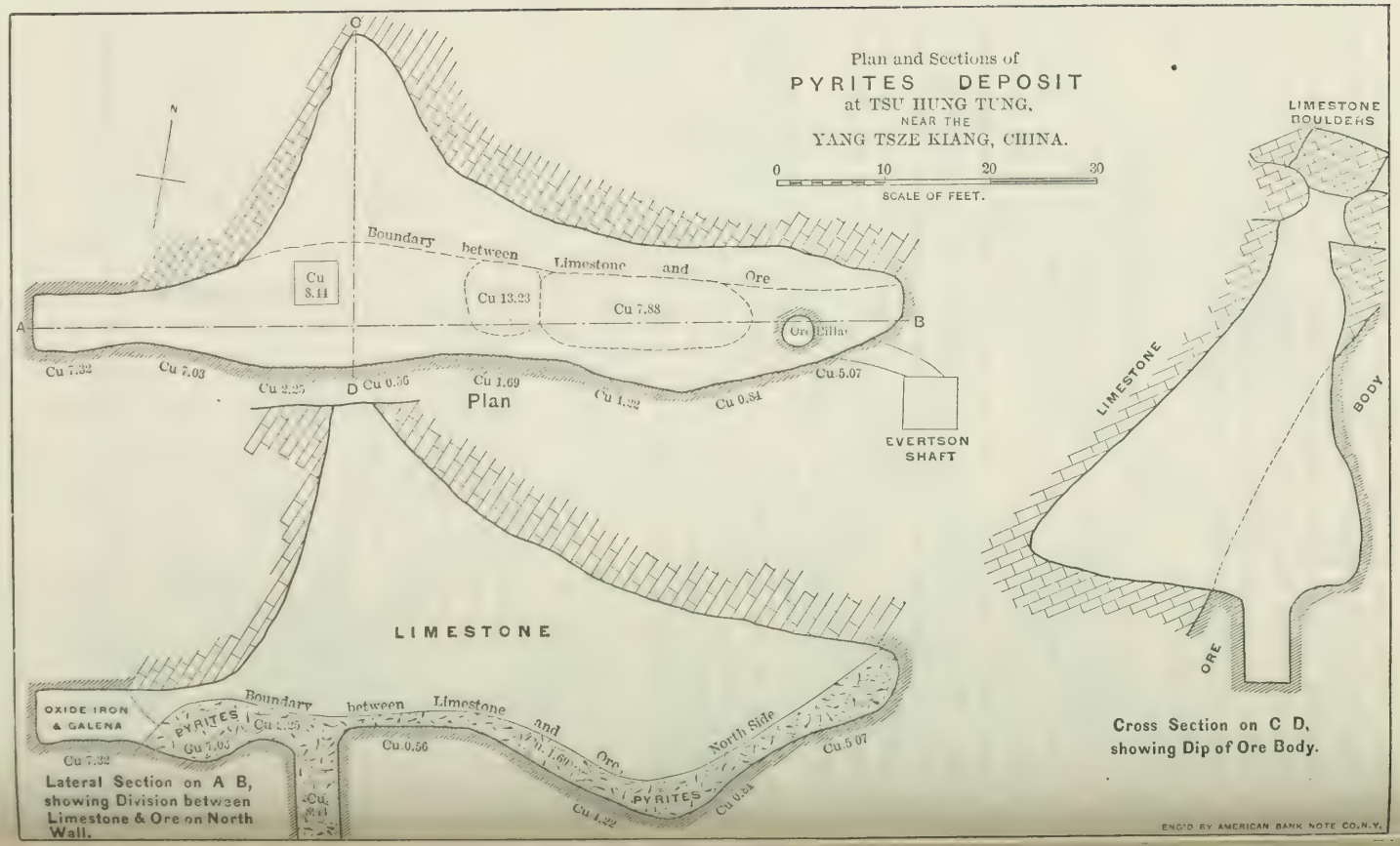
This district is reached by travelling 300 miles up the Yang-Tsze river on one of the well-appointed steamers running between Shanghai and Hankow, and taking a house-boat at Chi-Chao to Lao-Pao-Chi, where the offices and sulphuric acid works are situated. The mines are 20 miles distant, and can be reached either by trail or Bamboo raft. The Chinese company has worked and partially developed several deposits, the principal of which are the Tsu-Hung-Tung pyrites-deposit, the Nao-Lung-Ching hematite deposit, and the lead-zinc vein at Tung-Chi-Lung.

Tsu-Hung-Tung—The first of these (Fig. 2) is the most interesting of the series and consists of a lenticular deposit of copper and iron-pyrites, oxide of iron and a little galena, developed to a length of 80 feet, a thickness of 5 to 12 feet, and a depth of 40 feet. The ore as sampled by the writer in zones of 10 feet gave:

No.	Character of the Ore.	Copper, Per cent.
1	Red oxide of iron, pyrites, galena in specks	7.32
2	Oxidized pyrites, a little galena	7.03
3	Micaceous iron and pyrites	2.25
4	Red oxide of iron	0.56
5	Red oxide of iron, galena, pyrites	1.69
5A	Greenish pyrites	13.23
6	Pyrites and red oxide of iron	4.22
7	Red oxide of iron and pyrites	0.81
8	White pyrites	5.07
9	Pyrites	7.88
10	Pyrites and galena in specks	8.14

The deposit has a strike nearly east and west with a dip to northward of 70 degrees.

FIG. 2.


Plan and Sections of
PYRITES DEPOSIT
at TSU HUNG TUNG,
NEAR THE
YANG TSZE KIANG, CHINA.
0 10 20 30
SCALE OF FEET.
N
C
D
A
B
Boundary between Limestone and Ore
Cu 8.44
Cu 13.23
Cu 7.88
Ore Pillar
Cu 7.32
Cu 7.03
Cu 2.25
Cu 0.56
Cu 1.69
Cu 1.22
Cu 0.84
Cu 5.07
Plan
EVERTSON SHAFT
LIMESTONE
OXIDE IRON & GALENA
PYRITES
Boundary between Limestone and Ore
North Side
Lateral Section on A B, showing Division between Limestone & Ore on North Wall.
LIMESTONE BOULDERS
LIMESTONE
ORE BODY
ORE
Cross Section on C D, showing Dip of Ore Body.
ENG'D BY AMERICAN BANK NOTE CO. N.Y.

At the time of the writer's visit no copper in quantity had ever been made from the ore, the opening having been made years before for the purpose of distilling sulphur from the pyrites. Some ore was then shipped to the sulphuric acid works of Major Brothers, Limited, at Shanghai, the acid-works at Lao-Pao-Chi having ceased operations, but after an extended trial it was found that the ore contained too much arsenic for the profitable manufacture of sulphuric acid. The Evertson shaft, shown at the eastern end of the working, has been sunk under and parallel to the vein; it is 70 feet deep and terminates in a rich brown oxide of iron containing traces only of copper.

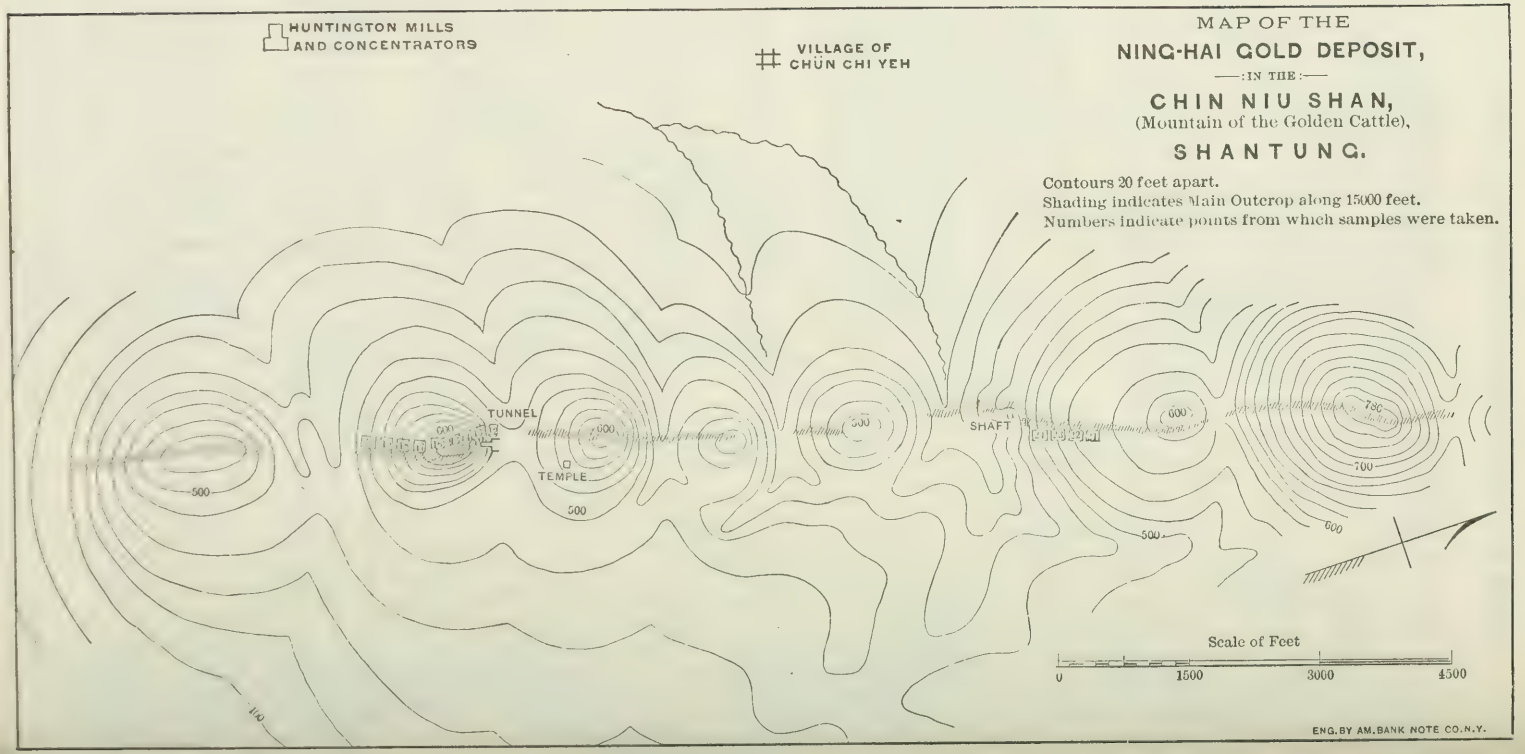
Nao-Lung-Ching.—The iron-deposit of Nao-Lung-Ching is about two miles northwest of Tsu-Hung-Tung. The ore consists of hematite which has been probably derived from pyrites, as specks of this material can be seen by careful inspection. The deposit has been opened by four superficial shafts at distances of 75 feet, and although the growth of weeds and creepers prevent the exact location of its boundaries from being ascertained, it is probably 400 feet long, with a width of 30 feet. The sulphur in this ore will probably prevent its use for the manufacture of iron.

Tung-Chi-Lung.—The lead-zinc opening of the Tung-Chi-Lung is situated 27 miles southeast of Tsu-Hung-Tung. It consists of a heavily mineralized fissure-vein between blue-gray slate walls, having an east and west strike and an almost perpendicular dip. In the shaft, which at the time of the writer's visit was 11 feet deep, the foot-wall was shown, but the hanging-wall had not been cut. Resting on the foot-wall are 1 foot 6 inches of zinc-blende and gangue, followed by 2 feet of copper- and iron-pyrites and blende, and 4 feet of galena, blende and gangue. An average sample across the vein gave 3.6 per cent. lead and 1 ounce of silver per ton. A sample of galena and blende taken from a pile of 10 tons gave 18.5 per cent. lead, and 6 ounces of silver.

There are several coal openings in the neighborhood of Lao-Pao-Chi which supply a rather poor, soft coal for the local blacksmith-consumption and the junk-freighters who carry it up and down the Yang-Tsze-Kiang.

A rather peculiar occurrence of coal is found at an opening 3 miles east of Tsu-Hung-Tung, on the side of a limestone mountain, in an almost inaccessible position, about 500 feet above the valley. The peculiarity of this wavy and contorted seam is its occurrence in limestone. The mine is entered by a low, narrow, tortuous gang-

FIG. 3.


HUNTINGTON MILLS AND CONCENTRATORS
VILLAGE OF CHÜN CHI YEH
MAP OF THE
NING-HAI GOLD DEPOSIT,
—:IN THE:—
CHIN NIU SHAN,
(Mountain of the Golden Cattle),
SHANTUNG.
Contours 20 feet apart.
Shading indicates Main Outcrop along 15000 feet.
Numbers indicate points from which samples were taken.
TUNNEL
TEMPLE
SHAFT
500
600
700
Scale of Feet
0
1500
3000
4500
ENG. BY AM. BANK NOTE CO. N.Y.

way with a slope of 30 degrees, which is that of the coal-bed and the limestone. The opening extends for 150 feet, and in this length the coal-bed has been crumpled into irregular lenses, varying in thickness from 6 inches to 4 feet. The coal is a compact, glistening semi-anthracite, rather high in ash and very much broken up by pressure into nodules and fine coal. The roof and floor are both limestone, the usual fire-clay being entirely absent. The seam is probably too irregular and too much broken to be economically worked on a large scale, but when the coal is ignited, which is a matter of some difficulty, it burns well in an ordinary stove.

Mr. J. C. F. Randolph, a member of the Institute, has analyzed a sample of this coal, but at the time of his visit to China in 1885 he did not personally examine the opening from which it was taken. He gives for the composition:*

	Per cent.
Moisture and volatile Combustible,	16.5
Fixed Carbon,	72.5
Ash,	11.0

The Shantung Promontory.

This promontory, in which the Shantung mining-district is situated, is about 450 miles north of Shanghai (Fig. 1). The three points at which most of the work has been concentrated are Ning-Hai, Ping-Tu and Chao-Yuen.

The Ning-Hai Gold District.—This district lies 30 miles southeast of the treaty-port of Chifu. Supplies and heavy articles are taken to the mines from the steamer anchorage near Ning-Hai city, 15 miles distant. The geological formation of the district consists of granite, gneiss and mica-slates, with occasional interbedded deposits of white crystalline marble and extensive quartzite ledges, which mark out some of the most prominent topographical features of the country. Such a ledge forms the backbone of the Chin-Niu-Shan, or Mountain of the Golden Cattle, which is about 15,000 feet long. By reference to the map, Fig. 3, it will be seen that this quartz-ledge rises out of a plain to the south 400 feet above sea-level, and forms a series of cone-shaped hills whose summits reach an elevation of 500 to 800 feet above the sea, while the gorges between them cut down nearly to the level of the plain. A tunnel has been opened on the ledge at the point shown on the map. Its course is south for 200 feet, cutting through the quartz ledges, of

* *Transactions* xv., 112.

FIG. 4.

LEGEND:
1. Hoisting Works.
2. Main Shaft.
3. Shafts on Vein.
4. Tunnel.
5. Twenty Stamp Mill.
6. Assay Office.
7. Chlorination Works.
8. Iron Foundry.
9. Pump House.
10. Foundations.
11. 1500 Tons Concentrates.
12. Coal Yard.
13. Dwelling Houses.
14. Yamen.
15. Dynamite Factory.
16. Foreign Dwelling House.
17. Water Reservoir.
18. 18 Inch Tramway.
Map of the Ping Tu Mine and Mill.
Shantung, China.
Scale
0 50 100 200 300 400 Ft
Eng. by Am. Bank Note Co. N.Y.

which there are two at this point, and through the interlying granite and white kaolin selvages. The vein-quartz is hard, and white to yellow in color; the sulphurets it contains are sometimes cubical crystals of pyrites and sometimes massive, forming pockets in the quartz. In its length the ledge shows a thickness varying from 10 to 80 feet; it is gold-bearing along its entire outcrop.

Besides the tunnel, extensive prospecting has been done along the whole length of the ledge, both recently under European direction and by the Chinese in olden times. The writer took and assayed twenty-five samples from this large deposit, which ran from 77 cents to 5.85 dollars per ton. One assay of selected quartz from a pocket gave 24.75 dollars; the average of ten samples from the tunnel is 1.55 dollars; the average of thirteen samples from the outcrop is 2.19 dollars; and the general average of all the samples is 1.87 dollars per ton, which is too low to warrant the erection of any but experimental works. The plant on the ground consists of two 3-foot Huntington mills and two Frue vanners.

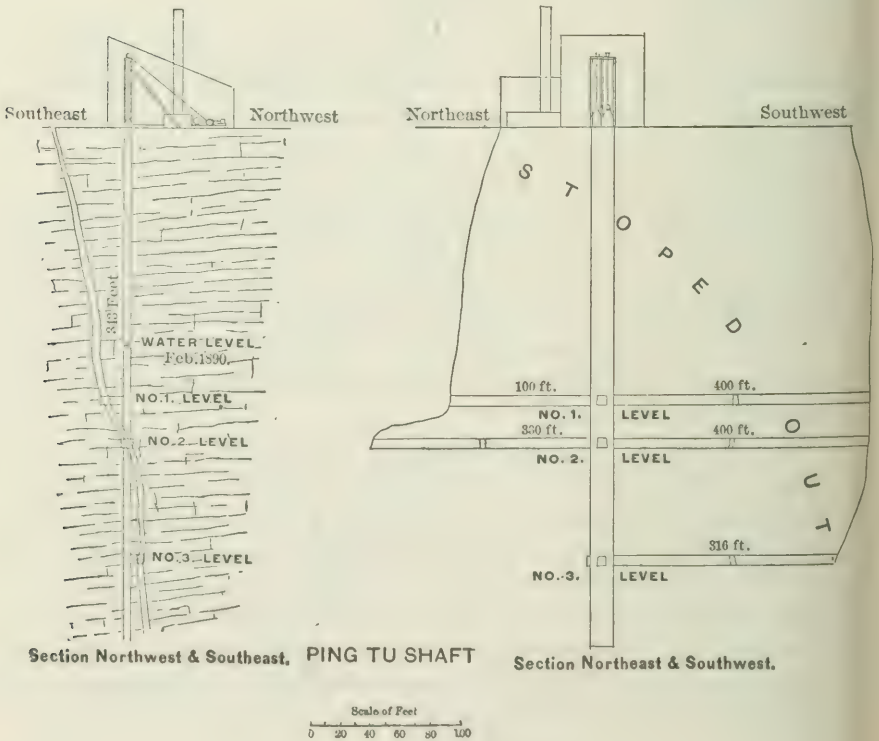
The Ping-Tu Gold District.—The Ping-Tu group of mines is situated in the western part of the Shantung promontory, 30 miles from a steamer-anchorage and 45 miles southwest of the Chao-Yuen group of mines. The immediate geological formation is gneiss, which, at a distance of 10 miles north and south of Ping-Tu, is cut by zones of highly metamorphosed limestone. The local topography is shown on Fig. 4. The two hills which rise to the height of 220 feet above the shaft-house are composed of distinctly bedded yellowish gneiss striking east and west, and intersecting the course of the vein, which is northeast and southwest, Fig. 5. This leads to the belief that the Ping-Tu deposit is a true fissure-vein, one of the few that occur in eastern China, and this theory receives further support from finding breccia in the vein-matter.

The mines have been worked by the Chinese for a number of years. Under European auspices and the superintendence of Mr. H. M. Beecher, of England, the systematic development of the mine at Ping-Tu was begun in 1886, and a twenty-stamp mill made by the Union Iron Works, of San Francisco, with eight Frue vanners and eight Hendy concentrators, was erected. After the mill was in operation, the hoisting-arrangements were found inadequate to keep it supplied with ore, and a new fifty-thousand-dollar hoisting-plant, with Ingersoll rock-drills, was erected under the superintendence of Colonel Ellsworth, of San Francisco.

The upper portion of the vein was quite rich. The writer was informed that the first clean-up amounted to $10,000. When in full operation the mine had a staff of twelve Americans, a very large Chinese clerical force, with its numerous servants, and the usual crowded labor underground and on the surface. As the mine increased in depth, the ore carried less free gold and more sulphurets, and the amount of amalgam that could be scraped from the plates rapidly decreased. The foreign employees, always more or less dissatisfied with an unprofitable mine, gradually left, and their places were filled with Chinese who had but imperfect knowledge of amalgamation and the general working of a stamp-mill. In the fall of 1889 supplies fell short, and, with the failure to receive coal, the mine-pumps were stopped, and the workings allowed to fill with water.

The vein varies in width from 4 to 10 feet, and consists of quartz mineralized in the upper levels with carbonate of iron, and in depth

FIG. 5.



with pyrites, some galena and a little copper-pyrites. In two years the product of the mill, with its sixteen concentrators, amounted to 1500 tons of concentrates, samples of which, taken from different parts of the pile, assayed as follows:

Concentrates from Ping-Tu.

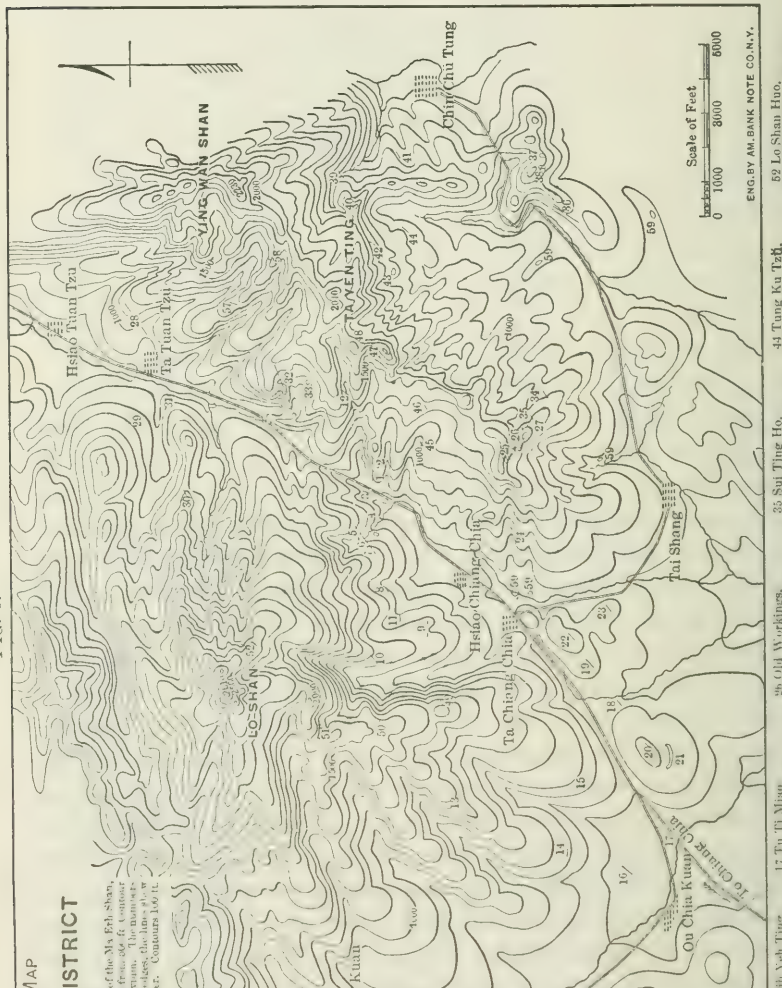
No.	Gold, Ounces per ton.	Silver, Ounces per ton.	Total Dollars per ton.
1	1.4	2.15	31.09
2	0.75	1.70	17.20
3	0.95	2.40	22.03
4	2.00	5.30	46.64
5	3.20	4.80	70.94
6	2.70	2.30	58.11
7	1.30	1.90	28.77

A sample taken by the writer as an average of the pile of 1500 tons, assayed gold, 36.31 dollars; silver, 2.94 dollars; total, 39.25 dollars per ton. A small chlorination-works, furnished with one reverberatory *Fortschaufelungsofen*, one chlorine-generator, and two chlorinating and precipitating vats has been erected. Its daily capacity is 1 ton, and about 100 tons of ore have been treated; but the process, though at first conducted under European supervision, was not a success, as the residues contained 26.87 dollars in gold and 10.70 dollars in silver per ton, so that if the richest concentrates—sample 5 of the assay, for example—were treated, about half the precious metals would be left in the residues. It is possible that the comparative coarseness of the concentrates and the low fineness of the gold, and large proportion of silver may be responsible for this failure, as according to Eissler (*Metallurgy of Gold*, p. 174), "gold of very low fineness, containing 40 to 50 per cent. silver, will probably resist the chlorination, unless it is in the finest state of pulverization, as the silver forms an impenetrable coating of chloride of silver which can be dissolved only by a chlorinated solution of salt, which dissolves the chloride of silver and also that of gold."

The Ping-Tu mine is probably better equipped than any other metal-mine in North China; besides the fine hoisting-works and pumping-machinery, it has an 18-inch tramway, 1600 feet long, running to the mill, a well-appointed assay-office, with a quicksilver retort, pumping-machinery for the stamp-heads, and a ditch supplying water to the concentrators; a dynamite factory, with a capacity

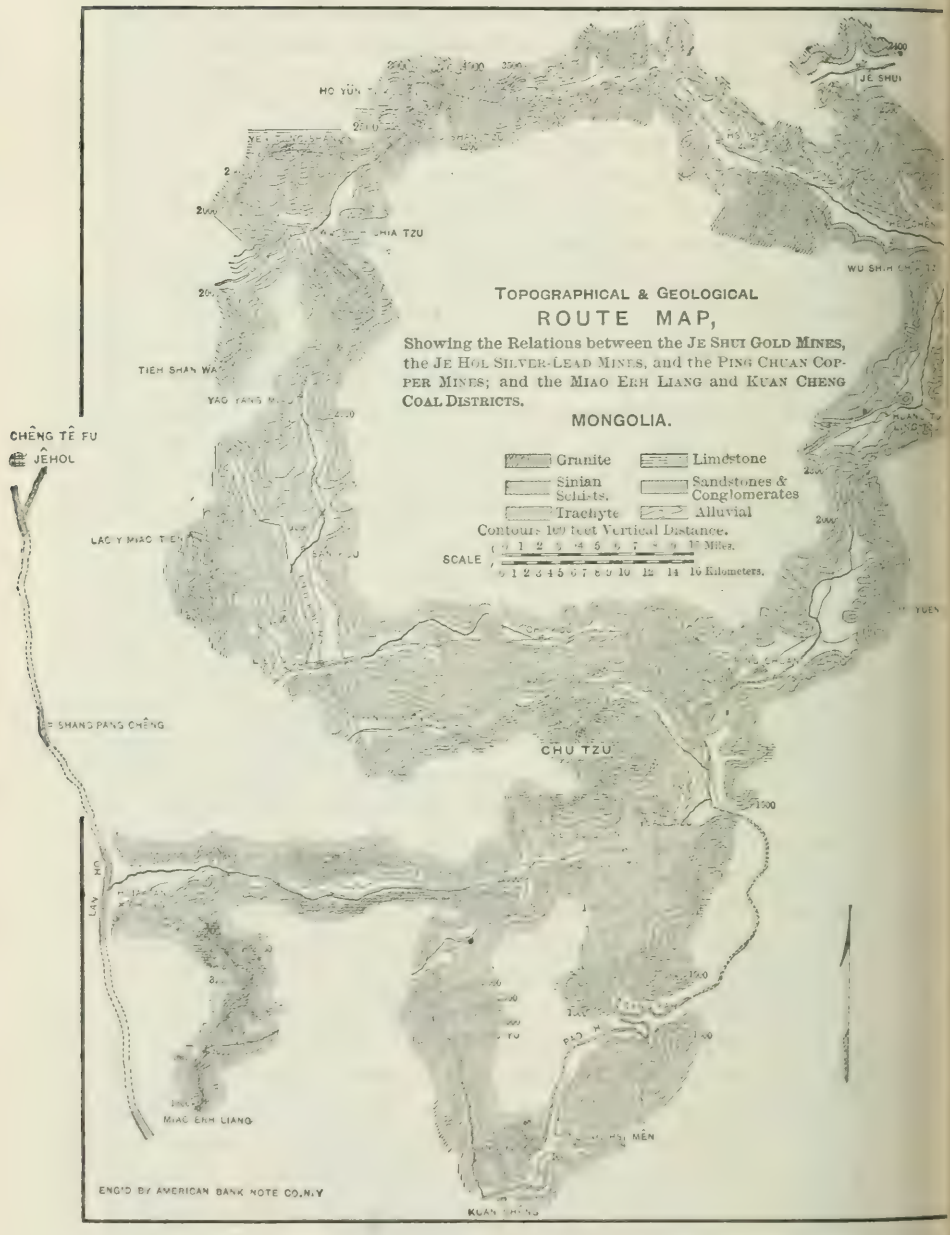
of 600 pounds per day and a foreign two-storied house with glass windows and board floors, which is a structure rarely met with in China outside the treaty-ports and missionary-stations.

FIG. 6.


RECONNOISSANCE MAP
— OF THE —
CHAO YUEN GOLD DISTRICT
SHANTUNG.
MA ERH SHAN
LO SHAN
YUNG MAN SHAN
TAYEN TING
Hsiao Kuan Kuan
Hsiao Chiang Chia
Ta Chiang Chia
Hsin Chia
Tai Shang
Ou Chia Kuang
To Chinkiang
Hsiao Tuan Tzu
Ta Tuan Tzu
Chin Chu Tung
Scale of Feet
0 1000 3000 6000
ENG. BY AM. BANK NOTE CO. N.Y.
1 Ling Lung Shan Temple.
2 Ling Lung Shan Old Work.
3 Ta Pao Kuang Shafts.
4 Ta Pao Kuang Ting.
5 Hsiao Pao Kuang Cut.
6 Hsiao Pao Kuang Ting.
7 Fu Tzu Tien.
8 Chin Wan Po Pei.
9 Old Workings.
10 Pei Shih Yeh Ting.
11 Pei Shih Shan.
12 Ling Lung Tzu Pei Ting.
13 Pei Shih Shan Ting.
14 Large Outcrop White Quartz.
15 Hsiao Chin Pei.
16 Ta Ma Chien.
17 Tu Ti Miao.
18 Pei Shih Shan.
19 Old Workings.
20 Mao Yü Ting.
21 Large Outcrop White Quartz.
22 Nan Chang Pei.
23 Chu Te Wan.
24 Pan Chia Yen.
25 Tai Shang Pei Pien.
26 Old Workings.
27 Tien Ling Tzu.
28 Yü Hsi Ling Tzu.
29 Niu Hang Kuang.
30 Mao Yü Ting.
31 Shou Yen Po.
32 Tsao Tzu Kuang.
33 Tsao Tzu Kuang, Nan Pang.
34 Chang Chung Yang.
35 Sui Ting Ho.
36 Ta Ting Ho.
37 Po Tou Ching.
38 Po Tou Ching, Hsi Pang.
39 Huo Shih Pen.
40 Sa Hang Chien.
41 Shih Kuo, Nan Kou.
42 Tung Chang.
43 Chih Yen.
44 Tung Ku Tzŭ.
45 Ta Ma Chien.
46 Ta Ma Chien, Old Work.
47 Ling Lung Ting.
48 Hu Chien, Hsi Ting.
49 Dark Granite Dyke.
50 Shih Wu Kou, Hsi Ting.
51 Pan Hsien Tung.
52 Ta Ma Chien, Hsi Ting.
53 Ta Ma Chien, Po Pei.
54 Ta Ma Chien, Hsi Ting.
55 Flagstone Quarries.
56 Hsi Nan Ting.
57 Yo Nan Hsien.
58 Chin Hsien Fei.
59 Gold Washings.
60 Temple.

The Chao-Yuen Gold District.—This district is situated 25 miles from the steamer anchorage of Hung-Kou and 18 miles from the large town of Huang-Hsien. The topography, as revealed in Fig. 6, is extremely rugged. The granite flanks of the Lo-Shan range rise out of the great Shantung plain, which, from an elevation of 500 feet at the base of the range, stretches away to the north and south in gradual descent to the sea. The rise of the range is rapid, the culminating points, within a few miles of the base, being the Lo-Shan, at 2610 feet, and the Ying-Wan-Shan, at 2230 feet. The bedding of the granite has a general strike of N. 45 E., and the quartz ledges which are all apparently bed-veins, take the same general trend, ranging from N. 30 E. to east. No regular mining has been done, although prospecting has been carried on extensively. The best-developed ledges are the Ling-Lung-Shan, the Hung-Chin, and the Hei-Shi-Yeh-Ting. Ling-Lung-Shan is a large quartz-ledge varying in width from 50 to 75 feet with an east-and-west course and a dip of 80° to the north. About 400 feet from the bed of the stream this ledge is faulted and thrown 100 feet to the northward. From the valley a tunnel has been driven for 75 feet through white and red quartz containing pyrites. Assays of two samples from the tunnel gave 1.08 and 1.38 dollars in gold per ton. Near the crest of Ling-Lung-Shan the ancient Chinese have done a large amount of work. In some places the rock has been quarried; in others they have followed a streak of ore from 2 to 3 feet wide for distances of some hundred feet; and in another they have made an immense underground working 100 feet long by 50 feet wide, and with an arched roof standing insecurely 30 feet above. Near by there are other places that were probably such cavern-workings, but as they have now fallen in they are inaccessible; 1500 feet west of the tunnel is the Hung-Chin series of ledges, bordering each side of a narrow ravine, through which a stream of water flows. On the east side a well-developed ledge of snowy white quartz 6 to 8 feet wide and nearly perpendicular has been opened. Several assays yielded 1.28, 3.35, and 4.43 dollars per ton. A continuation of this ledge on the west side of the ravine, known as Yu-Tzu-Tien, shows about the same characteristics and an assay-value of 4.43 dollars per ton. Between these two openings is a vein, differing from the first ledge slightly in its course, materially in its dip, and decidedly in the character of its ore. It consists of about 18 inches of massive white pyrites with a little quartz, assaying 8.10 dollars; and below this a talcose rock 2 to 3 feet thick, assaying 27.77 dollars.

FIG. 7.



Ma-Ting lies one mile east of the tunnel and consists of openings in two parallel ledges of quartz, respectively 2 and 8 feet thick and

separated by a horse of granite 5 feet thick. These ledges strike east and west, and have a perpendicular dip. One assay gave 15.17 dollars per ton.

The average of 19 gold-ore samples taken from prospect-holes of this district is 5.86 dollars; and the average of 10 samples, containing over 2.50 dollars, is 9.79 dollars, or nearly half an ounce to the ton. This is well within the working-limit, as the other surroundings are favorable. The location has sufficient water; an abundant and cheap fuel-supply is assured from hills thickly clad with pine, and a passable cart-road connects the mines with the sea-coast, 25 miles distant, affording easy transportation for machinery and, if necessary, for mineral-fuel. In the writer's opinion, these conditions make the Chao-Yuen region one of the most promising he has visited in China.

The Mongolia Mining-Region.

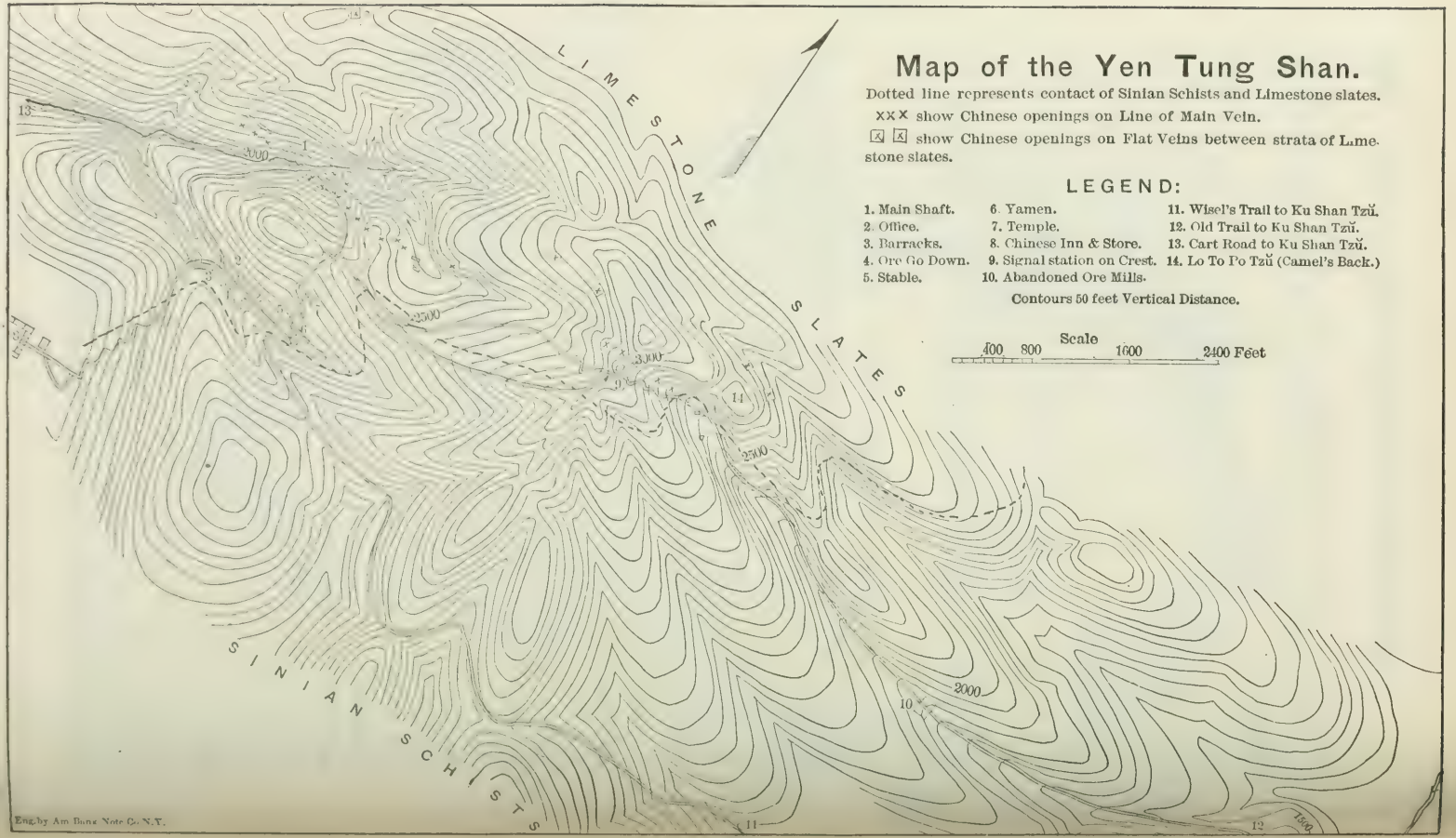
About 100 miles northeast of Peking and 80 miles from the coast is the Mongolian mining-region, indicated on Fig. 1 as a shaded square. It comprises several distinct districts, which are shown in more detail on the writer's route-map, Fig. 7.

The Jê-Hol Silver-Lead District.

Yen-Tung-Shan.—These mines, whose location is indicated in the northwestern part of the route-map, Fig. 7, together with those at Lo-To-Po-Tzu and Ku-Shan-Tzu, constitute the Jê-Hol mining-district, which was opened in December, 1887, with capital supplied by the Viceroy Li Hung Chang, under the direction of Prof. John A. Church, a member of this Institute. The mines had previously been worked by the Chinese, so that the entire outcrop of the vein at Yen-Tung-Shan is burrowed by their tortuous workings (Fig. 8), which in many places, however, have caved and are inaccessible.

The two formations of the district are limestone and a series of rocks called by Von Richthofen the Sinian schists. These schists consist of gneiss, mica-slates, and highly-metamorphosed shales, occasionally interbedded with granite and syenite. In some places the Yen-Tung-Shan limestones are highly metamorphosed, exhibiting a somewhat porphyritic appearance, but in other places they have a slaty cleavage. Yen-Tung-Shan appears to have been formed by the upheaval of these slates, and the boundary between them and the schists is shown on the map (Fig. 8) by a dotted line running

FIG. 8.


Map of the Yen Tung Shan.
Dotted line represents contact of Sinian Schists and Limestone slates.
XXX show Chinese openings on Line of Main Vein.
X X show Chinese openings on Flat Veins between strata of Limestone slates.
LEGEND:
1. Main Shaft.
2. Office.
3. Barracks.
4. Ore Go Down.
5. Stable.
6. Yamen.
7. Temple.
8. Chinese Inn & Store.
9. Signal station on Crest.
10. Abandoned Ore Mills.
11. Wisel's Trail to Ku Shan Tzŭ.
12. Old Trail to Ku Shan Tzŭ.
13. Cart Road to Ku Shan Tzŭ.
14. Lo To Po Tzŭ (Camel's Back.)
Contours 50 feet Vertical Distance.
Scale
400 800 1600 2400 Feet
LIMESTONE SLATES
SINIAN SCHISTS
3000
2500
2000
1500
Engby Am Bank Note Co. N.Y.

into the deep valleys and along the hog-backs, but keeping to a general trend a little north of east. The vein, as shown by the Chinese openings, has an east-and-west strike and a north dip of from 50° to 75°, and is entirely in the limestone slates, no ore having been found in the schists. Besides the main vein, there are a number of bed-veins lying to the north at a distance of from 500 to 1600 feet, and having a northerly dip of 10° to 15° from the horizontal. These flat veins, being interbedded in the strata of limestone, appear to be leaders from the main vein which have found their way between the stratification-planes of the limestone during or after their upheaval.

After reaching depths of 100 to 150 feet, at Yen-Tung-Shang, the Chinese experience great difficulty from water as the workings in the vein extend over 4800 feet, and in this great length intercept the water-shed of half the mountain, so that in the rainy season, during July and August, the workings in the west and lower end of the vein are invariably drowned out.

Under the supervision of Mr. H. F. Dawes a three-compartment shaft, 12 by 4 feet, was sunk to a depth of 215 feet near the west end of the vein and gangways were driven. At the 100-foot level a gangway was driven eastward for a distance of 337 feet under the old Chinese workings. At the 200-foot level gangways were driven 22 feet to the east and 37 feet to the west. The vein-matter is 3 feet wide on the 100-foot level, but diminishes to 18 inches in the west end of the 200-foot level, and pinches out to a mere streak in the east end of the same. In the 3 feet of vein-material there is a pay-streak which has no regular position, being found sometimes close to the hanging-wall, sometimes on the foot-wall, and sometimes in intermediate portions. This pay-streak varies from one-half to two inches in width; it may consist of zinc-blende, in which case it contains about 23 per cent. of zinc and from 12 to 18 ounces of silver per ton; or it may be galena, which will run from 300 to 500 ounces of silver per ton. There seems to be no regularity in these changes, and nothing in the nature of the rock that will indicate them. The ore mined by the Chinese carries 28 to 57 per cent. of lead and 54 to 300 ounces of silver per ton. Washings from waste-ore carry little or no lead, but 16 to 80 ounces of silver.

The smelting at Yen-Tung-Shan is still carried on in the Chinese way, an account of which may be found interesting. This metallurgical process consists of three operations—roasting, smelting with litharge to concentrate the silver in a bar of lead, and cupelling this

bar of lead for silver. The roasting is done in an open heap, the ore being kept in place by loosely piled adobes. A comparatively high heat is produced, which results in driving off a portion only of the sulphur and melting the remaining sulphuret of lead into a matte. The smelting is performed in two low crucible-furnaces built of clay, 2 feet 6 inches high, 12 inches inside diameter at the top and 18 inches at the bottom. The blast is supplied by a *Fêng hsiang*, or wind-box, worked by two men. The furnace-charge is made up of two piculs* of roasted ore, one picul of rich silver-slag from a former operation, and sixty catties† of litharge either purchased or obtained from a previous cupellation. The product of one operation is only fifty-eight catties of silver-lead, as a large portion of the lead and a considerable percentage of the silver go off in heavy clouds of smoke. The cupellation, though performed on small quantities and being extravagant in fuel, is an exceedingly neat operation—an oblong pile of slightly moistened wood-ash, 2 feet long, 18 inches wide and 9 inches high, is made, and the test cut out with a hoop and pressed down with the foot. Twenty catties of lead are placed in the test, and half-bricks laid at intervals along the edge with adobes covering them and the test, thus forming a muffle. A cone of charcoal is then built over the muffle, and this charcoal is used as the centering of a dome, built of mud and straw, which is plastered on the charcoal till it has a thickness of 2 inches. The charcoal is then ignited; the lead oxidizes into litharge and is absorbed by the test, while more lead is added through the front opening; fresh charcoal is also added through a hole in the top of the dome, and after eight hours the silver brightens and the cupellation is complete. The two operations of smelting and cupellation consumed 200 catties of ore, 100 catties of rich slag and 60 catties of litharge, and yielded 58 catties of lead containing 31 taels or 41 ounces of silver.

Ku-Shan-Tzu.—The Ku-Shan-Tzu silver-lead mines lie about 7 miles east of Yen-Tung-Shan (Fig. 7), and being the headquarters of the Jê-Hol Mining Company they have been very vigorously pushed. A remarkable amount of work had been accomplished by the Chinese prior to the arrival of the American staff. About 250 men were employed in mining, washing and smelting the ore. The mines are situated at a barometric elevation of

* One picul equals 133⅓ pounds avoirdupois.

† One catty equals 0.01 picul, or 1⅓ pounds avoirdupois.

2500 feet, on the flanks of one of the hills which bound the sides of an open, gently ascending valley and rise to a height of 2900 feet.

The formation in which the mines occur is a hard, grayish-white limestone, bounded on the south and east by the Sinian schists. The vein cuts through the limestone stratum, presenting a nearly parallel strike with it, east and west, but diverging considerably in its dip which varies from 50 to 70 degrees to the north. The width of the vein varies from 2 to 12 feet, though its normal width is about 3 to 4 feet. It is difficult to speak precisely of its average value, as it is rather pockety and varies considerably in character. The pay-streak, which is from 1 to 6 inches wide, is composed of galena, sometimes mixed with tetrahedrite. Two assays from different parts of the pay-streak gave:

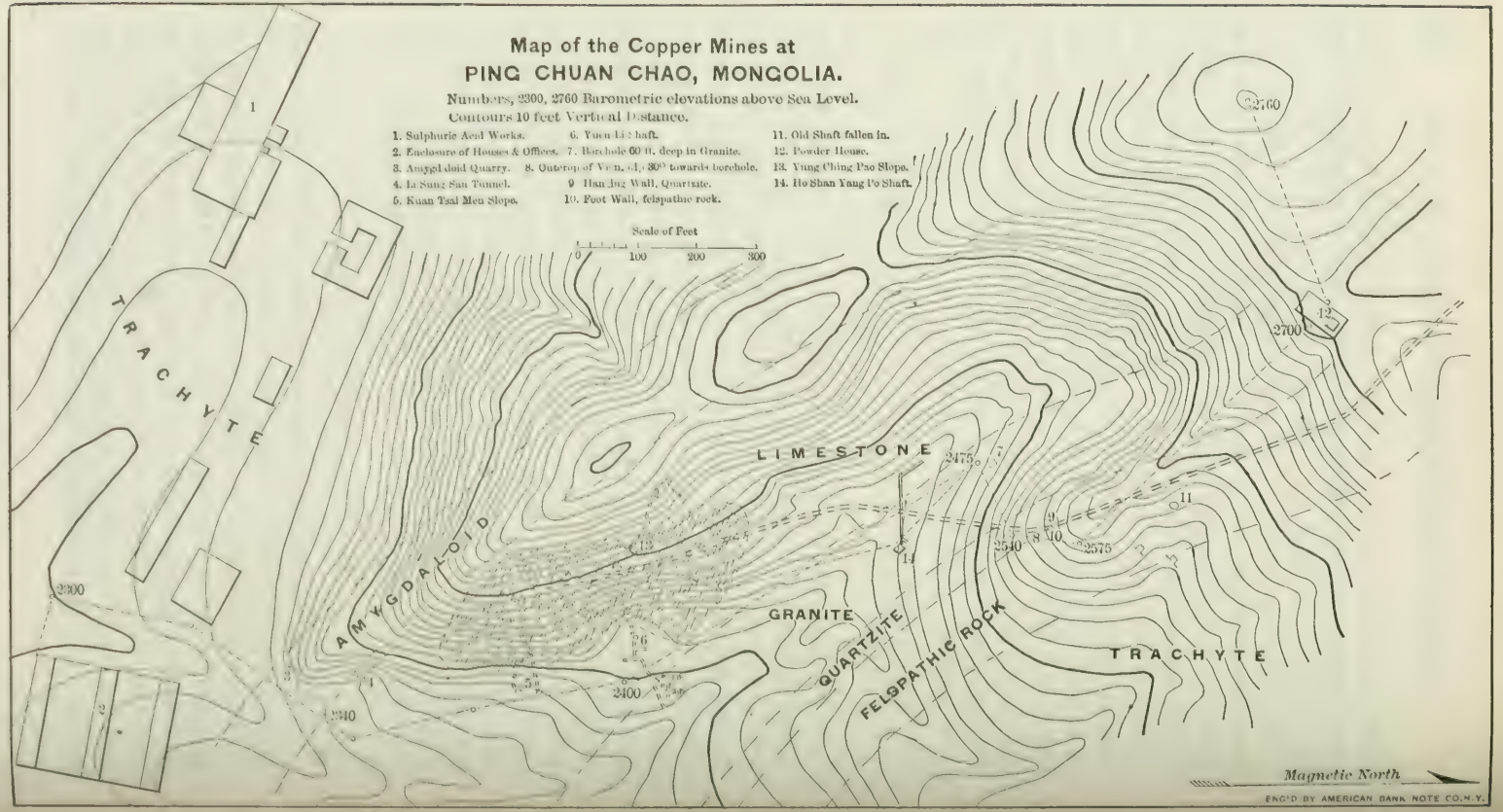
Lead, 4.9 per cent.	Silver, 90 ounces per ton,
" 15.2 "	" 102 " "

The Chinese workings prior to the advent of foreigners had been carried down 340 feet on the slope, reaching 40 feet below water-level, and stopes had been driven for a length of 240 feet. The foreign management, under the energetic direction of Prof. Church, changed the tortuous Chinese workings into a straight shaft, erected boilers and a hoisting-engine supplied by the Union Iron Works of San Francisco, and subsequently built a 50-ton coarse-concentrating mill, designed and constructed by Frazer & Chalmers, equipped with boilers and a Corliss engine, a Blake coarse crusher, a Sturtevant mill, a Buchanan fine crusher, rolls, one three-compartment jig, two two-compartment jigs and a rotary buddle. The smelting-works consisted at first of a 15-ton water-jacket furnace, and subsequently of a 40-ton furnace, with Baker blower and attached engine, dust-chambers and stack of boiler-plate, and a cupellation-plant.

The Ping-Chuan Copper-District.

The Ping-Chuan-Chao copper mines are situated 160 miles northeast of Tientsin (Fig. 1), at the point marked on the route-map (Fig. 7), as Chu-Tzu (signifying "office"). The controlling feature of the topography is a valley 2300 feet above sea-level (Fig. 9), descending and broadening southwards, and forking towards the north and west into a number of ravines which lead up to a divide, 2600 feet high, between the two branches of the Lan-Ho. The mines of Hsiao-Liang-Hsi and Ta-Liang-Hsi (Little and Great West-Mountain) are on the

FIG. 9.


Map of the Copper Mines at
PING CHUAN CHAO, MONGOLIA.
Numbers, 2300, 2760 Barometric elevations above Sea Level.
Contours 10 feet Vertical Distance.
1. Sulphuric Acid Works.
2. Enclosure of Houses & Offices.
3. Amygdaloid Quarry.
4. Li Sung San Tunnel.
5. Kuan Tsai Men Slope.
6. Yuen Li Shaft.
7. Borehole 60 ft. deep in Granite.
8. Outcrop of Vein, dip 80° towards borehole.
9. Hanging Wall, Quartzite.
10. Foot Wall, felspathic rock.
11. Old Shaft fallen in.
12. Powder House.
13. Yung Ching Pao Slope.
14. Ho Shan Yang Po Shaft.
Scale of Feet
0
100
200
300
TRACHYTE
AMYGDALOID
LIMESTONE
GRANITE
QUARTZITE
FELSPATHIC ROCK
TRACHYTE
2760
2700
2475
2540
2575
2400
2340
2300
Magnetic North
ENG'D BY AMERICAN BANK NOTE CO. N.Y.

western slope of this divide. Bounding the valley on the northeast and southwest are limestone hills, whose summits reach an elevation of 3000 feet. The geology at Ping-Chuan is rather complex. The rocks are of two general classes,—the stratified, to which belong the true limestones, crystalline metamorphic marbles and quartzites, and the igneous, which include amygdaloids, feldspathic rocks, trachyte, perlite and granite. The limestone overlies igneous rocks, and it is probable that the formation in the neighborhood of the mines has been tilted by their intrusion. The contact of the trachytes and the limestone may be compared to the outstretched fingers of one's hand, tongues of trachyte and other igneous rocks having apparently overflowed, and, in some instances, interpenetrated the strata of the limestone.

Travelling up the valley from Yuan-Tzu by the route shown in Fig. 7, the coal-bearing sandstones of the plain are followed for $2\frac{1}{2}$ miles until the limestone hills are reached, whose strata are seen to dip beneath the sandstones at an angle of 50° to the southeast. At a distance of 2 miles from the limestone-sandstone contact, the limestone forms a great air-anticlinal, subsequently dipping to the northwest, and continuing thus to Chu-Tzu, the office of the copper-mines. Near the office the trachyte shows itself as a wedge-shaped mass, 40 feet thick between strata of limestone, but it immediately spreads out like a fan, the limestone forming its eastern and southwestern boundaries. The igneous rocks appear to be divided into zones having a general northwest and southeast trend. The first of these zones begins with the contact of the trachyte and white marble at the Nan Shan (South Mountain) lead-mines, and extends northward for 2000 feet, crossing the main valley to the narrow ridge which separates the acid-works from the copper-mines, Fig. 7. The rock of this zone is a yellowish trachyte, consisting of feldspar with small, scattered flakes of mica. Traces only of copper are found in this zone. Succeeding it is a zone of very hard, yellowish-white, noncupriferous amygdaloid about 100 feet wide, which is exposed in the quarry close to the office. This is followed by a zone of grayish-white, indistinctly bedded, decomposed limestone, 300 feet thick, in which the greater part of the copper-openings have been made. It is seen most distinctly near the mouth of the Kuan-Tsai-Mên slope, and at the caved workings above the Yuen-Li shaft. This zone is succeeded by a disintegrated granite, which, at a depth of 40 to 60 feet, passes into an intensely hard, typical gray granite, composed of compact quartz, feldspar and mica, with occasional pockets of tough

quartz, and interspersed with chalcopyrite, the partial decomposition of which has frequently left green stains of copper in the seams of the rock. The width of this zone is 150 feet. It is followed by another, 50 feet thick, of hard, unweathered quartzite-rock, beyond which occurs the outcrop of a vein containing traces only of copper, but well marked by the character of its brecciated material and slickensides. This outcrop marks a change in the rock. To the east of the vein is a rather soft feldspathic rock, easily decomposed, and containing small cubes of pyrites, with sufficient copper to make green stains on the weathered surfaces. This feldspathic zone is 100 feet wide, and is succeeded by trachyte, in which several now abandoned shafts and slopes have been sunk. The trachyte is about 1000 feet wide, extending toward the northeast until it meets the limestone-formation, which there consists of a blue-gray limestone instead of white marble, as at Nan-Shan, the result probably of a less degree of metamorphism. This limestone-belt is a mile wide, and is followed again by trachyte, extending northward for an undetermined distance.

The copper-ore is found exclusively in the decomposed limestone-belt between the granite and the amygdaloid in pockets and small seams somewhat after the nature of a stockwork. In the Yuen-Li shaft it occurs as a contact-deposit between the granite and the limestone. The ore is a silicate of copper, containing some carbonate and occasional copper-pyrites. It is smelted in a 36-inch water-jacket furnace with iron- and lime-fluxes which are found in the neighborhood, and with a coke carrying 20 per cent. of ash.

The ore has an average composition of:

	Per cent.
H_2O,	6.0
CuO,	4.3
FeO,	8.3
Al_2O_3,	6.9
CaO,	2.0
Insoluble,	59.2
Undetermined,	13.3
	100.0

The iron-flux contains:

	Per cent.
FeO,	48.1
Al_2O_3,	10.0
CaO,	3.0
Insoluble,	15.0
Undetermined,	23.9
	100.0

The dolomitic limestone-flux contains:

	Per cent.
CaO,	29.0
MgO,	33.0
Insoluble,	0.5

With such conditions singulo-silicates were an impossibility, and it was only with the greatest difficulty and by the addition of large quantities of flux that the insoluble matter in the slag was kept down to 48 per cent. The large amount of silica retarded the running of the furnace, the rate not exceeding 25 tons per day and the product of pig-copper containing a large proportion of reduced iron.

The average slag contains:

	Per cent.
FeO,	11.0
Al_2O_3,	21.1
CaO,	12.4
CuO,	0.6
Insoluble,	48.8
Undetermined,	6.1
	100.0

The pig-copper contains:

	Per cent.
Lead,	0.48
Sulphur,	1.00
Iron,	30.05
Copper,	67.24
Insoluble,	0.24
	99.01

The pig-copper is refined by a native process, which consists in extracting the iron and other impurities by means of small quantities of lead in a hearth lined with Peking anthracite. A hole measuring 3 feet cube is dug and lined with brick; the cavity is filled with anthracite-culm in which a basin is hollowed out and coated with a paste made of anthracite-powder and water. A dome is then built over the basin with long crucibles serving as columns, and with lumps of anthracite, slate and clay. The hearth is rapidly dried and pig-copper melted in the basin by means of wood-fuel and blast from a wind-box. The fining is accomplished by constant stirring of the molten copper with an iron rod and the addition of small quantities of lead. The operation lasts from six to eight hours, whereupon the copper is ladled out and poured into moulds, the ingots being im-

mediately dumped into water. The copper is sufficiently pure for ordinary commercial purposes; it shows a fine silky fracture and is covered with deep rose-red oxide, so highly desired by purchasers. This native process is described only as a matter of interest, and not as an example to be followed, as the procedure is slow and the output small and costly.

The Jê-Shui Gold District.

The gold-workings of Jê-Shui, 35 miles north of Ping-Chuan-Chao, are carried on in a stockwork of quartz-seams occurring in gneiss-rock. The mines have never been in the charge of foreigners, but are operated in a very small way by Chinese contractors, who allow coolies to work for a royalty of 20 per cent. on the gross-proceeds. Their method consists in digging narrow, winding burrows from the surface to the small quartz-veins. All the work is done by gad and hammer without the use of powder, and the output is very small. The ore is carried in baskets to the hot-springs of the Jê-Shui where it is broken by hand to walnut-size and then crushed in an ordinary Chinese mill. The latter consists of a flat millstone with a wooden rod sunk in its center, around which turns a heavy stone roller set in a frame and drawn by a blindfolded donkey. The crushed ore is washed on inclined tables until the

No.	Name of mine.	Weight of sample.	Gold per ton by assay.		Gold obtained by Chinese washing.
1	Yang-Po.....	319 lbs.	1.17 oz.	$24 11	36.07 grains.
2	Hsi-Po.....	142 "	0.58 "	12 05	23.26 "
3	La-Ma.....	100 "	1.50 "	30 99	3.90 "
4	Chao.....	150 "	0.58 "	12 05	6.91 "
5	Kuo-Li.....	198 "	2.00 "	41 34	10.30 "
6	Licu.....	145 "	2.00 "	41 34	28.46 "
7	Wen.....	135 "	0.50 "	10 33	7.81 "
8	Shui	74 "	0.42 "	8 61	1.49 "
9	Mei.....	229 "	0.83 "	6 89	7.24 "
10	Sel'ct'd sample	.....	1.17 "	24 11	.....
11	Tailings from Chinese washings.....	.....	0.42 "	8 61	.....

Total weight of sample, 1492 pounds.

Average of first ten samples per ton, 1.025 ounces.

Average value of first ten samples per ton, $21.18.

Value of gold obtained by the Chinese in washing 1492 pounds of ore, $5.40, or per ton, $7.24.

gold and magnetic sand are concentrated in a black powder. As much as possible of the magnetic iron-sand is then carefully washed away, the operation taking place in bowls, and the remainder is extracted by hand with the aid of a lode-stone.

The writer sampled nine of the workings by firing a blast in the burrow, and taking all the rock that came down, without subjecting it to any picking process. The results are shown on page 594.

The Miao-Erh-Liang Coal-Field.

Forty miles southwest of Ping-Chuan lies the coal-field of Miao-Erh-Liang (Fig. 7), which supplies the coke used in the water-jacket furnaces at the silver- and the copper-mines. The bed is composed of of 1 foot of coal, 2 inches of slate, 2 feet of coal, 6 inches of slate, and again 2 feet of coal. It is bituminous and contains 12 per cent. of ash, which is increased to 20 per cent. in the coke. The mine is worked to a depth of 200 feet on a slope of 45 degrees, and is kept dry by bailing.

THE GEOLOGY OF THE HAILE MINE, SOUTH CAROLINA.

BY A. THIES, CONCORD, N. C., AND A. MEZGER, NEW YORK CITY.

(New York Meeting, September, 1890.)

THE Haile mine is situated in Lancaster county, South Carolina, twenty miles south of Lancaster. The topography of the country is characterized by low hills rising generally less than two hundred feet above the many small creeks which traverse the surface.

The road from Kershaw, the nearest railroad station, 3½ miles from the mines, leads through woods with sandy soil and cotton-fields. In many places it lies on the bare rock and shows that the adjacent covering of soil and earth is usually not more than one foot thick, and often less. This covering is to a great extent sand. Close by a bridge, about one mile from the mine, large round boulders of diabase may be observed.

The country-rock seems to be exclusively talcose slate, with a general strike varying from N. 45° E. to N. 70° E. These variations may be seen in the immediate neighborhood of the mines, the large open cuts of which present exposures favorable to such ob-

GEOLOGICAL SKETCH-MAP
of the
HAILE MINE S. C.
Scale, 1 in. = 400 ft.
Diabase underground
Diabase on surface
Old open mines
NEW SHAFT
OLD SHAFT
SPILSBURY SHAFT
MUC-FISSURE
MINE
DIABASE
CHASE HILL
Pond
R. R.
RED EARTH
CHLORINATION
ROASTING HOUSE
OFFICERS HOUSES
OFFICE
STAMP-MILL
SHAFT
HAILE MINE
GUMALO MINE
QUARTZ-VEIN
OLD MINE,
DIABASE

servations. The dip is N. W., generally more than 55°, and sometimes amounting to 85°. Beyond the mine-property there are no open-cuts or quarries in the vicinity; and the present paper being confined to studies of the structure within the somewhat narrow limits set by the mine-openings, the question how far the observed strike and dip of the talcose slate extend beyond those limits must be left unanswered.

The accompanying geological sketch-map, showing only actual exposures above or underground, may serve to render the following description clear. The most conspicuous geological feature of the tract is the presence of many dikes of diabase (heretofore erroneously taken for syenite) traversing the slates. The courses of several of these dikes have been determined and found to range from N. 5° E. to N. 35° W. They generally strike across the slate strata. One of them separates the Haile from the Bumalo mine, and, having been left standing between the two excavations, now presents almost the appearance of an artificial wall, with a vertical face on the southwest side and a slope of about 70° on the northeast, and striking N. 35° W. It is questionable whether this dike was formerly visible on the surface; and it is not known at the present time whether it continues to increase in thickness downwards at the rate indicated by the form of that portion of it which is now explored. As there are on both sides of this dike old shafts which are full of water, a knowledge of its size and condition must have been obtained, or at least obtainable, by the former miners. But there are no mine-maps which show whether the shafts themselves or galleries driven from them encountered the diabase underground. Work was discontinued here some years ago; the present product of ore is furnished exclusively by the Bequelin mine.

This mine (see map) has exposed underground two dikes, one of which is visible on the surface for a length of 20 feet, with a thickness of about 5 feet. Underground, however, in the 120- and 180-foot levels it has a thickness of 125 feet. Between this and the other dike just mentioned, both of which have a strike of N. 5° E., there is yet another, only 22 inches wide, visible on the surface as an open mud-fissure striking N. 5° W. The northeast dike of the Bequelin mine consists properly of two branches, separated by a strip of lode-matter, 12 feet thick, and the whole is connected with the other dike by a mass of diabase of unknown thickness, which forms the hanging-wall of the ore between them.

Near Chase Hill a railroad-cut has laid open another dike for a

length of 250 feet. The strike cannot be distinctly observed, but the red earth, commonly encountered as the covering of these dikes, extends for several hundred feet to the southwest. This appears to be the most important dike on the property. Other dikes are observed at *p* (with a probable strike of 45° E.), and at *t*, *n* and *o*, where the strike cannot be determined.

The dikes consist of a cross-grained mixture of augite with different feldspars. Iron pyrites is present in minute crystals, and in the fissures light green pellucid scales of serpentine are found. We think the pyroxenic character of the rock requires us to call it diabase. Occasionally, it should be added, the mass is very fine-grained, so that the magnifier does not distinguish its constituent minerals, and so dark as to resemble some varieties of basalt. In such cases the rock may also affect the magnetic needle.

The stratification of the talcose slates seems to be nowhere disturbed by the crossing of these dikes. The strata are simply cut off, without flexure or other change of position. The only exception to this rule is presented by the dike in the foot-wall of the Chase Hill mines, which may, perhaps, have caused the steep dip of the slates (80° to 85°) in these mines. At their contact with the dikes, and to moderate distances from the contact, the slates have, however, been so greatly affected in lithological character as to have become in many instances unrecognizable. They present resemblances to clay-slate, porcelain-jasper (though they can be cut with a knife) and even to grit and sandstone. Moreover, these altered zones contain an impregnation of gold-bearing pyrite, often concentrated to such a degree as to form rich layers 4 to 6 inches thick along the plane of contact with the dikes. This impregnation of the slates has involved the actual replacement of a part of their mass with pyrite and other minerals. The proportion of these minerals (which practically constitute the ore) is of course a point of the highest economic importance. The material mined and treated in the mill contains, as a general average, from 7 to 9 per cent. of pyrites.

Since the impregnated slate has been the material of economic value won by mining, the large open-cast mine-workings afford excellent opportunities for studying the cross-sections of the strata which they have exposed. Upon inspection of the old workings it is evident that the limit of the excavations has been determined by the extent of the pay-ore. At the Haile and Bumalo openings this limit forms a wall, standing open to view at right-angles to the strike, in places more than 100 feet in width. The space between the

two Bequelin dikes, *f* and *g*, is completely filled with ore, which is, however, poorer toward *g* (northeastward). The ore between *g* and the space northeast of *g* is all poor in gold, and the impregnation of pyrites is correspondingly feeble. Southwest from *g* the indications of profitable ground are favorable. In this quarter there were formerly open-cuts at the surface producing gold. A gallery is now being driven through this dike, and it is reasonable to suppose that ore will be encountered on the other side.*

Quartz appears in the deposits, especially at the Haile opening in highly irregular veinlets, and also at *k* as a separate vein 10 to 12 feet thick and snow-white, forming a conspicuous object in the forest. The strike of the vein is towards the foot-wall of the Haile mine, in the vicinity of which, however, nothing indicates the existence of such a strong vein. But in a line 30 or 40 feet southeast from that of the Haile deposit many acres of woodland show quartz everywhere, indicating the presence of many intercalated bed-veins of quartz. Near the hanging-wall of *k* no trace of an ore-deposit can be found, though this might be expected from the direction of its strike at the Haile mine—a sign that the impregnation does not extend much further than the old workings. It is obvious that the slates have been subjected to infiltrations and impregnations, the starting-point of which may be traced with convincing probability in every case to one of the *known* dikes, and there can be no doubt as to the genetic connection between the dikes and the gold-bearing character of the ore-deposits. The most natural explanation is the hypothesis that a leaching of the slates has taken place, perhaps for considerable distances, and that the fissures now filled by the dikes and also the fissures or permeable zones along their contact with the slates, have served as a system of channels, conducting the metalliferous solutions to the places where the conditions favored the chemical or physical reactions which resulted in the deposition of their valuable contents.

It may even be that a leaching of the dikes themselves has played a certain part in this process. It is true that their masses, as they appear at the surface (and, thus far, underground), do not seem great enough to be very important factors; but all of them, as far as they are known, become thicker downwards, and they may represent at greater depth enormous masses, which could have contributed considerable amounts of metallic salts to percolating waters.

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* This gallery has since been cut through the dike, and good ore found.

It is impossible to say at present whether the gold was an original constituent of the diabase, and now appears as the residuum, after all subsequent processes, of what was originally a much richer proportion, or whether, on the other hand, it is itself the product of infiltration from a foreign source. In other countries where a similar association between greenstone dikes and the gold-bearing character of neighboring zones has been observed there is reason to believe that gold was originally contained in the dike. Thus the Brazilian greenstone contains appreciable amounts of gold, while the *laterite* formed by its decomposition shows none, having presumably lost all of its gold by the leaching it has undergone. As the decomposition of the greenstone in that case appears to have involved the removal of certain constituents (notably alkalies) without their replacement by any other substances, we may infer that this Brazilian greenstone contained gold originally. With regard to the greenstone or diabase of the Haile mine-property, it is likewise impossible to say positively whether it originally contained gold, and now, after having enriched the adjacent zones, contains none. As yet, indeed, the latter proposition has not been established by adequately minute and careful investigation.

If the foregoing sketch of the geological conditions of this property be correct, it is evident that the productive ore-deposits are not in any sense veins. The dikes are fissure-veins filled with material economically barren. There are intercalated zones, segregations or bed-veins of quartz, but these are not mined for gold on the property. The deposits of pay-ore present, perhaps, in general a foot-wall of some regularity; but there is no defined hanging-wall, and for the miner the "vein" is limited on that side simply by the fading out of the impregnation to a point where he can no longer consider the material as "pay-ore." This is true at least of the Haile and Bumalo mines. The Bequelin constitutes an exception by reason of its hanging-wall of diabase. The vein-matter or matrix of the ore is only talcose slate, though this may have been altered beyond recognition. Everything goes to show that these phenomena are the result of infiltrations (very likely successive in time and differing in character) which have carried silica, pyrites with gold, and the compounds of magnesia, the effects of which can be traced everywhere.

The presence of gold in the vicinity of the dikes on the surface is another significant fact attested by old workings in many places.

It is still easy to pan out gold at many points in the neighborhood of the dikes.

Since the facts above set forth leave no doubt of the correlation between the dikes and the gold-bearing zones, or of the conclusion that the auriferous minerals have resulted from the infiltration and impregnation of the slates (the starting-point of which, in every observable case, can be traced back to a dike) it is evident that explorations for the discovery of new ore-bodies should be directed according to the indications obtained first by panning the red surface-earth, and then by making costeaning-cuts along the outcrops of the dikes. As this paper takes into consideration only the few known dikes in the vicinity of the mines, while it is probable that many others exist which have not yet been traced, the discovery and examination of such additional dikes may be expected to lead to results of economic importance, as well as scientific interest.

THE THIES PROCESS OF TREATING LOW-GRADE AURIFEROUS SULPHIDES AT THE HAILE GOLD MINE, LANCASTER COUNTY, SOUTH CAROLINA.

BY A. THIES, CONCORD, N. C., AND WM. B. PHILLIPS, UNIVERSITY OF ALABAMA, TUSCALOOSA, ALA.

(New York Meeting, September, 1890.)

1. *Introductory Remarks.*—The Haile Gold Mine is in Lancaster county, South Carolina, $3\frac{1}{2}$ miles east of Kershaw station on the Charleston, Cincinnati and Chicago Railway. It was first opened about the year 1832, and since that time has passed through various fortunes. Many different operations have been practiced here for extracting the gold: panning, grinding in Chilian mills or arrastras, stamping, the Designolle process, dry-crushing, and finally the Thies process of barrel-chlorination, which has now been in successful use for $2\frac{1}{2}$ years, and by which 36,000 tons of ore have been treated profitably.

The Thies process is in brief the treatment of dead-roasted auriferous concentrates (pyrite, sometimes also chalcopyrite, as at the Phœnix mine, Cabarrus Co., N. C.) with nascent chlorine, without artificial pressure or exhaust, in lead-lined iron cylinders; the throw-

ing of the mass on a sand filter; and the quick filtration and precipitation of the gold chloride with fresh and active ferrous sulphate. The gold is precipitated as metallic gold of a reddish-brown color, which, after being allowed to settle completely, is collected, washed, dried and melted with soda and borax in graphite pots and cast into bars.

The efficiency and economy of the process are such that, in working on a large scale, crude ore of the assay-value of $4 per ton, carrying about one-third of its gold free and two-thirds in sulphurets, can be profitably treated. At the Haile mine 36,000 tons of such ore have been successfully treated, and it is now mined and treated at the rate of 80 tons per day of 24 hours.

It should be observed at the outset that the chlorination system here used is not patented. Any one is free to employ it; and therefore what is said of it in the present paper is without any bias of commercial interest. It is the second attempt to introduce into the southern gold-fields, on a large scale, the chlorination of roasted ore in place of amalgamation, and as an adjunct to amalgamation before roasting.

The uniform and most gratifying success attending the use of this process for several years at the Phœnix mine, Cabarrus Co., N. C., (see *Trans.* xvii., 313) induced the owners of the Haile property to introduce it at these mines under the personal supervision of Mr. A. Thies. Mr. Thies's improvements on the old Mears chlorination process have been fully set forth in the paper above referred to. Without severing his connection with the Phœnix Co., he took charge of the Haile mine in January, 1888, where he operated 20 stamps from that time to January last, and since the latter date, has added 40 stamps, making a present total of 60 in operation.

By this process of chlorination there have been treated since 1880 at the Phœnix mine 5000 tons of concentrates, and at the Haile, since May, 1888, the date of the beginning of the work here, 2353 tons.

This statement is sufficient evidence that the process is conducted upon a commercial scale, and has long since passed the experimental stage. There is now no more reason to doubt its success as applied to low-grade auriferous sulphides than to doubt the success of the Thomas process in the manufacture of steel. Nor is it too much to claim for it that it stands metallurgically towards the other and older processes for the extraction of gold as the Thomas process stands towards the other and older processes for the manufacture of steel.

Both these processes were successfully applied about the same time. They are vital improvements upon the ideas of previous unsuccessful applications; and they deal with materials that hitherto had baffled all attempts at utilization, the one with high-phosphorus iron-ores, and the other with low-grade sulphuret gold-ores.

In order to realize the especial significance of the Thies process to the metallurgy of gold, it is only necessary to recall the history of southern gold mines for the past fifty years. It is a history of disappointed hopes. The capital invested in such mines has been, to say the least, extremely unproductive. Even during the last six or eight years, when any one who chose might have ascertained for himself the true method of treating these ores, the old state of affairs has continued, and the old story has repeated itself.

The elements combined in this successful solution of the problem are concentration, roasting and chlorination. They go hand in hand; each is incomplete without the others. But what distinguishes this process from all others is the method of applying the chlorine to the roasted ore, of which we shall speak later.

Since a complete account of the process requires some notice of the conditions under which it is employed at the Haile mine, this paper will briefly describe the geological occurrence of the ore; the operation of mining and transportation to the mills; and the milling and concentrating; after which an account will be given of the roasting and chlorination, including the delivery of the gold-solution to the precipitating-vats, and finally, the precipitation, collection and smelting of the gold.

2. *Geological Occurrence of the Ore.*—For a more detailed description of the geological conditions, the paper of Messrs. Thies and Mezger, presented simultaneously with this (p. 595, *ante*), may be consulted, and reference may be conveniently made to the geological sketch-map by Mr. Mezger, which accompanies that paper. It is not intended here either to controvert or fully to adopt the conclusions there set forth, the simple reason being that the two papers have been prepared independently, and there has been no subsequent consultation of all the writers concerned.

The ore at the Haile mines is a mixture of pyrite and stratified talcose slate. The strike of the strata varies from N. 52° E. to N. 73° E., and the dip (northwesterly) from 45° to 75°. The foot-wall is soft talcose slate, colored yellowish, brownish and red by iron-salts, and showing cross-joints. The hanging-wall is (often if not always) greenstone. From the foot-wall towards the hanging-wall

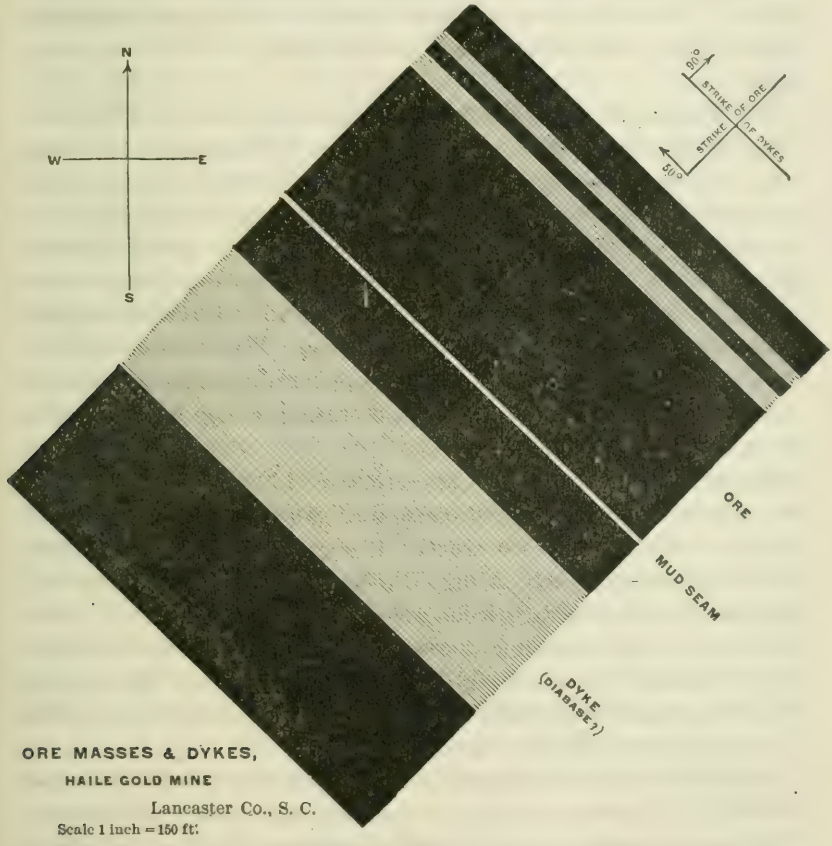
there is a well-marked increase in the hardness of the ore; bands of siliceous matter appear, and veinlets and veins of almost pure pyrite, from the thickness of ¼ inch to 1½ feet. These streaks of pyrite cut across the slates in all directions and are also at times parallel to them. Now and then masses of considerable hardness are also found near the foot-wall, but whether they occur here or in the vicinity of the hanging-wall, they are evidently derived from the talcose slates. It is believed that whatever be the earthy material mixed with the slates in the ore-bearing mass, it has been derived from the slates themselves. Even the very siliceous material found near the hanging-wall and bearing no close resemblance to the slates either in color, hardness or stratification, was doubtless derived from them.

Several dikes, seemingly of diabase, cross the slates at irregular intervals. On the accompanying diagram they are shown as cutting the slates at right angles to their strike. This is not quite true; but the variations are slight. Between these dikes lie the immense deposits of talcose slates, impregnated with gold-bearing pyrite and with more or less free gold. The free gold is generally fine. Wherever the slates are rich enough in free gold and in pyrite, they are mined. The richer streaks are of various widths, from 2 feet to 30 feet, and are of the same general character as the main body of the slates. In immediate proximity to the dikes, as well as to the hanging-wall, the slates are, as a rule (with some exceptions), richer than elsewhere.

As to the correlation between the dikes and the ore-masses, little will be said here. That the dikes do influence the richness of the ore favorably seems to be beyond question, but it is not so easy to say why. Nor can we speak positively as to the nature of the dikes themselves. They stand nearly vertical; they cut the strike of the slates nearly at right-angles; they exercise a favorable effect upon the richness of the ore; but whether they are strictly diabase or not, and whether they are of plutonic origin or not, will not be discussed here.

3. *The Mining and Transportation to the Mills.*—In such thick masses of ore, choice must be made between taking out all the ore (sustaining the walls with timbers) and taking such ore as can be extracted, using as little timber as possible, and leaving pillars. The first plan is not to be thought of here, on account of the varying value of the ore, the enormous amount of timber required and the expense connected with keeping it in sound condition. The

other plan was therefore adopted and has worked satisfactorily. The levels are driven from 60 to 70 feet apart and connected at different points by winzes which serve during stoping as ore-shoots. Between these winzes main pillars are left, while the ore is stoped on each side of the pillars from hanging to foot, leaving sufficient solid ore against the hanging-wall to insure safety.



Of the mines partly opened and worked in earlier years, the Bequelin (formerly known as the Blauvelt) is now in active operation. The main shaft has reached a depth of 194 feet on the dip of the ore, of which 15 feet are used for a sump. The levels are driven east and west from the shaft at intervals of 60 feet, and the ore is trammed from chutes at the 180-foot level to the main station

to be hoisted by double skips to the surface. From 80 to 100 tons are hoisted in 24 hours.

The system of mining would require for its full discussion a separate paper. It may be characterized here as the pillar-system (*Pfeilerbau*) generally employed when excessive timbering is to be avoided.

The drilling is done by double-hand, single-hand and machine-drills. For this latter purpose there are 4 Ingersoll-Sargeant Drills of $3\frac{1}{8}$-inch cylinder, now used entirely for development-work.

The plant at the Bequelin shaft consists of 2 boilers, of 90 and 35 horse-power respectively, which, under full headway, carry from 75 to 80 lbs. steam-pressure, consume about 6 cords of pine wood in 24 hours (at a cost of $1.40 per cord, delivered), and furnish steam to one 20 x 30 Ingersoll air-compressor, one 40 horse-power Dickson reversible link-motion hoisting-engine, one 20 horse-power crusher-engine for a 20 x 10 Blake crusher, and one No. 9 Davidson mining pump, of 200 gallons per minute capacity. The ore is hoisted in skips which discharge automatically upon two grizzlies and thence to the crusher. One man can load 60 tons per ten hours on the skips, which hold 1500 lbs. each.

The grizzlies have each a square surface of 32 feet, and the bars are set $1\frac{1}{2}$ inches apart for the "shaft-smalls," while the coarser ore goes directly to the 20 x 10 crusher. The grizzlies and crusher discharge into an ore-bin, holding 30 tons. From this bin the ore is hauled by a narrow-gauge locomotive $\frac{3}{4}$ mile to the mill; a train-load being 7 cars carrying 3 tons each. An engine-driver and two boys look after the loading of the ore from the bin on to the cars, the transportation to the mill and the discharging into the mill-bins. They also keep the track in repair.

4. *Milling and Concentrating.*—There are at present 40 stamps working; 20 more are up, but not quite ready to begin dropping. The average amount of ore crushed per stamp per 24 hours during the last 4 months is 2.01 tons. The stamps weigh 750 lbs. each, and drop $5\frac{1}{4}$ inches 84 times per minute. The screens are 36-mesh, of brass wire. Formerly 40-mesh slotted Russian sheet-screens were used, but these were discarded in favor of the wire-screens, the latter not only possessing greater durability, but giving a far more uniform pulp for concentration. The average life of the wire-screen has been 6 weeks, while slot-screens often had to be thrown out in 14 days. The average amount of water used per stamp is $3\frac{1}{2}$ gallons

per minute, and the average consumption of quicksilver per ton of ore is 0.35 ounce. The average wear and tear of shoes and dies per ton of ore stamped is 1.3 pounds.

The area of inside-plates per battery of 5 stamps is 1.75 square feet, and of outside-plates 32 square feet, with a 2-inch inclination to the foot. There are 3 boilers of 50 horse-power each and one Harris-Corliss engine of 150 horse-power. The 40 stamps are run with a steam-pressure of 50 pounds.

The mill is of the back-to-back type, 30 stamps on east and 30 stamps on west side. Each double battery has its own cam-shaft, which is driven by belt from a pulley on the shaft common to all the stamps on the same side. The east battery-shaft and the west battery-shaft are driven by steel bevel-gearing from the main shaft.

The railway-cars are bottom-dumpers, and discharge their ore on the iron-plated comb, dividing the east-side bin from the west-side bin. These bins hold 150 tons each, and discharge their ore into Hendy self-feeders. The angle of inclination of the bin-bottom towards the self-feeders is 60°. The battery-tailings are conveyed to the concentrator through open launders 8 x 10 inches, with cross-riffles, one inch high, every 8 to 10 inches. Total length of launders, 78 feet; inclination, $\frac{1}{2}$ inch per foot. The launders, discharge into a box from which runs a series of small launders, at right angles to the main launder, to each concentrator. There are 16 Embrey end-shake concentrators, 8 to each 20 stamps. The distribution-table for the battery-tailings is provided with an amalgamated copper plate for saving any free gold, amalgam or free quicksilver which may escape from the outside battery-plates. The belts are set with $2\frac{5}{8}$-inch inclination and travel 5 feet per minute. The number of strokes is 192 per minute. The yield in concentrates per ton of ore stamped averages 9 per cent., *i.e.*, for each $11\frac{1}{9}$ tons stamped, there is a yield of 1 ton of concentrates. The loss in sulphurets is about 10 per cent.

The fire-assay value of the ore delivered to the stamp is $4.50 per ton. The mint-returns of bullion give $3.90 per ton of ore treated, of which $1.45 is to be credited to the stamps, *i.e.*, to free gold, and $2.45 to sulphurets. Taking the value of the ore at $4.50, and the actual yield in bullion at $3.90, we have an indicated loss of 60 cents per ton, or $13\frac{1}{3}$ per cent. Taking the yield in free gold at $1.45 per ton from an ore worth $4.50 per ton (or approximately 32 per cent.), we have 68 per cent. to be sent into the concentrates. But the total yield in free gold and in gold from the sulphurets is $3.90

per ton, so that the ratio of the free gold saved to the total amount saved is approximately 38 per cent., and of the combined gold 62 per cent. An ore of this kind therefore carries about one-third of its gold free and about two-thirds combined.

We have used the term combined gold to express the condition of the gold that is not free. Whether the gold that is not free is chemically or mechanically diffused in the sulphurets, or both, is not known. Much has been written upon this subject; but as yet we are not warranted in adhering positively to either view. One thing appears certain, viz.: that, no matter what may be the state of the gold in such material, it must be thoroughly roasted before it can be profitably extracted. From the 80 tons of ore stamped per 24 hours, there are obtained 7½ tons of concentrates, which gives as the yield of each concentrator, a little less than one-half ton. The concentrates are piled in a shed and thence conveyed by rail to the furnaces, a distance of 1200 feet. The average assay-value of the raw concentrates has been, for the last 12 months, $30 per ton. This corresponds closely to the estimated value. Assays vary from $25 to $35 per ton of 2000 pounds. The percentage of sulphur in the raw concentrates varies from 40 to 45 per cent. In roasting, this is brought down to from 0.25 to 0.40 per cent. and the value of the material per ton is increased by one-third, *i.e.*, a raw concentrate of $30, becomes a roasted material of $40 per ton.

5. *Roasting and Chlorinating.*—The roasting-plant consists of one pan-revolving furnace (of the type described in the paper previously referred to, *Trans.* xvii., 313), two double-hearth reverberatories, each of 400 square feet roasting-surface, and one American Spence furnace.

The capacity of the pan-furnace for twenty-four hours is 2½ tons of roasted ore (equal to 3⅓ tons of raw concentrates) and that of each reverberatory furnace 2 tons of roasted ore (equal to 2⅔ tons of raw concentrates). The yield of roasted ore per day of twenty-four hours is 6½ tons from one pan and two reverberatories. The consumption of wood is one-half cord for each furnace per ton of roasted ore. Each furnace is attended by four men for twenty-four hours, who do all the work of charging, firing and rabbling, and also deliver the roasted ore on the cooling-floor. They are paid $1.00 per day of twelve hours. The wood used is pine and costs $1.40 per cord delivered. Mr. Spilsbury, in a paper read before the Institute in 1883 (*Trans.* xii , 103), puts the roasting-capacity of the reverberatory furnace at 6 to 10 tons per day of twenty-four hours.

This is not practicable when the product is to be so roasted as to be suitable for chlorination. No such feat has been accomplished in *that kind* of roasting by any furnace, modern or ancient, automatic or otherwise. It should be added that the roasting to which Mr. Spilsbury refers was followed by amalgamation.

In November, 1889, an American Spence furnace was built, but the expectations entertained concerning its operation were not realized, and the furnace, after many unsatisfactory attempts, has been idle since. The difficulties encountered were: 1st, the matting of the iron rakes whereby they were prevented from swinging on their axis; and 2d, the banking of the ore on both ends of the shelves.

The first difficulty, viz., the clogging of the rakes by the fine ore, was in part overcome by providing the rake with a sheet-iron hood. But although this gave some help, it did not altogether remove the trouble. The swinging of the rakes on their axis is a *sine qua non* of their efficiency, and just in the measure in which this is prevented, in the same measure is the roasting hindered. If the rakes cannot swing they cannot rabble the ore, and without this no thorough roasting is possible. This trouble is intimately connected with the second difficulty, viz., the banking of the ore at the ends of the shelves.

Under the present construction of the furnace the difference between the lengths of the rods which move the rakes is 6 inches, so that when the ore drops from a shelf to the one underneath, the rake on the lower shelf is 6 inches in advance of the rake on the upper. This distance is not sufficient to prevent the fine red-hot ores from falling upon the lower rake as they are discharged from the upper shelf. These ores move very easily—they may almost be said to flow; and they begin to fall through the opening when the rake is 12 to 14 inches from it. In this way the ore falls in front and on top of the lower rake, and is pushed by it towards the end of the shelf. In its back-stroke, the rake, being clogged by the fine ore falling on it, cannot take with it all of the ore that it pushed forward. Some is left, and the pile grows larger and larger, until finally the entire end of the shelf is full of ore. The alternate pushing and pulling of the clogged rakes causes a banking of the ore at both ends of the shelves. After several weeks of continual trouble and worry it was finally decided to discontinue the attempt to utilize this furnace for chlorination-roasting.

The principle on which the Spence furnace works is correct in

every detail, and it is undoubtedly the proper furnace for roasting coarse ores for smelting-purposes where a complete elimination of the sulphur is not wanted. A chlorination-roast, however, must be a uniform dead-roast, and no furnace which will not deliver material in that condition can be used.

From the furnaces the ore goes to the cooling-floor where it is allowed to cool; then it is dampened with water and elevated to the chlorination-floor.

The chlorination-plant consists of two chlorinators, eight filters, two storage-tanks for the filtered solutions, thirteen precipitating-tanks, two settling-tanks for precipitates, a storage-tank for sulphuric acid, and a tank for preparing the ferrous sulphate (copperas).

As regards the chlorination, no better description of what is now done at the Haile can be written than is contained in a letter from Mr. Thies to Mr. C. N. Aaron, published in the Eighth Annual Report of the State Mineralogist of California, 1888, pp. 844–846.

Mr. Thies says:

PHŒNIX MINES, OCTOBER 15, 1888.

C. N. AARON, ESQ., SAN FRANCISCO, CAL.

Dear Sir: In my former letter I promised to write an article on the "Hydrometallurgy" of gold, but seeing, through the pressure of business, that it will be impossible for me to do justice to the subject to get it ready for print by the 1st of November, I will here give you a brief description of the barrel-chlorination as in practice at the Phœnix mines, North Carolina, and the Haile gold-mine, South Carolina.

The success of barrel-chlorination I attribute to the generation of chlorine in the ore-pulp, to attrition and to quick filtering. The original Mears process was based upon the pressure of chlorine, either generated outside and forced by a pump into a lead-lined cylinder through a hollow trunnion, or generated by the use of chloride of lime and sulphuric acid in such quantities as would create a pressure of from 30 to 40 pounds per inch. To ascertain the pressure, a lead-lined gas-pipe in the from of a goose-neck (and called so), passed through the hollow trunnion, was securely fastened, provided with a stuffing-box and connected with a pressure-gauge. In using chloride of lime, the inner lead-lining of the cylinder was provided with a sulphuric-acid chamber capable of holding 100 pounds of acid. While, indeed, the result in working well-roasted ores was satisfactory, the annoyance through leakage; the continued wearing-out of goose-necks; the collapsing of the inner lead-lining by a too strong exhausting of the gas; by being too often deceived as to the pressure of chlorine, when pressure but no chlorine was present, causing rehandling of the ore, made the process an expensive and unsatisfactory one.

In this annoying way I worked for nearly two years, testing meanwhile the effect of a highly saturated chlorine-water under attrition without pressure, and when, after repeated tests, I found my results equally as good without pressure as with the highest pressure (and better when I divided the requisite amount of chlorine and sulphuric acid, so as to have nascent gas during the time of working), I remodelled the chlorinator by closing up the hollow trunnion, removing the goose-

neck and acid-chamber, and substituted a lead valve, connected with the inner lead-lining in such a way that the presence of free chlorine can be ascertained at any moment, and no charge subject to chlorination need be thrown on the filter without full knowledge of the work being complete.

The cast- or sheet-iron cylinder (chlorinator) is 42 inches in diameter by 60 inches long. The heads are cast and securely bolted to end-flanges and provided with tight and loose pulleys. The bung for the introduction of the roasted ore and chemicals, 6 inches in diameter, is provided with a lead-lined cover, which, before rotation, must be closed hermetically.

The interior of the cylinder is lined with sheet-lead of 10 to 12 pounds per square foot. The capacity of the chlorinator is from 1 to 1¼ tons of roasted ore. Before introducing the ore, the chlorinator is charged with from 100 to 125 gallons of water, or, I might say, with enough water to make an easy-flowing pulp. This done, the roasted ore is introduced, half the requisite quantity of sulphuric acid is then poured in, and lastly, half the required chloride of lime, when the bung-hole is closed and the chlorinator is set in motion at the rate of fifteen revolutions per minute.

For Phœnix ores I used 40 pounds of chloride of lime and 50 pounds of commercial sulphuric acid per ton of roasted ore; but I charge 20 pounds of chloride of lime and 25 pounds of acid first, rotate for three or four hours, open the bung and charge the other half, having found better results by dividing the chemicals. Rotate for two or three hours longer, and if, by the aid of the lead valve, free chlorine is found present, the cover is removed from the bung-hole and the chlorinated ore is thrown on a shallow filter, 6 by 8 feet, provided with a 5-inch filter-bed, over which the pulp spreads to a thickness of about 4 inches.

The filter, before the ore-pulp is thrown on it, is first flooded with clear water from below, and when the water stands over the filter the discharge-hole is corked, so that the water acts as a cushion against the ore-pulp, prevents the packing of the filter-bed and admits of a free filtering.

When the chlorinator has been emptied on the filter the cork is removed and the solution allowed to pass into a stock-tank below. As soon as the first solution has passed through, so that the ore-surface is exposed, from 3 to 4 inches of water is added over the whole surface; and when this is filtered through and the ore-surface exposed again, the whole space above the ore, about 11 inches in depth, is filled, which, by practice on Phœnix ore, has proved sufficient to remove all the chloride of gold; but should there be still a reaction with ferrous sulphate more water must be added.

The filters are lead-lined, 18 inches deep, and have a fall of 1 inch towards the outflow. The bottom is first covered with perforated glazed tiles or clay or mineraline, which is impervious to the action of acids and chlorine. On this rests the gravel filter-bed, which is topped off with ordinary clean river-sand. To prevent the filter from getting an uneven surface, longitudinal 1¼-inch wooden slats, 8 or 10 inches apart, keep it in place.

The filtering should be accomplished as quickly as possible, but, as this depends generally on the fineness of the ore treated, no rule can be established. As long as the solution shows the presence of chlorine when the last wash-water has passed through the filter, there is no danger of not having clean tailings. The solutions accumulating in stock-tank are let off into smaller tanks for precipitation with ferrous sulphate, which should always be regenerated, if not active, so as to destroy any ferric sulphate. Care should be taken by examining, after twenty-four hours, to ascertain if all the gold has been precipitated, as losses have occurred by a par-

tial precipitation. The tanks for precipitation should not be too deep; a convenient size is 6 to 8 feet in diameter and 3 feet high, holding the solutions from about 3 tons of roasted ore. A sufficient number should be on hand to allow the precipitate at least three days to settle. After three days settling in shallow vats, the supernatant liquor can be drawn off and fresh solutions added for precipitation.

At the Phœnix, the liquor is passed over metallic iron, and the copper is recovered as cement. From the precipitating-tanks the precipitate is finally collected, washed as clean as possible to remove the iron-salts, dried and melted.

The amount of chloride of lime and acid used at the Phœnix I have stated as 40 and 50 pounds respectively, which is due to the presence of an appreciable amount of chalcopyrite. An excess of acid should always be used so as to convert all the lime into a sulphate to remain in the filter. The solution should be slightly acid. If neutral, soluble chloride of lime will cause a bulky precipitate with ferrous sulphate. At the Haile mine, where I have to deal with a pure "iron sulphuret," I use at present, but 10 pounds of chloride of lime to 15 pounds of acid, and treat 4 tons of roasted ore in two chlorinators during ten hours, and I have not failed to extract 94 per cent. of the assay-value.

The Haile mine ores, which have been so long the bugbear for economic treatment, and on which so many experiments have failed, are offering no obstacles to the barrel-chlorination, and are cheaper and easier to chlorinate than the Phœnix ores.

As the success of chlorination, by whatever process, depends on a thorough roast, assuming that we have clean concentrates, it is of the utmost importance that the roaster should have some guide to go by, and to this end, I let him test every charge before drawing by a bright-filed iron-rod. A small portion of the roasted ore is boiled in water and stirred with the bright iron. The least trace of sulphates will stain the iron,—a sign for the workmen that the roasting is not completed.

At the Phœnix, I use a revolving pan furnace of 12 feet diameter, with a short reverberatory attached. From two working-doors the roaster can rabble the ore. When a charge is finished, the ore is discharged through the hollow axis on which the pan revolves into an outer circle below, and is then removed by scrapers attached to the bottom of the pan into a car, and delivered to the cooling-floor, from which it is elevated into the chlorination-house. Such a pan-furnace roasts 1 ton of raw ore in twelve hours, with a consumption of three-eighths of a cord of wood and 90 cents for labor. The power necessary to drive the pan is a small item, and will not exceed 25 cents per ton of raw ore.

At the Haile mine, a double reverberatory furnace furnished 2 tons of roasted ore every twenty-four hours, with an average consumption of 1 cord of wood at $1 25 per cord, and four laborers. The cost per ton of roasted ore amounts to $2.62½.

The cost of chlorination by the barrel-process depends chiefly on the number of tons chlorinated per day. Two men can easily chlorinate 4 tons in ten hours, elevate the ore and clean out the filters, of which I have four to each chlorinator; and having arranged on this basis the work at the Haile mine, the cost for chlorinating 4 tons daily is as follows:

40 pounds of chloride of lime at 3 cents,	$1.20
60 pounds of sulphuric acid at 2 cents,	1.20
2 laborers at 90 cents,	1.80
1 chlorinator-man,	2.00
Motive-power,	.50
Total,	$6.70

or $1.67½ per ton.

Add to this 12½ cents for sulphuric acid for making sulphate of iron, and 20 cents for repairs and wear, which is more than liberal, and we have the sum of $2 per ton for chlorination, or $4.62½ for roasting and chlorinating 1 ton of roasted ore, representing 1⅔ tons of raw iron-pyrites. Inside of seven hours from the time the ore is in the chlorinator, the solutions are ready for precipitation and the tailings are clean.

The wear on the inner lead-lining of the chlorinators is imperceptible,—a chlorinator in use at the Phœnix for over five years does not show any wear on the lead.

That the barrel-chlorination has advantages over the Plattner process cannot be gainsaid, and its successful working here and at the Bunker Hill, Amador county, California, where it surpassed the Plattner in results, will undoubtedly lead to its adoption in regions where the auriferous sulphurets are an important factor in the production of gold.

Hoping that these notes will aid you in your work, and ready to give any desired information,

I remain very truly yours,

A. THIES.

This description is as true of the Haile to-day as it was two years ago. The amount of chemicals varies between 10 and 15 pounds for chloride of lime and 15 and 20 pounds for sulphuric acid. The assay-value of the tailings from the chlorinator, which are removed from the filter-bed and thrown over the dump, is, on an average, $2 per ton.

6. *Precipitation, Collection and Smelting of the Gold.*—The ferrous sulphate employed as a precipitant should be used fresh, and to this end it is made up fresh for each day's run. It is easily prepared from one day to another.

As already observed, care must be taken to see that the precipitation is complete by testing the solutions about twenty-four hours after precipitation. Generally, the total absence of chlorine and a sweetish odor are fair indications of a complete precipitation after a thorough stirring with ferrous sulphate.

At the end of a month's work, or whenever it is deemed necessary, the completely-settled precipitate is taken up from the precipitating-vats into small lead-lined tanks 2 feet by 4 feet in size and 2 feet deep, in which it is allowed to settle again for twenty-four hours.

The supernatant liquor is then siphoned out, and the precipitate is washed with boiling-water until freed from most of the iron-salts. It is then collected on filters, dried and melted in Dixon crucibles with borax and soda, and shipped to the United States Assay Office at Charlotte, N. C. Its fineness varies between 975 and 985.

The successful treatment of 2353 tons of concentrates from the Haile mine by this process is all the more gratifying because of the failure of all other processes here to work at a profit.

It is commended to those who have to deal with large quantities of low-grade sulphurets in the belief that it will solve their difficulties, and enable them to work such ores profitably. For simplicity, cheapness and adaptability to such ores, it cannot be too highly praised.

INTERNATIONAL STANDARDS FOR THE ANALYSIS OF IRON AND STEEL.

NOTES ON THE WORK OF THE AMERICAN COMMITTEE.

BY JOHN W. LANGLEY, PITTSBURGH, PA.

(Pittsburgh International Session, October, 1890.)

In the summer of 1888 it was the fortune of the writer to present the subject of the desirability of establishing a set of samples of steel, which should be analyzed with extreme care, in order that they might become standards to which scientific and commercial analyses of iron and steel could be subsequently referred; also, that greater uniformity in the results of analyses might be brought about, since these standards would bear towards analytical methods somewhat the position which the original units of weight and length, the gramme and meter, or the pound and yard, preserved in Paris, London, and Washington, do to the mechanical arts.

The plan met with hearty co-operation, with the result that committees were appointed in Sweden, Germany, France, England, and America, to receive the material and to see that the necessary analyses were executed.

In England, the British Association for the Advancement of Science appointed a committee, which subsequently published a plan

of operations for that country, details of which will be found in the Reports of the British Association for 1888 and 1889.

The writer of this paper was entrusted with the duty of preparing the material and distributing it to the five countries named. This has been done, and analytical work on the steel is now going forward.

In the United States a committee of seven chemists was appointed to make the analyses, consisting of: Andrew A. Blair, Philadelphia, Pa.; Regis Chauvenet, Golden, Colo.; Thomas M. Drown, Boston, Mass.; Charles P. Dudley, Altoona, Pa.; John W. Langley, Pittsburgh, Pa.; Albert B. Prescott, Ann Arbor, Mich.; Porter W. Shimer, Easton, Pa.

This committee held a meeting at Washington, February 19th, 1890, and came to the decision that the whole subject of the existing methods for the determination of carbon in iron and steel seemed to be affected by so many discrepancies and differences of opinion that, before beginning work on the international standards, it would be desirable to make a preliminary study of methods. Accordingly, the writer was asked to prepare some additional material which could be used in somewhat large quantities for the above purpose only, and which should be known as the "experimental standards," to distinguish them from the "international" ones.

Experimental Standards.

An ingot of tool-steel, $3\frac{1}{2}$ inches square, poured from a single crucible in which the metal had been thoroughly "dead-melted," was heated and hammered barely enough to round off the corners. The ingot was then cut into two pieces, one of which was reheated and hammered down to a bar $1\frac{1}{2} \times \frac{3}{4}$ inches. The other half of the ingot was turned in a lathe with a blunt tool, exactly as in the manufacture of the international standards, and the turnings, after sifting from fine dust, were thoroughly mixed and then sealed up in glass jars. The hammered bar was drilled, and the drillings similarly treated. This material was given by the Crescent Steel Company of Pittsburgh.

The object of this was to have two bodies of metal presumably identical in original composition, but one of which had received a large amount of mechanical work, the other approximately none. These samples are respectively designated Experimental Standard Ingot, and Experimental Standard Hammered.

Scope of the Proposed Work.

The committee decided to limit the investigation to the following points at first:

a. The best method of conducting the igneous combustion of the carbon previously liberated from the metal by a solvent.

b. The best method of conducting the aqueous combustion of the carbonaceous residuum left by the solvent.

c. The influence of the solvent on the quantity and mode of release of the carbon.

For division *a*, it was decided to commence with a study of the method of burning the carbon in a stream of oxygen.

This work was assigned to Messrs. Blair, Dudley, and Shimer.

For *b*, the examination of oxidation methods by hot solutions of sulphuric and chromic acids. Work by Mr. Langley.

For *c*, the use of a double chloride of copper and ammonium, or of copper and potassium, either neutral or acid. Work by Messrs. Blair and Dudley.

A description of the apparatus for igneous combustion is as follows:

The apparatus used by Dudley consisted of an ordinary Bunsen furnace, with fifteen burners, in which was a porcelain combustion-tube five-eighths of an inch internal diameter, and eighteen inches long. Inside this tube, about three inches from the end toward the absorption-apparatus, was granulated and porous oxide of copper, occupying about four inches of the length of the tube, and filling the bore. This was held in place by asbestos plugs at either end. Next to the oxide of copper, toward the boat, was placed a roll of metallic silver about four inches long, nearly filling the bore of the tube. Rubber corks were used at each end. The combustions were all made in oxygen gas, using air to finish the aspiration and clean out the tube. The oxygen gas used was obtained in the market in cylinders, the gas being compressed to 250 pounds per square inch. Between the oxygen-holder and the combustion-tube was placed, first, a copper tube about four feet long, and one quarter-inch in diameter, fitted with three coils at the middle; next, a potash-bulb, and after that a chloride of calcium tube. All the oxygen gas, and the air used in the combustions, was passed through these parts. The copper coils were kept red-hot by a Bunsen burner during the passage of the gases, the idea being to burn any possible traces of combustible in the gas or air, and absorb the CO_2 formed before they should go into the combustion-tube. Between the absorption end of

the combustion-tube and the potash-bulbs were, first, a washing-tube containing about five cc. of sulphate of silver solution, with perhaps one-half gramme of silver sulphate; and next, a chloride of calcium tube. The potash-bulbs were of the Geissler form, of small size, made to order. These were fitted with a small chloride of calcium tube containing chloride of calcium simply, and commonly called the prolong. The whole absorption-apparatus weighed, when charged, about sixty grammes. The absorbing solution in the bulbs was the ordinary caustic potash solution recommended by Fresenius. The absorption apparatus was protected against the aspirator bottle by a full-size chloride of calcium tube. The joints were made with ordinary rubber tubing. The carbon from the steel was caught on an asbestos filter in a platinum boat, and dried after very complete washing at about 90° C. The boat, with its filter and carbon, was put into the tubes without transfer. During combustions, a slight pressure above the atmosphere was always maintained in the combustion-tube. Careful experiments show that the sulphate of silver washing-tube is a complete protection against hydrochloric acid, and the metallic silver in the tube was relied on for protection against free chlorine or other chlorine compounds. The time of making a combustion was about an hour and a quarter; fifteen to twenty minutes being taken to heat the oxide of copper, about half an hour for the burning, and twenty-five to thirty minutes for the air aspiration. Blair's apparatus is substantially the same as the above. His purifying train following the combustion-tube contains a tube of anhydrous sulphate of copper on pumice-stone. The tube is always freshly filled.

Shimer's apparatus is a porcelain tube containing oxide of copper. A stream of purified oxygen is used. An anhydrous copper sulphate absorbing-tube is employed, and then the customary $CaCl_2$, KHO and $CaCl_2$ absorbents.

Preliminary Results by Oxygen Combustion.

In the following analyses a solution of double chloride of copper and ammonium, except where otherwise noted, was used as the solvent of the steel. The carbon sponge was washed with dilute HCl and water until the filtrate was free from chlorine; then the asbestos, with adhering carbon, was put into the combustion-tube. These determinations were made before the elements of uncertainty attaching to the use of double chloride were discovered, and it will be seen in the latter part of this paper that quite different numbers can be obtained.

Under date of April 22, 1890, Dudley reports: Experimental standard "ingot" contains:

	Per cent.
Carbon,	1.058
"	1.065
"	1.052
"	1.058
Average,	1.058

Ditto, "hammered" standard:

	Per cent.
Carbon,	1.050
"	1.051
"	1.051
"	1.060
"	1.050
"	1.056
"	1.056
Average,	1.053

Under date of April 5, 1890, Blair reports that, using double chloride of copper and ammonium, with enough ammonia added to produce a slight cloud, and re-dissolving the basic salts formed in acidulated double chloride, he obtains:

	Ingot. Per cent.	Hammered. Per cent.
Carbon,	1.016	1.022
"	1.013	1.020
"	1.015	1.025
"	1.010	1.022

He then weighed out four portions of the "ingot" sample, and added to each 200 cc. double chloride of copper and ammonium, after adding to each the following amounts of strong HCl:

HCl.	Carbon. Per cent.
10 cc.,	1.048
20 "	1.048
30 "	1.048
40 "	1.051

He then took four portions of the "hammered" sample and treated them each with 200 cc. of double chloride and 20 cc. HCl. Results:

	Per cent.
Carbon,	1.046
"	1.046
"	1.044
"	1.049

Showing that the addition of HCl in excess of 5 per cent. does not sensibly vary the results.

A combustion of sugar at the same time showed:

	Per cent.
Carbon,	42.05
"	42.14
Theoretical carbon in sugar,	42.11

In answer to the suggestion that, possibly, oxides of nitrogen might be derived from the breaking up of ammonia retained by the carbon sponge, Blair reports, under date of April 17th, that he has made four combustions on the "ingot" sample, using a four times re-crystallized double chloride of copper and potassium.

	Per cent.
No. 1, carbon,	1.010
" 2, "	1.044
" 3, "	1.012
" 4, "	1.022

No. 1 was dissolved in the filtered solution of the above double chloride.

No. 2 was dissolved in 200 cc. double chloride, with 200 cc. strong HCl.

Nos. 3 and 4 were dissolved in the filtered double chloride, to which enough potassic hydrate had been added to produce a slight permanent precipitate.

Blair offers the hypothesis that a strictly neutral or slightly alkaline double chloride dissolves a small portion of the carbonaceous matter, which is consequently lost so far as the combustion is concerned. Therefore, the addition of acid, by preventing this action of the liquid on the carbon sponge, will give higher and more nearly accurate results.

That the use of an acid double chloride solution gives higher results was promptly verified by Dudley and Langley. This subject is more fully discussed in the latter portion of this paper.

Shimer reports, April 25th, that by using four grammes of steel in a neutral solution of double chloride of copper and ammonium, he obtains:

"Ingot," average of two combustions, 1.055 per cent.

"Hammered," average of two combustions, 1.052 per cent.

Preliminary Results by the Wet Method.

The chromic acid combustion was conducted in an apparatus originally designed by Prof. Hermann Wedding, although the method of

absorption used by him has been radically modified. Wedding's design consists of a flask capable of holding 220 cc., and provided with a glass cap of two tubulures. Through one of these passes a funnel-tube with stop-cock, and through the other the delivery-tube, surrounded by a water-jacket. All the joints are of ground-glass.

The merit of this apparatus is its compactness and its freedom from rubber or cork joints, at least in the parts liable to come in contact with chromic acid. The absorption-train, which is wholly different from Wedding's, consists of:

1st. A tube filled with pumice-stone and anhydrous sulphate of copper.

2d. A small washing-bottle containing the "pyro" solution referred to later on.

3d. A solution of sulphate of silver.

4th. A wash-bottle of strong sulphuric acid.

5th. A chloride of calcium tube, having a small plug of moist cotton in its anterior end.

6th. A Liebig tube, containing potassic hydrate solution.

7th. A U tube, having solid potassic hydrate in its anterior limb, and granulated calcium chloride in the posterior one.

8th. A chloride of calcium guard-tube.

9th. An aspirator filled with water.

The absorption portion for CO_2 is constituted by Nos. 6 and 7. The other parts, except Nos. 2 and 3, need no explanation, since they are well-known adjuncts of the chromic acid method described in such treatises on analysis as those of Fresenius and of Blair.

The method of conducting a combustion is to place the carbon, which has been liberated from 3 grammes of steel by the action of 150 cc. of solution of double chloride of copper and ammonium, to which 10 cc. of HCl has been added, in the flask, and run in through the funnel tube 100 cc. of a solution made by dissolving 10.5 grammes CrO_3 and 66 cc. H_2O in 210 cc. strong H_2SO_4. Purified air is drawn through at the end of the combustion.

The train at first used did not contain the "pyro" tube No. 2. Under these conditions the following numbers were obtained from the experimental "ingot" standard:

	Per cent.
Carbon,	1.099
"	1.145
"	1.040
"	1.058
"	1.109

A search was instituted to find the reason of these discordant results. It is well known that the carbon sponge from the action of the double chloride retains chlorine or hydrochloric acid in a form not removable by washing; suspicion at once fell upon the HCl known to be evolved from the carbon sponge.

A series of blank combustions was made, and also mixtures of air and HCl were drawn through the absorption-train. It was found that the anhydrous $CuSO_4$ would stop HCl only when it was perfectly fresh and in relatively large quantity, a quite small degree of hydration impairing its absorbing powers seriously. A tube containing an aqueous solution of sulphate of silver was then added to the $CuSO_4$ tube with the result that vapors of HCl are perfectly and completely arrested. When this point was established blank combustions were made and small quantities of chlorides introduced into the chromic acid mixture; it was invariably found that the potash-bulbs gained in weight. As one example out of many the following is quoted:

One-tenth of a gramme of NaCl in generating flask; $CuSO_4$ and Ag_2SO_4 absorbents were interposed. The silver solution showed not a trace of chloride and yet the potash-train gained .0055 gramme.

This gain appears to be owing to chloro-chromic anhydride, or to oxides of chlorine, not absorbable by copper or silver sulphates.

This observation is not new, for it had previously been noted by A. A. Blair that chloro-chromic anhydride was not arrested by copper sulphate.

After a long series of experiments an absorbent for these bodies has been found in the following solution:

Pyrogallic acid,	.2	gramme.
Potassic oxalate,	5.	"
Sodic chloride,	3.	"
Sulphuric acid,	.2	"
Water to make up to	20	cc.

The theory of the action of this mixture is as follows:

The pyrogallic and oxalic acids are deoxidants. The sulphuric acid will form potassic sulphate liberating a very little free oxalic acid to guard against the solution becoming alkaline. The sodium chloride is not essential, but it is advantageous by diminishing the solubility of CO_2 in the liquid.

The action of this absorbent is very energetic. It will allow HCl to pass through it without change, but the smallest traces of free chlorine, or of oxides of chlorine, or of chloro-chromic anhy-

dride, at once turn the solution brown, and are either arrested or changed into HCl. Indeed, as a qualitative re-agent this liquid will approximate to iodized starch paper.

In using this in the absorbing train, the copper sulphate tube may be dispensed with, the pyrogallic liquid being used in its place, and followed by a sulphate of silver tube, so that the train between the generating flask and the potash-bulbs will contain: 1. A cooling tube; 2. The "pyro;" 3. The sulphate of silver; 4. Sulphuric acid; 5. Chloride of calcium.

Blank "combustions" were made as before with NaCl using the pyro-train till it was proved that there was no systematic gain in weight of the potash-bulbs.

As examples of carbon oxidation in *presence of chlorides:* Taken .0712 gramme sugar with a crystal of NaCl; found .1098 gramme CO_2. Theory .110 gramme. Taken .030 gramme Ceylon graphite, of about 3 per cent. ash; found .0292 carbon, equal to 97.3 per cent. carbon.

The following analyses were made, using the new train:

	Per cent. C.
Experimental Standard, Hammered,	1.075
" " "	1.070
" " Ingot,	1.041
" " "	1.034

It seemed possible that the carbon sponge left on the asbestos, after filtering from the double chloride solution, might suffer a slight oxidation and loss of carbon while drying. A set of duplicate analyses were therefore made on a variety of steels. The terms wet and dry, below, refer to the state of the carbon sponge when introduced into the chromic acid flask. The drying was performed at a temperature not above 100°C.

	Per cent. Carbon. Wet.	Dry.
Steel A,	.981	.936
" B,	.836	.813
" C,	.904	.891
" D,	1.054	1.073
" E,	1.145	1.154
" F,	1.095	1.108
" G,	1.027	.968
Pig	3.695	3.722
Totals,	10.737	10.665
Average,	1.342	1.333

These determinations were made before the discovery of the variability of action of double chloride with age and manner of

crystallizing, to be pointed out hereafter. Moreover, they were distributed over a period of three months, so that some of the differences may be properly attributed to the chlorides used. Still, these would pretty well neutralize each other in sixteen determinations, so that the practical identity of the final averages would appear to show that the sponges do not lose appreciable amounts of carbon by drying.

On the other hand, the "Hammered" Standard, similarly treated, but having the sponge dried in an air-bath at a temperature not exceeding 120° C., gave,

	Wet. Per cent.	Dry. Per cent.
Carbon, ...	1.064	1.027
"	1.059	1.023
"	1.064	1.018
"	1.045	1.032
Average, ...	1.058	1.025

Apparently, therefore, there is a loss of carbon by drying the sponge at temperatures above 100° C. The same double chloride, acidulated, was used in these eight determinations.

Mode of Release of Carbon.

In pursuance of the special work assigned to Dudley, in a study of the phenomena attending the release of the carbon from its connection with iron in the steel, some quite surprising results have been obtained. It is, perhaps, not too much to say that sufficient work has already been done to throw doubt on the accuracy of all recent carbon determinations made with preliminary solution of the steel in double chloride of copper and ammonium.

Dudley began his study of this problem of the release of the carbon by quite an amount of preliminary study on the reliability of his apparatus and method of making combustions. A number of modifications were tried, but finally the apparatus and method previously described were settled upon, and this apparatus and method were checked up by the following determinations:

In an ordinary glass combustion-tube some 60 grammes of the well-known combustion-mixture of chromate of lead and bichromate of potash were placed, taking care that the material did not reach to either end of the tube, and holding it in place at each end by asbestos plugs previously ignited. The tube was then placed in the furnace and a combustion made in the regular way, the position of the material, not reaching to either end, allowing the whole of it to

be uniformly heated. This combustion was simply a blank to eliminate any possible error due to impurities in the combustion mixture. A small increase in weight in the absorption apparatus was found, and a second blank gave exactly the same results. This being done, three samples of three grammes each of the experimental standard "ingot" were treated with 200 cc. each of the same double chloride solution rendered acid by 10 cc. HCl. As this solution of double chloride will be referred to hereafter, we will say that it was made in Dudley's laboratory by obtaining from the market commercial chloride of copper and commercial chloride of ammonium, and dissolving these in water and mixing them in the proportions to form a double chloride containing one molecule each of the two salts. A small amount of free ammonia was added, enough to cause a perceptible separation of hydrated oxide of copper. The solution was then allowed to settle, and was always filtered through previously ignited asbestos before using. This will be called solution "A," and it will be observed that the solution had not been crystallized. After solution was complete, the three tests were filtered into platinum boats, and three combustions made. Two of these combustions were made in glass tubes, using exactly the same amount of combustion-mixture as in case of the blanks above described, and following the same manipulation, except that the boats with their carbon were buried in the combustion mixture in the tube. The third combustion was made in the apparatus, and by the method previously described as Dudley's apparatus and method. The results obtained from these three combustions were as follows:

	Per cent.
Combustion-mixture No. 1, after allowing for blank, gave carbon,	1.102
Combustion-mixture No. 2, after allowing for blank, gave carbon,	1.109
Combustion with Dudley's apparatus and method,	1.106
Average,	1.106

The results were regarded as showing that Dudley's apparatus and method gave fairly reliable results so far as the combustion was concerned.

But these results were confirmed by two further tests, as follows: First—A direct combustion of crystallized and pulverized sugar in the same apparatus, and immediately following the last steel analysis above, gave:

	Per cent.
Carbon,	42.02
Theoretical carbon in sugar,	42.11

Second, this was followed by a blank combustion, everything being done exactly as in a regular combustion, except that nothing was put in the combustion-tube. This blank showed a loss in the weight of the absorption apparatus of two-tenths of a milligramme.

The apparatus and method being regarded as satisfactory, attention was next given to the release of the carbon. Experiments were planned to demonstrate the effect of different states of dilution of the double chloride solution on the release of the carbon; also, to determine whether large or small amounts of double chloride solution changed results, and also to determine whether varying amounts of chloride of ammonium relative to the chloride of copper present produced any change. Before this part of the work was fairly entered on, however, a shipment of crystallized double chloride was obtained from the market. Since it had already been noticed in the work done on apparatus and method previously alluded to that discordant results were frequently obtained, it was at first suspected that the apparatus and method were at fault, but after the tests of the apparatus, above given, were made, suspicion fell on the double chloride, and accordingly a nearly saturated solution was made from the shipment above alluded to. This solution will be called solution "B." Four combustions of the experimental standard ingot were made, using 200 cc. of solution "B" rendered acid with 10 cc. of HCl. These gave results as follows:

	Per cent.
No. 1, Carbon, ...	1.070
No. 2, "	1.067
No. 3, "	1.060
No. 4, "	1.065
Average, ...	1.065

These results, it will be observed, are a trifle over .04 per cent. lower than were obtained with the same steel, with the same apparatus and method, but using the uncrystallized double chloride solution "A." The problem at once arose how to reconcile these discrepancies. Obviously, if the apparatus and method could be trusted, either the solution "A" contributed something to the absorption apparatus, which was weighed and counted as carbon, or solution "B" dissolved and carried off some of the carbon in the steel—or possibly both. Who could tell where the truth lay?

The first attempt to unravel the mystery consisted in taking 450 cc. of solution "A" with 25 cc. of HCl, and adding to it 1 gramme

of steel. After the reaction was complete, and the solution had been thoroughly mixed, it was filtered through asbestos. The result of this treatment was that this 450 cc. of solution "A" had now in it some sub-chloride of copper, some chloride of iron, and also carbon liberated from steel which had been floating around in it, affording an opportunity for any material that might be separated from the solutions by the reaction to attach itself and be filtered out. The reasoning was as follows: If double chloride solution "A" contains any carbonaceous material in solution which separates when the solution is treated with steel, thus causing high results, a part, at least, of this material will separate when the 450 cc. are treated with 1 gramme of steel, and be removed by the subsequent filtration, and consequently combustions made with the use of this treated double chloride solution "A," will show lower results. Two combustions were accordingly made of the experimental standard ingot, everything being exactly as in the previous combustions, except the use of the treated double chloride solution "A." These combustions gave:

	Per cent.
No. 1, Carbon,	1.076
No. 2, "	1.070
Average,	1.073

Here was a reduction of a little over .03 per cent. from the figures obtained, using solution "A" untreated with steel, and the figures seem to indicate quite clearly that uncrystallized double chloride solution "A" contained something in solution that separated during the reaction with the steel, and gave erroneous and too high results.

In confirmation of this view, the following experiments were made: 200 cc., the usual amount for a combustion, of solution "A" with 10 cc. of acid was treated with 5 grammes of the experimental standard ingot, instead of 3; the assumption being, if there is a certain amount of carbonaceous matter in solution "A" which comes out during the reaction with the steel, lower results will be obtained if this error is divided over a larger amount of steel. Two combustions were made on this plan, which gave results as follows:

	Per cent.
No. 1, Carbon,	1.049
No. 2, "	1.082
Average,	1.065

It is possible that the first of these duplicates is a little low, as the

carbon sponge from the larger amount of steel was a little difficult to manage in the boat; but as both the results are lower than those previously obtained from solution "A," it is evident they point in the direction that there is something in solution "A" that should not be there.

One more experiment was made on solution "A," as follows: 1000 cc. of this solution was treated with 50 cc. of HCl, and allowed to stand for some time. This solution was then filtered on the boat, and although nothing was visible to the eye, a combustion was made. The absorption apparatus showed an increase in weight of $3\frac{1}{2}$ milligrammes. This corresponds to a trifle over .006 per cent. of carbon in the steel which had been used in all the previous tests. This apparently indicates, that although solution "A" probably contains something which it should not, that something only separates very slightly when the solution is treated with acid.

Thus far the evidence seems to point quite strongly in the direction that uncrystallized double chloride solution contains some carbonaceous matter which separates during the reaction with the steel. But who knows that crystallized double chloride does not also contain carbonaceous matter? And still further, who knows that either crystallized or uncrystallized double chloride does or does not dissolve some of the carbon of the steel during the reaction, and thus produce a loss?

Upon the first of these questions the following experiments were made: 450 cc. of solution "B," which, it will be remembered, was made from crystallized double chloride obtained from the market, was treated with 1 gramme of steel and acid, as previously described for solution "A," and two combustions made. These two combustions gave:

	Per cent.
No. 1, Carbon,	1.060
No. 2, "	1.061
Average,	1.060

These results are, it will be observed, only about .005 per cent. lower than were obtained with the same solution untreated with steel, and seem to indicate that crystallization removes at least the largest part of the carbonaceous material, such as was characteristic of solution "A." But the mother-liquor, from which the crystals forming solution "B" were obtained, may not have contained any large amounts of carbonaceous matter.

In continuation of this same study four more combustions were made, two of them using solution "B," but no acid, and the other two likewise using solution "B" and no acid, but treating the solutions with steel, as previously described. These combustions showed as follows:

	Untreated. Per cent.	Treated. Per cent.
No. 1, Carbon, ...	1.036	1.030
No. 2, "	1.035	1.024
Average, ...	1.035	1.027

These results are about .008 per cent. lower for the treated double chloride than for the untreated, but the number of analyses and the number of samples of crystallized double chloride experimented upon seem hardly sufficient to set at rest the question as to whether crystallization completely removes the probable difficulty due to carbonaceous matter in the mother liquor.

Thinking that possibly carbonaceous matter might be introduced into the double chloride solution "A" by the commercial chloride of ammonium used in making it, four combustions were made, using some of the same shipment of chloride of copper alone which had been used in making solution "A." All four of the solutions had acid present. Two of them were previously treated with one gramme of steel, and filtered as already described; the other two were not so treated. The same steel was used as in all the previous cases; the combustions gave the following results:

	Untreated. Per cent.	Treated. Per cent.
No. 1, Carbon, ...	1.072	1.040
No. 2, "	1.056	1.050
Average, ...	1.064	1.045

These figures, as will be observed, seem to indicate that part of the difficulty at least is due to the chloride of ammonium, since the chloride of copper alone gives nearly .04 per cent. lower results than solution "A." They also seem to indicate that previous treatment with steel throws out something even from the chloride of copper alone. Too much importance should not, however, be attached to these results, as the reaction between the chloride of copper and the steel was very slow; and as acid was present, and it was impossible to keep the solution agitated continuously, it is probable a slight

loss of carbon may have occurred. The variation in the duplicates may possibly be accounted for in this way.

Up to this point the evidence obtained seems to indicate: 1st, that uncrystallized double chloride made from commercial materials probably contains carbonaceous matter in solution; 2d, that this may come partly from the chloride of copper, and partly from the chloride of ammonium; and 3d, that crystallization is probably advantageous in diminishing, if not completely removing, this difficulty. But some points are not quite certain, and accordingly the following experiments were undertaken to throw as much light as possible on the subject.

Commercial chloride of copper and commercial chloride of ammonium were obtained from the market. These were dissolved in hot water and mixed in the proportions to give a salt containing one molecule each of the two chlorides. On cooling the hot solution to a low temperature a large crop of crystals of the double salt was obtained. The mother liquor was drained off from these crystals and set aside. Also a portion of the crystals obtained were rinsed with pure water and set aside. The remainder of the crop of crystals was again dissolved in hot water, evaporated a little and cooled, giving a second crop of crystals, from which the second mother liquor was drained off; they were then rinsed with pure water. The first and second crop of crystals were then each dissolved in water, forming nearly saturated solutions at ordinary temperatures. This gave three solutions to work with, viz.: (1) The mother liquor from the first crop of crystals, (2) a solution of the first crop of crystals, and (3) a solution of the second crop of crystals. In other words, solutions of uncrystallized, once crystallized, and twice crystallized double chloride. The mother-liquor solution was treated with ammonia sufficient to give a slight precipitate, and then filtered through asbestos. It was then divided into two parts, and one part treated with HCl to the amount of 5 per cent. The part treated with ammonia alone we will call basic mother-liquor and the other acid mother-liquor. Three combustions each were made with these two solutions, using the same steel and everything else as previously described. The results obtained were as follows:

	Basic Mother-Liquor. Per cent.	Acid Mother-Liquor. Per cent.
No. 1, Carbon, ...	1.086	1.145
No. 2, "	1.081	1.130
No. 3, "	1.067	1.150
Averages, ...	1.078	1.142

Two results are quite noticeable in these figures: 1st, the acid mother-liquor gives higher results than any previous determination; and 2d, the differences between the basic and acid solutions is greater than in any previous case.

The second solution, viz., that from the first crop of crystals, was divided into two unequal parts. The smaller of these was treated with ammonia and filtered as previously described, forming what we will call basic first crystals. The other part was filtered through asbestos and divided into two parts, one of which was treated with acid, as above described, and will be called acid first crystals. The other part was left without further treatment, and will be called neutral first crystals.

Three combustions each were made with these three solutions, showing as follows:

	Basic First Crystals. Per cent.	Neutral First Crystals. Per cent.	Acid First Crystals. Per cent.
No. 1, Carbon, . .	1.026	1.029	1.070
No. 2, " . .	1.017	1.034	1.081
No. 3, " . .	1.037	1.029	1.082
Average, . .	1.027	1.031	1.077

The marked diminution in the amount of carbon between the mother liquor and the first crop of crystals is very noticeable. In the basic results the difference is .051 per cent., and in the acid results the difference is .065 per cent. It will also be observed that there is a slight difference between the results with the basic and the neutral crystals.

With the second crop of crystals only basic and acid combustions were made, the solutions being obtained as previously described. The results obtained were as follows:

	Basic Second Crystals. Per cent.	Acid Second Crystals. Per cent.
No. 1, Carbon, ...	1.016	1.049
No. 2, "	1.028	1.056
No. 3, "	1.023	1.052
Average, ...	1.023	1.052

These results, taken in connection with what precedes, are, to our minds, exceedingly interesting. The marked diminution in the amount of carbon obtained with the second crystals over the mother-liquor, especially in the acid solutions, cannot fail to be noticed, and this would seem to indicate that double chloride made from commercial materials without crystallization contains carbonaceous matter. This point would seem to be settled if it were not for the doubt

whether the double chloride does not dissolve some of the separated carbon. Upon this point no positive data have yet been obtained. Two points further seem worthy of note: 1st, the basic first and second crop of crystals give practically the same results, while the acids do not; and 2d, the farther away from the mother liquor the nearer the results from the basic and acid treatment come together. To our minds these facts, together with what precedes, seem to indidicate that possibly the commercial material contains dissolved cellulose, and that crystallization diminishes its amount. Thus with the material used in the above experiments, once crystallizing diminished the amount of cellulose, so that the basic liquid retained in solution what was left (since alkaline oxide of copper is a well-known solvent of cellulose), while the acid solution, even of the twice crystallized salt, still gives up a little cellulose to the carbon sponge.

In the special ground covered by Dudley in a study of the release of the carbon, the results of which are given above quite at length, the services of Dudley's principal assistant, Mr. F. N. Pease, have been found of the greatest value. He made all the combustions, and his suggestions have been most pertinent and useful.

A few attempts have been made to estimate the carbon in the experimental standards by methods not employing double chlorides, but with not much success.

Langley had proposed dissolving the steel in a hot solution of copper sulphate made as nearly neutral as possible, then filtering and burning the mixed copper and carbon sponge in oxygen.—*American Association for the Advancement of Science*, vol. 24, 1875, p. 107.

This method occasionally gives too low results, because of the impossibility of being sure that the last particles of steel are dissolved. Dudley made the following trials on the "ingot" standard:

$CuSO_4$ was ignited to remove possible organic matter, then dissolved and filtered. The solution was made as neutral as possible, and allowed to stand on the steel several days. Result:

	Per cent.
Carbon,	1.011
"	.972

Then to 300 cc. of the sulphate solution was added 15 cc. HCl. The attack of the steel was rapid, but some gas was evolved. Result:

	Per cent.
Carbon,	1.002
"	1.036
"	1.040

He then tried attacking the steel by solid silver chloride under water, and burning the silver and carbon sponge. Result on "hammered" standard:

	Per cent.
Carbon,	.796

Langley made two determinations by putting the steel directly into the chromic-sulphuric acid mixture, without previous treatment by chlorides. Result on "hammered" standard:

	Per cent.
Carbon,	.788
"	.790

In this case more gas was evolved than what corresponded to the CO_2. It was found that methane would pass through the boiling chromic mixture without sensible oxidation; therefore, some of the carbon of the steel may have escaped in union with hydrogen.

Summary.

The conclusions here presented are not those of the committee in their official capacity. They are the views provisionally held by the members whose work is given above.

First. The combustion of carbon in a porcelain tube in a stream of purified oxygen, when the precautions indicated are used, gives sensibly accurate results.

Second. If the carbon contains chlorine, it is desirable to use a coil of metallic silver in the combustion tube, and it is apparently also *essential* to use some solution of silver, preferably the sulphate, in the purifying train.

Third. The chromic acid method is capable of burning all the carbon used. If this carbon also contains chlorine, then it is *essential* to use some deoxidant in the purifying train, preferably pyrogallic acid with oxalate of potash; also a liquid silver absorbent must follow the deoxidizing tube. Under these conditions this method gives sensibly accurate results.

Fourth. The use of a small quantity of HCl in a solution of the double chloride of copper and ammonium, invariably gives higher results than when a neutral solution is employed.

Fifth. There is doubtful evidence that the substitution of KCl for NH_4Cl in the double chloride is of any advantage.

Sixth. The most important discovery made by the committee in

this work pertains to the variable action of the double chloride solutions. This apparently throws doubt on the reliability of all carbon determinations previously made by this reagent, since they show variations on the same steel lying between 1.016 and 1.150.

When the degree of acidity is kept constant, the apparent quantity of carbon found is affected by the mode of preparation of the double chloride, by its age, and by the number of times it has been crystallized.

Seventh. A carbon sponge derived from a double chloride solution does not appear to lose any carbon by drying at a temperature under 100° C., but loses if the heat is higher.

The problems now before the committee, as suggested by the above results, are: The determination of carbon in steel by some direct process not involving the use of double chloride. The direct combustion of finely divided metal in oxygen or chromic acid is a process of this kind, and fusion in a mixture of bisulphate and bichromate of potash is another.

The determination, if possible, whether a neutral or alkaline double chloride liquid may not dissolve a portion of the carbonaceous residuum, and thus lead to results which are too low.

The determination whether the addition of acid simply prevents this tendency, or whether the use of acid may not favor the separation of pre-existing organic matter in the liquid, and its retention by the carbon sponge, thus leading to results which are too high.

The investigation of the cause of the influence of repeated crystallizations of the double chloride on the apparent quantity of carbon.

Finally, it is to be presumed that many, if not all of the above points, have attracted the attention of other analysts, and the committee will be glad to learn of the results and experience of others along these lines.

Discussion.

G. J. Snelus, Workington, England: With regard to the analytical methods which have been mentioned by Prof. Langley, it appears that the American members of the Committee have confined their attention to a means of estimating carbon which in my own practice I have always felt to be doubtful. When I was systematically engaged in estimating carbon, I never ventured to use any but the direct method of burning the steel and burning the carbon

with it. I know that the estimation of carbon by the double-chloride method is very easy, and much quicker than the method by direct combustion of the steel. But in my own practice I have felt that there were many risks about the short and easy way, and now we are shown that these risks have been verified. I hope the American members of the Committee will not discard the old direct-combustion method on account of its being somewhat tedious. If the steel is sufficiently finely divided, and the heat is sufficiently intense, there is in my opinion no difficulty in burning all the carbon out of the steel by the direct method, and if that is done it is perfectly clear that the simple collection and weighing of the products presents no difficulty whatever. If, however, the carbon should be dissolved out by any such method as the double-chloride, various causes would produce a loss. The operation could not be properly performed without excluding the atmosphere, as its oxidizing action would have some effect. We know, also, from the common method of estimating carbon by the color-test how easily this carbon, which is deposited from steel, dissolves under the conditions which prevail in that test. I therefore cannot help thinking that the direct-combustion method will prove more accurate, and I know that we in England who are anxious inquirers on the subject, are certainly verifying every result of other methods by comparison with the direct-combustion process.

G. LUNGE, Zurich, Switzerland: Allow me to congratulate Prof. Langley upon the initiative which he has taken in the effort to attain uniform standards of analysis for iron and steel. I would call attention to a short paragraph in his valuable paper which illustrates a difficulty I have encountered for many years, and which has puzzled me sorely. In speaking of the methods of absorbing carbonic acid, and removing the impurities connected therewith, he says "it was found that the anhydrous $CuSO_4$ would stop HCl only when it was perfectly fresh and in relatively large quantity, a quite small degree of hydration impairing its absorbing powers seriously." This is the well-known method of Fresenius; he was the one to propose the use of the anhydrous acids for removing the acid vapors of hydrochloric acid. Many years ago I found that the results of the ordinary carbonic acid determinations—not at all connected with the analysis of iron—gave extremely unsatisfactory results; and therefore I abandoned the use of the anhydrous sulphate, and worked with other methods. I never have found time to investigate the cause of the difficulty, but in this paper I discover

an explanation of the mishaps which at one time were constantly troubling me in my own work.

J. E. Stead, Middlesbrough, England: All English chemists will feel indebted to Prof. Langley and the American Committee for the careful way in which they have entered upon this investigation. It is, I think, very truly stated, that in England ninety per cent. of the combustions for the determination of carbon in steel are made by one of the methods described by Prof. Langley. I myself have employed the method used by Mr. Blair for many years, and have been of the impression that with the adoption of all precautions we could feel perfectly certain in our results. But now this paper raises a fog of doubt, and I am anxious to test my results with all possible care. My solutions of the double copper and ammonium chlorides have been slightly acid, or not quite neutral; but I have not added the 5 per cent. of hydrochloric acid, and I shall be curious to learn whether that addition will give us a higher result. Many of the most prominent and most experienced chemists in England have agreed very closely in their carbon combustion-tests; if my results are wrong then I believe theirs must be wrong also; but yet we hope that we have not been very far out. The greatest errors here mentioned amount to from 5 to 10 per cent; that is, in 100 parts of carbon, this method of analysis will determine from 90 to 100 parts.

With reference to the remarks of Mr. Snelus, I think that all chemists who habitually make a large number of combustions will agree that to burn a sample measuring a quarter, or even an eighth of an inch square, or in diameter, will probably require a week or a fortnight. Unfortunately the samples do not reach us in the finely pulverized state which is necessary for the combustion. We would all agree with Mr. Snelus if the material could be obtained as a fine powder, for the carbon-determination is certainly more accurate, from the powder, by direct combustion than by dissolving in the double chlorides. In burning ferromanganese, ferrochrome, and substances of that kind, I invariably grind the sample to an excessively fine powder, and then burn it directly in oxygen-gas. But when the substance is a piece of very fine steel, or very coarse drillings, such an operation is impossible, and then we are obliged to determine the carbon by some such method as has been here described.

C. B. Dudley, Altoona, Pa.: It is hardly necessary, I think, to say much more than this,—that we have succeeded in getting our-

selves into a condition where we really do not know where we stand. If, six months ago, any one had asked me to make a carbon-determination of a piece of steel, and to testify in a court of justice that I considered the results reliable, I should have been perfectly willing to do so. But if to-day a metallurgist were to ask me how much carbon there is in a piece of steel I should reply that I could not tell. That is practically the state of affairs at the present moment. We have run across a possible source of error in one of the oldest and best established methods in chemistry. For nearly twenty years I have been using, both as student, and later, the method which we are now criticizing. It has been taught us by the most reliable schools as the very best method that was known. Where the real difficulty lies I cannot say at the present moment.

Since this paper was written continual experiments have been in progress in my laboratory to discover the source of the error. The double chloride of copper and ammonium made from commercial materials gave us 1.15 per cent., or, more accurately, 1.147 per cent. of carbon. The self-same materials were twice crystallized, and the second crystallization gave us 1.052 per cent. carbon. The problem is: Does the commercial substance without crystallization (I am speaking of the acid solution) contain something foreign, or does the crystallized material dissolve some of the carbon when it is released from its embrace with the iron? I do not know. We have lately pursued the following investigation: It is well known to chemists that after steel is added to the double chloride solution the result of the reaction produces a proto-chloride of iron and a sub-chloride of copper. Now, on the hypothesis that all commercial double chlorides contain a small amount of cellulose, obtained possibly from the filtering-paper that may get into the solution during the commercial manufacture and be dissolved—I say that on this supposition, if we take some commercial double chloride, or the salt made from commercial materials, and form in it the same sub-chloride of copper and the same iron-salt, just as we would do if we added steel, then if cellulose is present the presumption is that it would separate out, and we would be able to catch it and burn it. We have made three experiments on this hypothesis—the first by adding sesqui-chloride of iron to a copper-solution, and reducing the salts with sulphurous acid. To account for the difference we should have had from 30 to 35 milligrammes, but we actually found less than 3 milligrammes.

The second experiment was made with the same materials which have been used in the experiments described by the author, but in

this case we simply added metallic copper to the double chloride solution. This would result in the formation of sub-chloride of copper in the solution, and would give the same conditions as though steel had been added minus the iron. The result of this experiment was a little less than 3 milligrammes again.

In the third experiment we treated the double chloride solution with sulphuric acid, the result of which would be to form sulphate of copper and ammonia, and form free hydrochloric acid in the solution. This experiment likewise failed to yield satisfactory results.

The hypothesis on which these experiments were based, namely, that cellulose was present in solution, and that by producing the above reactions the cellulose would be precipitated, does not apparently seem to be borne out; if we may trust these experiments, cellulose is not the cause of the difficulty, and we still remain in the dark.

The chemists who are here have consulted together, and have laid out a large amount of work, so I leave you just where I began—in the fog, as Mr. Stead has said. We will make our best efforts to overcome the difficulty, and hope to appear before long with renewed confidence in the correctness of our results.

Edward Riley, London, England: As a member of the Committee on International Standards, I may testify to the very careful way in which all the analyses have been made, and also to the use of every possible precaution to avoid error. And yet, the work is said to be marred by a serious error which has been discovered by the American committee, and which it will be our duty to investigate.

For my own laboratory I do not buy chloride of copper; I roast copper-scales and dissolve them in hydrochloric acid, so that, by inference, there cannot be any cellulose in the chloride of copper which I use. I have had frequent communications with the largest steel-works in Germany, which have operated upon the same samples of steel as we in England have had, and hitherto our results have agreed.

As for the direct-combustion method commented upon by Mr. Snelus, there can be no question that, if practicable, it would be the best method; but, as a result of my experience, I am free to confess that it can be carried out only with considerable difficulty. I hope that Professor Langley will favor us with his views on this point. It seems evident that if the members of the American committee had had confidence in the accuracy of the direct-combustion method, they would have employed it.

J. W. Langley, Pittsburgh, Pa.: I have listened to the remarks which have been made upon this paper with much interest; and, speaking in behalf of the committee, I may say that we are very glad to hear the suggestions which have been offered. We are anxious only to arrive at the truth—at the best method, upon which we can all agree. With regard to the comments of Mr. Snelus and Mr. Riley upon the direct combustion of steel in oxygen, I regret to say that my own experience is so limited that I will not venture to rest any statement upon it. That it is a very valuable process for research in scientific matters, I think there can be no question. But, for a chemist engaged in the rapid work required in steel- and iron-works, or for commercial analyses, or for any other purpose except scientific research, in which an unlimited amount of time can be taken, it appears to me that the method by direct combustion is practically ruled out on account of the time and care necessarily involved in preparing the sample; and, as Mr. Stead has pointed out, the working chemist rarely receives his sample in such a form that he can reduce it to the extremely minute and almost microscopic state of division which is necessary for the complete combustion of the carbon in the steel when oxygen alone is used as the attacking agent. It seems as though it were indispensable that the iron should be removed by some solvent before we can hope to determine the carbon accurately. Therefore, the question remains where it was placed at the beginning, namely, the necessity of seeking the co-operation of chemists all over the world in studying the vagaries of this method of steel-analysis, determining where these vagaries arise and how we can eliminate them.

THE PROTECTION OF IRON AND STEEL SHIPS AGAINST FOUNDERING FROM INJURY TO THEIR SHELLS, INCLUDING THE USE OF ARMOR.

BY SIR NATHANIEL BARNABY, LEE, KENT, ENGLAND, LATE CHIEF CONSTRUCTOR OF HER MAJESTY'S NAVY.

(Pittsburgh International Session, October, 1890.)

We must not conceal from ourselves that, against the perils arising from perforation of the shells or hulls of ships, we are greatly worse off in these days of steel and iron than we were when our ships were built of oak, teak and pine. Apart from the greater

risks arising from increased speeds and increased momenta in collisions, the material of which the hull is composed submits so easily to perforation that the writer has numbered among his coadjutors in his profession eminent men who were strongly opposed to the abandonment of wooden bottoms both in commerce and in war.

We have it on the authority of Lloyds' surveyors that about one-fiftieth of the value of the vessels in the mercantile marine is required annually to make good losses and repairs entailed by collisions alone. When the bottom of a wooden ship is struck in a collision with another vessel, much of the force of the blow is expended in bending and breaking the fibres of the wood, and the blow is gradually absorbed with but little permanent displacement and but little admission of water. In an iron or steel ship the bottom may be pierced and permanently displaced over large areas without any feeling of shock in the ship giving the blow.

But despite the local weakness of the material forming the side, and the care it requires to prevent corrosion in salt water, its good qualities have secured its wide adoption. Long steamships cannot be built of wood. Steamships built of wood cannot be secured against premature decay; they cannot be built quickly without the certainty of rapid decay; the risk of fire is always serious in them, and, except in very small vessels, high speeds are impossible.

The ships in use and building for commercial purposes on the western side of the Atlantic are still constructed generally of wood. The large sailing navy of the United States may be said to consist almost exclusively of wooden ships. America possesses nearly one-sixth of all the wooden sailing-ships of 100 tons net and upwards in the world, and she will probably find it to her advantage for many years to continue to build such ships of wood.

Of iron and steel sailing-ships, Great Britain owns considerably more than three-fourths of all that exist.

When we come to the steam mercantile marine, we find that the United States has as many of her mercantile steamships, of 100 tons gross and upwards, built of iron and steel as of wood. There can be no doubt that both here and in Canada you must follow the example of Great Britain, and build steamships almost exclusively of iron and steel.

It is the one prominent defect of iron or steel ships that I propose to consider, and that only, viz., the readiness with which they submit to perforation in collisions.

Security against risks of fatal disaster on this account depends

very little, if at all, upon the thickness of the bottom-plating or the closeness of the ribs. The advantage of a steel plating one inch thick over another half an inch thick would be very doubtful, and closeness of frame-spacing is not much more valuable. When open-hearth steel was first employed for the outer bottoms of ships, with a reduction of some 25 per cent. in thickness as compared with iron, it was supposed that the structure would be found lacking in rigidity. In 1877, Chief Engineer King, of the U. S. Navy, commenting on the first of such ships, thought that for a ship 300 feet long, and of 3700 tons displacement, and having a double skin, $\frac{1}{2}$-inch steel bottom-plating was too thin. He considered it doubtful "whether the hulls would have the strength to endure, for any length of time, the immense strain of 7000 indicated horse-power." Engines of 9000 horse-power are now put into sea-going steel vessels of 3000 tons displacement.

Experience has amply justified the strength of the material under all strains. One of the vessels which Mr. King saw ran aground badly in the Mediterranean and suffered no structural injury. A vessel of the same class, the *Leander*, having similar scantlings, had the misfortune at the outset of her career to be rubbed very hard indeed against the "Hornet Rock." This experience taught us that in the event of collision or grounding there is advantage not only in thin plating, but also in light framing within and adjacent to the plating.

In grounding, the frames and bottom-plating of this ship had been forced in over a very large portion of her length, and there was a permanent indentation of one foot between the frames. The positions of the frames were indicated like the ribs of a starved horse, but there was not a crack in the plating, and the riveting of the plating and butts remained sound. There would, apparently, have been no leakage in the double bottom had it not been for these stiffening-frames. Where they crossed the plating there were rows of rivets to connect them with the bottom-plating. The frames had been forced in like the plating, but they had permanently buckled, since they had no other way of submitting to displacement. When the plating recovered itself and came toward its original position, it broke the countersink off the rivets and provided admission for water. It appeared that, few as the frames were, it would have been better, under these circumstances, if the stiffening had been effected exclusively by bulkheads and longitudinal frames.

Security in an iron or steel ship against fatal injury, arising from the perforation of the shell under water, depends upon two things

only—size and subdivision. Pumps may be dismissed at once as being practically useless in cases of collision in iron and steel ships. No pumps will deal with a continuous inflow of water of even less than half a square foot in cross-section coming through a hole well under water. Apart from the perfection of subdivision, size is in itself an element of safety. A ship of small or moderate size is sometimes completely severed at the point of striking by the blow of a larger ship.

I am supposed by some persons to object unduly to large ships. But I confine my objections as to size to ships of war in a powerful navy. I believe that a large naval power, having very widely spread interests, and subject to modern modes of attack, would suffer more in a war from fewness of ships than from comparative smallness of ships and crews. I have never thought that size is a disadvantage in merchant-ships, provided they can be worked financially. On the contrary, the advantages arising from size in passenger-ships seem to me to be so great that I do not see where we shall stop.

If it were necessary to increase the draught of water, as we increase the length and breadth, that would impose a fatal bar to any considerable advance upon present dimensions. But there is no such necessity. By the use of several engines, working side by side, and suitable propellers, the draught of water of 26 feet need never be exceeded.

I was consulted some years ago by a business man, well known on both sides the Atlantic, as to the possibility of building a steel ship which would not roll or pitch or heave in the sea, and from which, therefore, the bulk of passengers would be in a less desperate hurry to go ashore. He thought fifteen knots a sufficient speed. Such a ship appeared to me to be perfectly practicable with a draught of water of 26 feet. I thought the minimum length and breadth would be 1000 feet and 300 feet respectively. I estimated that with engines of 60,000 horse-power, an ocean speed of fifteen knots could be obtained.

Two sets of apparent difficulties had to be overcome, viz., those connected with the building of the ship afloat, and those relating to receiving and discharging cargo. The ship would be a steel island, incapable of entering any docks. The building difficulties soon disappeared. They had no real existence. To meet the other difficulties, I proposed to form shallow, still-water harbors or docks within the ship, entered by gates in the sides, and to carry, always afloat there, the loaded barges and tugs; turning the barges out and

taking in fresh ones already loaded at the ports of discharge and shipment.

Such a ship would require to be fortified and garrisoned like a town. She could be made absolutely secure against fatal injury arising from perforation. The subdivisions required for this purpose might be made to serve effectually against the spread of any local fire.

I do firmly believe that we shall gain the mastery over the seas, and shall live far more happily in a marine residence capable of steaming fifteen knots an hour than we can ever live in seaside towns. The question whether we shall effect our conquest by mere size, or by mechanical devices, in ships of more moderate proportions, depends upon the success of certain efforts which are now in progress in another direction.

In these efforts there have been three stages:

First, there were the observations by the British naval constructors at the Admiralty, that when a gun, or water, or other loose weight is moved about violently by the rolling of the ship, these movements of the loose weight tend to bring the ship to rest. The phenomenon is fully discussed in the *Encyclopædia Britannica* article "Shipbuilding." The physical conditions have been briefly stated by one of the Admiralty officers as follows:

Explaining in a model the effect of a moving weight representing 100 tons in bringing nearly to rest and almost immediately what represented a vessel of 10,000 tons, he says the same effect must always occur in a rolling ship if we have a loose weight of any kind, whether the weight be water or a gun. If this reduction in rolling did not take place, "we should have something to explain which would be quite inexplicable. For, suppose we have two ships alike in all respects, as regards size, shape, weight, time of oscillation, etc., and situated on precisely the same seas, but one having all her weights properly secured, and the other with a weight capable of traversing the deck every time the ship rolls. If the two vessels were to roll to exactly the same extent, we should have the sea not only rolling the ship with the loose weight to the same extent as the ship with all her weights fixed, but the sea would, in addition, be doing all the work involved in the traversing of the heavy weight across the deck, which is quite impossible under the assumed conditions of perfect similarity. The sea can only do the same work on both. In the one case, the work consists entirely in rolling the vessel; in the other, it consists partly in rolling the ship and partly

in dashing the weight about. The rolling in the latter must, therefore, inevitably be less than in the former case."

This principle is now applied by constructing water-chambers above the shot-proof decks in the larger ships in the British Navy.

But by this system a certain amount of rolling is required in order to start and work the rolling-quencher, and something further is needed in passenger-ships. It is desirable to have a more delicate appreciation of the first tendency to rolling, and an instant application of the quenching apparatus.

The next step was taken by Mr. Beauchamp Tower, an able English mechanical engineer. He contrived a mechanical watchman, dependent upon a long pendulum indication. This watchman has sufficient power to control a small platform carrying a gun or other instrument. In a heavily rolling ship the watchman keeps this platform and its load level and at rest.*

The third step has been taken by Mr. Thornycroft, of torpedo-boat fame. He also makes a mechanical watchman, differing from Mr. Tower's, and dependent upon two pendulums, a long and a short one. This watchman is capable of controlling a large movable weight. Mr. Thornycroft describes his plan as follows:

The controlling apparatus depends first on a pendulum, or its equivalent, having a short period compared with the natural period of oscillation of the vessel to be steadied, or of the waves on which the vessel floats. A second pendulum, or its equivalent, having a long period, compared as above, and a large moment of inertia, is adapted to join in control with the short pendulum, and more or less balance its tendency to reduce stability when the vessel is inclined.

The steadying apparatus comprises a heavy movable weight fixed to a shaft in such a manner that when free it places itself automatically in the fore and aft position. It is controlled by a hydraulic motor capable of turning it to one or the other side of the center-line of the ship as may be necessary to balance a turning moment exerted on the vessel by waves, or to prevent or lessen the rolling. The distributing valve of the motor is controlled by the pendulum apparatus above described. It is proposed to employ a movable weight of about 12 tons for a vessel of 250 to 300 tons displacement. Its range of movement would be such as would transfer the center of gravity of the weight through a transverse distance of 6 feet.

* See *Transactions of the Institution of Naval Architects*, 1889, p. 348.

If this third step is made successfully, and Mr. Thornycroft's plan is applied to passenger-ships, there will be no necessity for pressing forward large ships in advance of their due time. I think we may assume that for some years to come we shall have to deal only with ships which can be built upon the land and launched into the sea without extraordinary risks,—ships like the *Oregon* and the *City of Paris*, and subject to the same dangers as those ships from collisions and injury to bulkheads.

Those who are familiar with the history of the accidents to these two ships will have realized the fact that in the former case good internal subdivisions saved the passengers and that much better division saved both the ship and passengers in the latter.

When an iron or steel ship is sunk by collision, it is sometimes erroneously said that the ship had a number of bulkheads or divisions, and that since these failed to keep her afloat, it matters little whether bulkheads are fitted or not. It often happens that a ship is sunk when only a comparatively small hole is made in a single compartment. The badly arranged bulkheads do not even delay the sinking. The water spreads itself differently owing to the presence of the bulkheads, but that is all. Instead of levelling itself from end to end throughout the ship, and gradually rising all fore and aft, the water first fills one compartment; then finding the top of the partition separating it from the adjacent compartment has been brought below the level of the outside water, it proceeds to fill that compartment, and so on.

In the year 1866 this question was fully debated by an influential committee of naval architects, shipbuilders, shipowners, Lloyds' surveyors and sailors. That committee was the Council of the Institution of Naval Architects. That Council decided that no iron passenger-ship is well constructed unless her compartments are so proportioned that she will float safely though *any one* of them be filled with water, or placed in free communication with the sea. They recommended that all iron ships should be so divided that not only the one largest compartment, but *any two adjacent* compartments, might be given up to the sea without sinking the ship.

The second is an advisory clause, the wisdom of which is not disputed. The first is condemnatory of badly constructed ships.

No deliverance of any kind has been made during the twenty-four years which have elapsed questioning these important decisions. Yet they are absolutely ignored by everybody concerned, and all iron and steel sailing-ships built since then, and the great bulk of steam-

ships, have been built in disregard of the recommendations of that Council; and in so far as they are used for carrying passengers or emigrants, all these ships have been, and are, in the judgment of that Council, badly constructed.

It is quite true that there are one or more water-tight bulkheads in every such ship; but they exist for structural purposes only, and not to carry out the view of the Council or to prevent the foundering of the ship when run into.

Sailing-ships generally have but one bulkhead, and that well forward. A few owners have added another "collision" bulkhead aft; while others again, with a desire to provide extra safety from foundering, have introduced one or two others in addition to these two "collision" bulkheads. But these are merely so much dead weight; though adding, of course, to the strength of the ship, they would not appreciably alter the time of foundering if the ship were struck anywhere between the two end bulkheads. They do not go far enough to satisfy the necessary conditions.

There are, however, more than fifty British sailing-ships of 600 tons and upwards which have more than two, and in some cases as many as five or six bulkheads. Of these, fully 23 per cent. were steamers which have been converted into "sailers," and in which the then existing bulkheads have not been disturbed. The highest number of bulkheads believed to be built in any existing sailing-ship proper —that is, an "unconverted" ship—is four, and these are not placed so as materially to alter the time in which a vessel would founder if struck in any compartment.

It must be generally understood that records in registers and examination by surveyors referring to bulkheads in steamships give no security in respect of the fulfilment of this most important condition.

I am contented here to state the facts.

In 1876, ten years after the publication of the views of the Council of the Institution of Naval Architects, only two or three owners of steamships made it a rule to build their ships on these principles. In the years 1881–1883 there were lost, from various causes, within twenty-one months, one hundred and twenty British iron steamships. Every one of these had at least one compartment of such large size that the filling of it would have caused the ship to founder. Last year there were added to the British Register about 1,000,000 gross tons of steamships. Of these, not one-third could lay claim to have been built with any intentional and satisfactory regard to safety by means of water-tight bulkheads.

Most of the well-divided steamers included in the small minority above referred to come under the rules of Lloyds' Register Committee of 1882, which happily decided that steam vessels 280 feet long and above, purposing to pass its survey, must have a certain number of efficient bulkheads, increasing in number as the ships increase in length. Considering that this Register is a voluntary association, with no statutory powers, I think it has done as much as it could in the right direction.

In entering upon the great contemplated expansion of iron and steel merchant shipping upon this side the Atlantic, it may be hoped that you will treat the views of the Council of the Institution of Naval Architects with more respect than they have been treated in Great Britain. The question of material has been settled for you. The troubles we had with iron on our side are a matter of history.

In 1874 we were giving for iron ship-plates of about $7\frac{1}{2}$ cwts. each, delivered at Portsmouth Yard, over £20 ($97) per ton, with a scale of extras from $3\frac{1}{2}$ cwts. upwards per half-hundredweight. These plates were tested by resident inspectors at the iron-works. This had been found necessary, as inspection at the ship's side prior to acceptance had been found to cause vexatious delays and heavy rejections after such delays, so that there was often a paralysis of work with both ironmakers and shipbuilders.

This being the condition of things in 1874, a Naval Committee was appointed to consider the causes of the rapid deterioration of boilers. This committee visited Portsmouth, and reported that it was necessary to secure a better quality of iron. They said that "the rapidity of deterioration of boiler-plates and ship-plates is greatly dependent upon the purity of the iron, which can be obtained only by increased expense of manufacture." They recommended the payment of higher prices than £20 ($97) per ton for $7\frac{1}{2}$ cwt. plates.

It was then determined to meet the recommended increase in cost of bottom-plates by using for inferior purposes untested plates, *i.e.*, plates bought on the faith of the "brand" of the maker. Such "branded" plates were at that time employed for all purposes in ships built under the survey of Lloyds' Committee. Here we had a curious experience, which satisfied us that if we could once get material of definite manufacture, and precise and easily recognized qualities, wide competition would speedily equalize prices.

We took ten prominent makers of iron plates, and secured delivery of "branded" plates to an identical specification. The plates weighed

about 7½ cwts. each. They were delivered at Chatham, and were then tested and examined by me, with the following results:

Prices of 7½ Hundredweight Untested Iron Ship-Plates in 1874.

Cost per Ton.		Quality of Material.	Condition of Surfaces of Plates.
Dollars.	£ s.		
46.80	9 13	Strong, but harsh	Two-thirds unfit for use for ship-building.
44.62	9 4	Strong, but harsh	Two-thirds unfit for use for ship-building.
85.36	17 12	Strong, and fairly ductile	Good.
87.78	18 2	Strong, and fairly ductile	One-fourth unfit for use for ship-building.
46.80	9 13	Irregular in strength; fairly ductile	All unfit for use for ship-building.
109.61	22 12	Good	Good.
58.68	12 2	Strong and fairly ductile	One-sixth unfit for use for ship-building.
92.87	19 3	Weak, and not ductile	Good.
47.28	9 15	Fairly strong; ductile	One-sixth defective.
61.35	12 13	Strong; ductile	Good.

In all these cases the extras for breadths over 4 feet, and lengths over 12 feet, raised these prices still higher, or involved the cost of fitting and riveting small plates.

Now, in 1890, you have a perfectly regular material, stronger and more ductile than any of this iron, with no trouble from surface-blistering and lamination. You can get Siemens plates of 20 cwts., almost without extras for any increased size, delivered at the ship's side, at less than £8 ($38.80) per ton. It has, unfortunately, been as low as £6, 5*s*. ($30.31).

The change from iron to steel was not made without a vast amount of anxiety on the part of responsible engineers who led the way. The result may not be altogether pleasant to the iron and steel makers. To the shipbuilder and engineer it calls for ungrudging acknowledgment of indebtedness to two eminent British subjects, Bessemer and Siemens, and for the grateful recognition of the enterprise and skill of French, German, and British makers.

We all owe to Germany a lasting debt of gratitude for the birth and training of that true prince—William Siemens.

The Use of Armor.

In modern ships of war the main use of armor in the formation of shot-proof decks adjacent to the load-water-line, and of more or less complete belts of side armor combined with such decks, is to protect the machinery and magazines from shot and shell.

There has been a wide abandonment of side armor, founded upon the recognition of several facts:

1. The increasing penetrating power of the gun. The 12-inch gun is now more than a match, at close quarters, for the best 18-inch steel plates. It requires the best 9-inch armor to keep out steel projectiles from the 6-inch gun at short range.

2. The great volume and accuracy of shell-fire makes the original arrangement of broadside ports in an armored side inadmissible. The crowding of men in the rear of an open or unarmored port can no longer be accepted.

3. The propelling machinery and the magazines are of more consequence than the whole battery of guns, because the ship has become, by virtue of her ram and torpedoes, a powerful fighting machine apart from her guns.

4. The large use of side armor, by reason of its weight and cost, limits the number of ships obtainable with a given sum of money.

But while side armor has been largely suppressed and superseded for these reasons, it still possesses one great advantage. It prevents the entrance of water into the zone between wind and water through holes made by light projectiles not capable of perforating armor, but capable of damaging seriously an unarmored side. Those who have appreciated most keenly the value of side armor at the water-line, for preserving the floating-power of the ship against light guns, have sometimes failed to see that my apparent unreadiness to concur with them has arisen from real sympathy.

My contention has been that all armed fighting-ships present equal claims to be kept afloat against the attack of the guns which they have to face. I consider that in all such ships the first duty of the designer is to enable the crew to inflict damage on the enemy. Arms and mobility, therefore, come first; endurance under gun-fire next. There must be degrees, varying with the size of the ship, in the strength of the arms and in the amount of the mobility. But all the fighting-men, except those in forlorn hopes, are equally entitled to a chance of existence under the artillery fire which they must receive and endure. I have been unable to understand the exclusive claim for protection by means of side armor in a particular class of ships in a large navy, such ships not being distinguished by exceptionally large crews, nor as being centers of operation for a fleet, nor as being required to attack fortresses. And the question as to the use of belt-armor in classes of ships not distinguished as above is perplexed by the following considerations:

First, and generally, whether, in strengthening the water-zone by side armor against feeble projectiles, we do not expose it more

to the very serious attack of larger ones, than it would be if formed with an armored deck near the water.

Secondly, whether, with a ship of given size and cost, we are not sacrificing active offensive qualities for the sake of a passive defensive quality. Speed of ship, power of armament, or stored-up capital (expended in the ship) may be made to pay too dearly for a passive defence against the invasion of water at the water-zone, seeing that the invasion of water beneath that zone becomes a more imminent and more serious peril to the State in proportion to the reduced speed and the increased cost.

So we arrive at an alternative arrangement, a solid raft-body at the water-zone in place of side belt-armor. It appears that such a raft-body can be made which, when undamaged, weighs, with its casings, not more than one-third the weight of water, and which, when saturated, does not exceed the weight of an equal bulk of dry fir. Such a system of water-line defence may prove suitable for all fighting-ships, large and small, by reason of its lightness and small cost. As I understand the position, it is proposed to give a trial to this system in the navy of the United States.

I think the time is coming, if it has not already come, when there will be a demand, not only for the recognition of the equal claims of all fighting-ships to be kept afloat against the attack of the guns which they will have to face, but also for the equal protection of their crews between decks against the overwhelming effects of the bursting of high explosives there.

For belt- and battery-armor it is still a matter for debate whether the process of manufacture first introduced, and since most successfully developed, by M. Schneider, or that subsequently devised and manufactured in Sheffield, is the better.

After this paper has left my hands, and before it will be read, there will probably be an interesting series of trials of armor on both plans at Annapolis; what I say may therefore be very brief.

Both systems have been in open competition from the beginning, and gun-trials have been made by all the maritime powers. The experience of the Sheffield compound armor-plate makers as to their own manufacture and that of their rivals has been very great. The amount of steel-faced armor which has been manufactured by them, or by their licensees, during the last twelve years, together with that which is in process of manufacture, equals 112,000 tons.

They urge several considerations in favor of steel-faced armor. They say that steel armor cracks through under the effects of blows

which are not sufficiently powerful to perforate the plate. They consider that the manufacture of steel armor is less uniform and certain, because it is of very great importance to oil-harden and anneal such armor; and these processes are not required for steel-faced armor. They point out that this treatment can only be applied commercially to large and awkward plates with great risk, expense and difficulty, and that under these circumstances a good test-plate offers no security as to the quality of the plates it represents. It is conceded that hard-faced armor endures better under oblique blows. They say that a cubic foot of steel armor weighs 492 pounds, whereas a cubic foot of compound armor weighs only 480 pounds, a difference of $2\frac{1}{2}$ per cent. in favor of the latter.

An experiment has recently been made in England to ascertain whether it can be confidently stated that oil-hardening and annealing, or some equivalent finishing process, is necessary for steel plates. Messrs. Brown and Messrs. Cammell, the two great Sheffield firms, makers of compound armor, are satisfied as to this necessity.

A 9-inch plate of steel was manufactured and cut into two plates, each 4 feet square. One piece was left untreated, and the other was oil-hardened and annealed. They were fired at by a 6-inch gun with Firth steel projectiles weighing 100 pounds. The striking energy of the blow upon the untreated plate was 2389 foot-tons, and the energy of the blow upon that which had been treated was 2378.5 foot-tons. In the latter case the projectile made an indent of 10.5 inches, so that light was just visible through the center of the bulge at the back of the plate. The projectile rebounded, broken into three pieces. The plate was cracked through, but was whole, and no material was splintered out at the front or back of the plate. In the case of the untreated plate, the shot passed through, and the splintering of the steel round the hole in the front of the plate spread over a space 15 inches across. The splintering round the hole at the back of the plate covered a space 33 inches across. The plate did not remain whole, but broke into six separate pieces.

I am myself no lover of armor. As a member of a great and peaceful trading community, I dislike everything which, by differentiation, tends to lower the comparative fighting value of armed mercantile ships. I dislike also that which reduces the available number, by increasing the individual cost, of ships of war. There is, I think, no such necessary and vital distinction between the man-of-war and the properly armed and protected State auxiliary of high speed as would justify the neglect of such auxiliaries for maritime

warfare. One must admit that at present there are very great differences between them in fighting value, ship for ship, due partly to the very superior subdivision, and partly to the use of armor of high quality in the regular ship-of-war.

The man-of-war proper will probably never abandon the use of armor. We may rather expect that steel and steel-faced armor of the highest quality, instead of being confined to a few ships, called battle-ships, will be employed universally for the defence of the absolutely vital parts of every ship built expressly for war service.

Discussion.

Commander F. M. Barber, U. S. Navy: The distinguished author's paper opens with the observation that it is far more difficult to check leaks in iron ships than it used to be in wooden ships. This is an acknowledgment of a difficulty which is not yet overcome, and one that we are all obliged to confirm. In old times, with wooden ships, there were a few simple and effective means for stopping a leak. We had the old-fashioned shot-plug, looking very much like a loaf of Martinique sugar, which could be driven into a shot-hole of any size, and it would close it up. We had also felt and planking which could be nailed to the side of the ship; and we had a method of flowing oat-meal and sawdust into a seam, which would fill it up. But none of these means will answer for a single-skin iron ship. For local injuries in such vessels we are obliged to resort to various devices which only too frequently prove inefficacious. A hole in an iron ship is nearly always smooth on the outside and rough on the inside. As it is frequently inaccessible from the outside, the difficulty of making a tight joint becomes almost insuperable. One of the methods attempted is to use felt mats; another, to push a kind of umbrella out through the hole, relying on the pressure of the water to spread it and close the opening. We have also tried canvas, weighted so as to go over the side, or a big canvas sail hauled under the ship. But, take a long leak in a seam where the rivets have started, and it is almost impossible to close it.

The author's remarks on the large vessel which he would propose to build are to me very curious and interesting because the plan is perfectly feasible. But if the *Great Eastern*, with her 24,000 tons' displacement was unprofitable, would not this vessel, with over 100,000 tons' displacement prove equally so if she did not possess enormous speed? The vessels described by Sir James Kitson, a short time ago, could soon wrest from this competitor all her freight

business, for in vessels of her dimensions so much time would be consumed in loading and in making the voyage that, with an ordinary steamer, half a dozen trips could be made in the same period.

In discussing the subject of bulkheads, an essential feature of modern practice, not mentioned by the author, is that a bulkhead should be so high as to extend above the water-line when the adjoining compartment is full; for it will be readily understood that if the water can overflow the bulkhead, it will fill the next compartment and then the next, and finally sink the ship. A water-tight bulkhead is of no value unless it be really water-tight. Hibernian as this may sound, I mean that everything depends upon absolute tightness. Holes through a bulkhead must always be avoided. Sir Nathaniel cites the *Oregon*, implying that the ship might have saved herself as well as her passengers if she had been provided with tight bulkheads. But I believe the fact was brought out in an English court that while stowing her cargo, one of her water-tight doors had been opened and that it was left open. Another important feature which I might add regarding bulkheads is that all men-of-war and all the largest merchant-steamers now have twin-screws, which make it practicable to get fore-and-aft bulkheads right through the ship. This creates a longitudinal division which is of first importance; the transverse subdivisions then section the vessel into much smaller spaces.

The distribution of armor-plate is a subject which should be illustrated by drawings of battle-ships; but, in a few words, I may say that the distribution of armor has been largely due to the development of artillery, and that side-armor has not been abandoned, nor will it be, though Sir Nathaniel would lead us to infer that the only function which it can now usefully subserve is that of keeping out light projectiles.* In the old days of smooth-bore guns, a four-inch armor-plate was enough to keep out all projectiles, and it was put all over the side of the ship. When we advanced to rifled guns and greater projectile energy, it was necessary to increase the thickness of the plate. As twenty or twenty-five per cent. of a vessel's displacement is all that can be allowed for her armor, an increase in the thickness of the plate was necessarily attended with a contraction

* I quote the author's words on this subject: "But while side-armor has been largely suppressed and superseded for these reasons, it still possesses one great advantage. It prevents the entrance of water into the zone between wind and water through holes made by light projectiles not capable of perforating armor, but capable of damaging seriously an unarmored side."

or limitation of the area that it became possible to protect. Naturally the armor was thickened over the vital parts. At first a heavy belt was maintained along the water-line, fore and aft, but soon it was found impracticable to carry this belt in sufficient thickness over the full length of the ship; next, therefore, a so-called "protective deck" was introduced below the water-line; and now the thick armor-line is generally confined to the center of the ship.

The value of any belt-armor at the water-line, except for a few special cases, is questioned by the author, one of his objections being that the water-line, though thus protected against light projectiles, would become more exposed to the attack of heavier ones which would penetrate it. If I have understood the argument correctly, I believe Sir Nathaniel has made an erroneous assumption. In our own battle-ships, we provide eighteen inches of belt-armor on the water-line in the middle of the ship, which is just the same thickness as has been adopted for the largest British battle-ships now building. At sea, this could safely keep out any projectile. The armor tapers towards the ends where there are less chances for being hit with a severe direct blow than with a lighter glancing one. The chances at sea are always very much improved for the armor, but as much diminished for the gun. It has been said by able British artillerists that armor virtually doubles itself when it goes to sea. I was fortunate in witnessing the bombardment of Alexandria; not one of the vessels engaged there was penetrated through its armor, though many of the guns on shore were heavy enough to have done it with a square blow, and the vessels were struck many times.

As to the size of vessels, I cannot agree with Sir Nathaniel in considering very large men-of-war less useful than small ones, for I believe that the 14,000-ton ship, as now built in England, represents nearly the maximum of power. It is impossible to get all that is now demanded in power of offence, defence and speed until that displacement is reached.

The future of the armor-plate looks promising; indeed, it appears as though we should soon be able to break up the projectile. The introduction of nickel into steel armor-plate has wrought a great difference; if the projectile energy required to penetrate a wrought-iron armor-plate be taken as 1, then the projectile energy at the English Government reception-tests for compound armor-plates would be represented by 1.07, or seven per cent. greater, and that of the Annapolis competitive test, on September 18, 1890, by 1.17,

or 17 per cent. greater.* I may add that I have seen a nickel-steel plate fired at with a projectile energy 30 per cent. greater, and yet the plate did not crack nor the projectile get through. There is a possibility of hardening nickel-steel armor, without impairing its tenacity, to a degree that we have not yet reached. As long as we do not crack the plate there is a possibility of stopping a hole, for it may be plugged; but this is no longer practicable when the cracks run from one hole to another and pass through the plate.

The development of homogeneous steel armor in England has been greatly retarded by a ruling of the British Admiralty that cracks should not be allowed to pass through the plate. We all know that if ordinary steel is struck very hard it will crack, and that the crack will be likely to go clear through the plate. In a compound armor-plate, which is composed of a steel face for one-third of its thickness, and a wrought-iron back for two-thirds, the cracks only run in and not through, but the whole steel face is liable to come off, as occurred at Annapolis, leaving only thin wrought-iron on the ship. The French Government, on the other hand, has encouraged the manufacture of all-steel plates. It has allowed the through-cracking because that did not appear to diminish the resistance of the plate to repeated shots, and it has sought to compensate for the cracks by increasing the number of bolts which hold the plate to the side of the ship. In this way, the French have preserved the advantages accruing from the high resistance of the steel, and recently, by the addition of nickel, they have obtained the non-cracking qualities of wrought-iron while maintaining the full thickness of the plate. As another result of their persistence, a system of bolting has been developed which, I think, is superior to any other. A large number of small bolts is used in preference to fewer large ones.

The substitution referred to by the author of a solid raft-body at the water-zone in place of side-belt armor is at present a most interesting experiment in the construction of men-of-war. Attention was turned some years ago to stopping the inflow of water through holes made by projectiles in both armored and unarmored vessels. The difficulty of accomplishing this for single-skin iron vessels has already been referred to. In the case of men-of-war, built with a double-skin, the efforts have been directed to packing the space between the two skins. The French were the first to try this plan. After many experiments they have adopted a material called cocoa-

* For the report on this test, with photographs of the target-plates and projectiles, see *Proceedings of the U. S. Naval Institute*, vol. xvi., No. 5, 1890.

cellulose which is made from the hair of the cocoa-nut. It is very light, weighing only 8 pounds per cubic foot, and costing but 17 cents per pound in France. This material is packed tightly between the skins; it is so wonderfully absorbent of water and so expansive, that the hole made by a 10-inch shot going through a $2\frac{1}{2}$-foot belt of it, will close before half a dozen barrels of water can get through it. The French government specifies that the hole thus filled shall remain tight for eighteen hours, it being assumed that in that interval it will be possible to put a patch on the inner skin. The difficulty with cellulose in this form is that it goes on absorbing water, and hence it has become necessary to subdivide the double-skin of the ship into a number of vertical compartments. The vertical bulkheads thus employed add to the weight of the hull and are objectionable.

The next plan was to introduce the raft-bodies which I have mentioned. Their adoption is now advocated for many men-of-war with a view to preserving the protective-deck, which is a curved deck built four-and-a-half feet below the water-line along the side and at the bow and stern of the ship and rising somewhat amidship. All the machinery and all the essentials are beneath this deck, and if the water can be kept from getting below it, the vessel will float. The plan is therefore to fill in, wherever possible, above this protective deck with a material which will obstruct the passage of water without absorbing it. In England, the material talked of for this purpose is so-called "woodite," described by Sir Nathaniel as a substance weighing twenty-one pounds to the cubic-foot, and made of cork and rubber, but none of it has yet been used by the government. In using cellulose for this purpose, it is made up into cubes and covered with lacquered paper. The cubes weigh only eight or nine pounds per cubic foot and are tight under a head of 10 feet of water. In America, we have adopted the cellulose, and it has also been employed by the Russians, Turks and Japanese, but the French are the largest users of it. Its application to a man-of-war is quite simple, but for merchant-ships I question whether we have not to advance much further with machinery and equipment before we can afford to build them with a double-skin all around. When that is done, however, the cellulose filling will add largely to the life-saving and preserving qualities of the ship.

In his concluding paragraph the author gives it as his opinion that every kind of fighting ship should be armored, and in this I would express my full concurrence. If armor is adapted to the

construction of vessels specially intended for fighting purposes, a combination of armor with cellulose or similar material can unquestionably be made to produce a vessel which will be almost impregnable and unsinkable, provided that size is not restricted below 14,000 tons. Vessels of that size can command the speed and, contrary to popular notions, they can be turned in a shorter time and are handier than large merchant-ships, as they have a greater proportion of beam to length. As for unarmored cruisers, I think we are better off without any, for they will be sure to get us into trouble in time of war. The development of our commercial companies by encouraging them to build fast steamers and letting them earn their own living in time of peace, and by providing them with rapid-firing guns in time of war, will furnish us with an auxiliary fleet more efficient than any unarmored men-of-war.

PROGRESS IN MAGNETIC CONCENTRATION OF IRON-ORE.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(New York Meeting, September, 1890.)

SINCE the presentation of the paper upon "The Concentration of Iron-Ore," which the writer, associated with Mr. Thomas A. Edison, read before the Institute in February, 1889 (*Transactions*, vol. xvii., 728–744), much time and money have been devoted to applications of various forms of concentrating apparatus, particularly in treating magnetic iron-ores. An indication of the interest which has been developed is shown in the four papers presented at and printed in advance for this meeting.*

The question, does magnetic concentration pay? is now being discussed by many owners of magnetite deposits, and affirmative answers can be readily obtained from patentees or agents of special machinery, while ultra-conservatism may give a prompt negative reply, based mainly upon the apparent impracticability of handling, crushing, sizing and separating large quantities of ore, and upon the generally

* "Magnetic Concentration at the Michigamme Iron Mine, Lake Superior," by John C. Fowle; "Ore-Dressing by Electricity at the Tilly Foster Mine," by F. H. McDowell; "The Magnetization of Iron-Ore," by Clemens Jones; "The Ball-Norton Electro-Magnetic Separator," by C. M. Ball.

accepted disadvantage of charging material in a finely-divided state into the blast-furnace. A letter received from a member of the Institute (whose name is withheld solely because his absence from the country prevents obtaining his consent to publish it) contains the following statements:

"The texture of the ore itself, the way in which it breaks when crushed, and the tenacity of its adhesion to the gangue (involving also the nature of the latter), the degree of its magnetic quality, etc., have a great deal to do with the general practicability of concentration and also with the choice of an apparatus. I have never yet encountered a case in which, however favorable the conditions, it would pay to mine and concentrate an American magnetite. So far as I can see, at this stage of our practice, waste-dumps only, or the rejected portion of an output, other portions of which have been shipped at a profit covering the whole cost of mining, can be used successfully as the raw material of concentration.

"The fatal element seems to be the actual loss of iron in the tailings; in the case of a material which may be taken as having cost nothing, and which could not be smelted at any price (say a 25 per cent. material), there is, of course, a chance for profitable concentration if the value of the product only exceeds the cost of the process itself. But I found in one case, where the crude material contained 38 per cent. of iron, that the saving in freight, fluxes, and general furnace-expenses effected by a magnetic concentration was not sufficient to balance the waste of iron. In other words, it was cheaper in that case to haul the whole product of the mine to the furnace, add the extra flux required, incur the extra expense due to smaller furnace-product, and take the consequences, than to lose the iron in the tailings after having once paid for mining, crushing, and handling them. I need not say that no money was to be made out of the smelting of a 38 per cent. magnetite under the circumstances, but the net loss of the whole operation of mining and smelting would have been increased, not diminished, by concentrating the ore."

The purpose of this contribution will be to discuss these statements in connection with the data presented in the four papers referred to, and also to offer a *résumé* of efforts made in the United States to solve the commercial problem of concentrating iron-ores by magnetic appliances.

A combination of circumstances has given the writer special opportunities for familiarizing himself with the features of many machines designed for this purpose. They have been examined when operating either experimentally or on a commercial scale, and the possession of several hundred analyses of the results obtained in these investigations, many of them from samples personally selected, permit the possibilities of success to be judged.

The writer's practice in supervising the construction of large separating-works, supplemented by visits to most of the concentrating-plants in the country, and the direction of tests of separating-apparatus prepare him, in point of experience, perhaps, for the present task;

and, as a further qualification, he is free from personal or financial interest in any patented appliance. An endeavor to discuss the subject without prejudice and without bias, drawing conclusions from the data of the four papers, should be interpreted not as commending or condemning special machines, or passing a verdict on the merit of particular concentrates, but rather as broadly considering the possibility of any further improvement.

Review of the Papers.

Concentration at the Tilly Foster Mine.—The contribution of Mr. F. H. McDowell on "Ore-Dressing by Electricity at the Tilly Foster Mine" is valuable on account of the tabulated results, which show the cost of various steps in the process. The Conkling magnetic separator, used at the concentrating-works of the Tilly Foster mine, consists of an endless travelling belt, placed upon an incline, with two pairs of magnets located between the ascending and descending sides of the belt, the magnets of each pair being connected by iron pole-plates, so as to exert their influence over a considerable area of the ascending portion of the belt. The ore is fed upon the belt near the lower magnet; the magnetic particles, drawn by the magnets, are carried up the incline by the belt, while the tailings are washed downward by a stream of water in the opposite direction to the travel of the belt. The lines of magnetic force passing across the belt from the north to the south pole-plates cause the magnetic particles to be grouped in a path which approximates the center-line of the belt, thus connecting the two magnetic poles. It will be noticed that the machine differs materially from the Conkling magnetic separator, described and crudely illustrated in the *Transactions*, vol. xvii., 739.

No continuous attempt has been made at this plant to secure a rich product, as the Lackawanna Iron and Coal Company, which operates the mine and uses the ore in its blast-furnaces, appears to be satisfied with concentrates which, though comparatively lean in iron, carry valuable fluxing material. How far the economic result would be affected by finer comminution and closer separation can only be conjectured from other experiments. In the results it will be noted that the percentage of iron in the crude ore sent to the mill during seven months ranged from 24.96 to 28.57 per cent., with an average of 27.14 per cent., while the concentrates varied from 47.4 to 51.04 per cent., with an average of 49.63 per cent.

A critical examination of this record indicates that the method employed causes a large loss in slimes—that is, in dust and fine

material carried away by the water; for we find in the three months for which analyses of crude ore, heads, and tails are given, that the units of iron in the heads, added to those in the tails, in each case fall short of the iron-contents in the crude ore.

Thus, in May, 3.07 tons of crude ore, carrying 26.8 per cent. of iron were required to produce 1 ton of 50.19 per cent. concentrate and 2.07 tons of tailings, carrying 11.53 per cent. of iron. We then have:

	Units of Iron.	
Used 3.07 tons of crude ore, at 26.8 per cent. iron, . . .		82.28
Produced 1 ton of concentrate, at 50.19 per cent. iron, .	50.19	
Produced 2.07 tons of tailings, at 11.53 per cent. iron, .	23.87	
		74.06
Deficiency, ...		8.22

By this showing 10 per cent. of the iron in the crude ore is not accounted for in either the concentrate or tailings, and only 61 per cent. of the iron in the ore is recovered as concentrate. Similar calculations for some of the other statements give the following results:

For June, out of 82.86 units of iron in the crude ore, 70.96 units are accounted for, leaving a deficiency of 11.9 units, or 14.36 per cent. of the iron in the crude ore. The concentrate contains 61.6 per cent. of the iron in the ore.

For July, out of 71.91 units of iron in the crude ore, 65.58 units are accounted for, a deficiency of 6.33 units, or 8.8 per cent. of the iron in the crude ore. The concentrate contains 68.3 per cent. of the iron in the ore.

These figures emphasize the criticism of the letter, above quoted, upon the loss in tailings; yet the items of cost offered by Mr. McDowell indicate that in handling, under difficulties, the material from a waste-dump carrying less than 30 per cent. of iron and recovering less than 70 per cent. of its iron-contents, an apparently satisfactory commercial return is obtained.

The average cost of producing one ton of concentrate during the six months tabulated by Mr. McDowell was $2.25; and, in addition, 20.7 cents were expended in the preliminary crushing and delivering of one ton of the crude ore to the mill—an item of cost affected by special conditions at each mine. The material and labor expended upon repairs to the mill and machinery also assume proportions which are not uncommon in new installations, particularly where the operations are still more or less experimental. The cost of fuel is

influenced by the nature of the machinery, and by the necessity of using steam-power for elevating the large quantity of water which is required for this particular plant. Mr. McDowell gives the figures for August, 1890, which show that a ton of concentrate was produced at a cost of $1.89, and a critical examination of the items in his statement indicates a possible further reduction.

From the data given, we may assume an average of 2.9 tons of crude ore to be required per ton of concentrate, and the cost of preliminary crushing and delivery to the mill to approximate 20 cents per ton of crude material. Mr. McDowell shows that the mill at Tilly Foster has operated at less cost, while Mr. Fowle gives the expense of the entire operation (working, however, both coarse and fine ore) at the Michigamme mine, Michigan, at only 18 cents per ton. Then 2.9 x 20 = 58 cents, for the cost of crushing and preparing for the stamps and separators sufficient ore to produce a ton of concentrate.

As the output of the mill is increased, the repairs, supplies, fuel, etc., per ton of product will be diminished, and there appears to be good reason for expecting that, in the plant under consideration, a ton of concentrate of the present quality should not cost above $1.75, making no allowance herein for the cost of the crude material. Where fuel is less expensive and water obtainable by gravity, the cost in a similar plant might be brought below $1.50 per ton of concentrate, thus showing a satisfactory margin of profit at ordinary prices, after allowing for interest and fixed charges. To comminute the ore more finely would add to the expense, but it would also augment the value of the ore by increasing the percentage of iron recovered.

The results set forth in Mr. McDowell's table support his first proposition, that "unless the location and other conditions are exceptionally favorable, it will not pay to erect works to treat the material of waste-dumps carrying less than 25 per cent. of iron;" but if leaner tailings had been obtained at the Tilly Foster mine, with the same relative cost of crushing and sizing, then 3 tons of waste-rock, yielding 20 per cent. of iron, would produce 1 ton of 50 per cent. concentrate, and 2 tons of tailings carrying 5 per cent. of iron. Here again the loss of iron in the tailings is emphasized.

The second proposition, "where the lean ore is mined in connection with shipping-ore, there must be a corresponding increase in the percentage of iron to offset the mining and royalty charges," is true if the lean ore is not necessarily or advantageously removed to obtain

the shipping-ore. There are cases where even a partial reimbursement of the cost of mining lean material more than compensates for its removal in permitting convenient operations in deposits where the vein of shipping-ore is small.

The third proposition, "where no shipping-ore is produced, there must be a still further increase in the percentage of iron to warrant the erection of hoisting-, pumping- and dressing-works," is contingent upon the extent and accessibility of the deposit, the character and composition of the lean ore, the cost of mining, comminuting and concentrating it, and the market facilities.

It does not seem equitable, therefore, to base conclusions upon what may properly be regarded as an imperfect separation; for the reasons which influence the Lackawanna Iron and Coal Company to accept a concentrate of fairly large size and comparatively low grade, carrying a considerable amount of fluxing material, would not necessarily hold good for other ores. The conditions which would make the mining of an ore impracticable where the concentrates and tails are of the composition named by Mr. McDowell, would not equally affect the winning of an ore raised to a higher degree of concentration with less loss in the tailings. The circumstances attending the operation of the concentrating-plant at the Tilly Foster mine are peculiar; the owner of the mine and concentrating-plant is, at the same time, its customer, who, as above mentioned, finds it advisable to accept comparatively lean product on account of the fluxing characteristics of the gangue, and the convenience of having the particles of large size.

Concentration at the Michigamme Mine.—"Magnetic Concentration at the Michigamme Iron Mine, Lake Superior," presented by Mr. John C. Fowle, describes the operation of a plant which may also be regarded as one for enriching comparatively lean ores rather than aiming at a high degree of concentration. The present plant is merely the outcome of a practical experiment to utilize ore which would be sufficiently rich for smelting if there were furnaces in the vicinity of the mines, but is too lean to bear the charges of long transportation. No fine crushing is attempted; the smallest screen employed is $\frac{5}{8}$-mesh, while some of the material which passes the separators is of greater size: pieces which the writer saw on the Wenström machine were nearly as large as a hand, and the average size of the concentrate shipped from the Michigamme plant is much larger than that of the product from the Tilly Foster mill. Owing to the coarseness of the product, the daily output of this small plant

assumes very satisfactory proportions. Mr. Fowle gives the iron-contents of the ore fed to the crusher at 52 to 56 per cent.; that of the concentrates at 60 to 65 per cent., and of the tailings at 21 to 24 per cent. Like the Tilly Foster, this plant operates upon an old dump and also upon lean ore brought from the mine; in neither case is any attempt made to approach a perfect separation.

The Michigamme plant starts with a material which, in its crude state, is richer in iron than the Tilly Foster concentrate, and richer than would be indicated by analyses of samples from all the iron-ores fed to American blast-furnaces in proportion to the amount used. While the loss in tailings appears excessive, yet owing to the richness of the crude ore it does not bear so great a relative proportion to the total as the instances cited by Mr. McDowell. At Michigamme mine part of the loss of iron in tailings is also traceable to the hornblende. We may take approximate examples from Mr. Fowle's general statement, using the maxima in one case and the minima in the other. In neither is allowance made for the unknown quantity of dust which is lost.

The minima show:

	Units of iron.
1.26 ton of crude ore carrying 52 per cent. iron,	65.52
From which 1 ton of concentrate was produced,	60.00
Leaving in 0.26 ton of tailings,	5.52

Or 8½ per cent. of the iron in the crude ore.
Iron in tailings by calculation, 21 per cent.

The maxima show:

	Units of iron.
1.28 ton of crude ore carrying 56 per cent. iron,	71.68
From which 1 ton of concentrate was produced,	65.00
Leaving in 0.28 ton of tailings,	6.68

Or 9.3 per cent. of the iron in the crude ore.
Iron in tailings by calculation, 24 per cent.

An interesting feature of the operation of the Michigamme plant is that in all the analyses given by Mr. Fowle, the phosphorus-contents are less in the concentrates than in the crude ore. A finer degree of comminution would doubtless favor a more thorough elimination of phosphorus, but would increase the cost of treatment. The two forms of magnetic separator used at this plant (the Wenström and Buchanan) were illustrated in the *Transactions*, vol. xvii., 737 and 738.

Concentration at the Benson Mine and Elsewhere.—Different from

the above papers is that of Mr. C. M. Ball upon "The Ball-Norton Electro-Magnetic Separator," which illustrates a double-drum machine, and details the results of some experimental trials with various fine and carefully sized ores, the object being either to obtain as nearly as possible a perfect separation of the magnetic portion of the material fed to the machine, producing a rich concentrate and lean tailings, or to eliminate phosphorus from ore rich in iron but carrying apatite. The application of alternate polarity to the material passing the magnets is not novel, although the methods pursued by the inventors possess some new features and practical applications, much commended to those desiring to concentrate magnetic iron-ores.

In comparing the results obtained with the Monarch separator and those detailed by Messrs. McDowell and Fowle, it should be noted that the records at the Tilly Foster mine and at the Michigamme mine were taken from practical daily operation, while, with two exceptions, the operations of the Monarch separator were on a small scale and virtually experimental. Indeed, it seems hardly fair to compare the operation of the Monarch separator, as detailed by Mr. Ball, with the results obtained at the separating-plants given by Messrs. McDowell and Fowle. In a short time, however, the Monarch machine is expected to be in continuous operation on a large scale, and then its working-results will be properly comparable with the figures of the other contributors.

The most comprehensive experiment with Mr. Ball's machine was made at the Benson mines, St. Lawrence county, New York, covering several months' operations. Another fairly large test was one of a fortnight's duration, made under the writer's direction, with the earlier form of the Monarch separator, at the Port Henry mines. A description of this older form, and some of the results obtained from it, are to be found in the *Transactions*, vol. xvii., 740. By Mr. Ball's data the concentrates show a material reduction of phosphorus from that in the crude ore, and therefore may be regarded as a further indorsement of the possibility of bringing some iron-ores within the Bessemer limit by magnetic concentration.

Referring to his tabulated results, it is to be regretted that the mesh to which the crude material was crushed is not mentioned in each case. It is noticeable that some of the instances where this information is given show the effect of fine comminution in increasing the percentage of iron in the concentrate. Thus, we have for Chateaugay ore:

Size.	Percentage of Iron.		
	Crude Ore.	Concentrates.	Tailings.
4-mesh	33.99	59.58	10.64
8-mesh	31.23	62.11	7.32
16-mesh	36.23	66.98	8.12
16-mesh	44.56	66.69	1.26
16-mesh, jig-tailings reworked	19.60	60.48	8.96

In the following two cases we may find another illustration of the loss of iron in tailings. Thus, without allowing for dust which is lost, by computing from Mr. Ball's table A we find:

	Units of iron.
Used 2.096 tons of crude ore, crushed to 4-mesh, and analyzing 33.99 per cent. iron,	71.24
Produced 1 ton of 59.58 per cent. concentrate,	59.58
Leaving 1.096 ton of 10.64 per cent. tailings,	11.66

Or about 16 per cent. of the total amount of iron is lost in the tailings.

The second case:

	Units of iron.
Used 2.09 tons of crude ore, crushed to 16-mesh, and analyzing 36.23 per cent. iron,	75.72
Produced 1 ton of 66.98 per cent. concentrate,	66.98
Leaving 1.09 tons of 8.12 per cent. tailings,	8.74

Or about 11.5 per cent. of the total amount of iron is lost in the tailings.

If, however, the best results on Chateaugay 16-mesh ore, given by Mr. Bell, can be realized on a commercial basis, then we would obtain from 1.51 tons of crude ore, analyzing 44.56 per cent. iron, one ton of concentrate carrying 66.69 per cent. iron, and leave 1.26 per cent. of iron in the tailings, showing a loss of less than 1 per cent. of the iron in the crude ore. This is simply to indicate the value of securing tailings low in iron; such results have not yet been obtained in practice, or if they have been continuously secured the fact has not been made public.

Magnetization of Ores.—Clemens Jones, in discussing "The Magnetization of Iron-Ore" opens up a field of inquiry which was presented to the writer by some experiments made at the Edison

laboratory two years ago, and which, as stated by Mr. Jones, has been practically studied at Pribram.

In "Notes on Iron and Iron-Ores," published by the Geological Survey of Canada in 1870, Dr. T. Sterry Hunt says:

"A process of partial reduction, at a low red-heat, will render non-magnetic iron-ores attractable by the magnet, a reaction of which Chenot long since proposed to take advantage, for the purification of such iron-ores as are not naturally magnetic."

The wide distribution and large deposits of brown hematite ores are attractive inducements for any course of investigation which may advance their economic value by mechanical enrichment. But Mr. Jones apparently undervalues the average yield of iron in American hydrated hematites; the standard of 34 per cent. iron which he gives being evidently for ore mined in Eastern Pennsylvania. The brown hematites of Alabama, Tennessee, Texas, Wisconsin, and many in Virginia and New England will show better results, and the proportion which these and other richer ores in Pennsylvania now bear to the total output will bring the average yield of American brown hematite ores to 40 per cent. or more. Similarly his apparent standard to which hydrated ores can be raised by the use of washers or jigs is lower than good practice warrants.

The treatment of this magnetized brown hematite is assumed to include only the cost of washed ore, cost of roasting and cost of concentration, the latter being given at 2 cents per ton. But the roasted ore except the washer sand (assuming it to be thoroughly magnetized) has to be crushed, sized and handled before it is ready for magnetic separation. Mr. Jones's figures of cost for concentrating and for royalty might with advantage be reversed, for one of the conditions on which any system of concentration must depend for success is its operation on such a liberal scale that the large daily output does not favor the probability of collecting a royalty of 25 cents per ton. Moreover, the host of existing patents for magnetic separators invite many opportunities for interference, by reason of similarity of detail, or broad claims.

To calcine spathic iron-ore mixed with zinc-blende, Mr. C. Linkenbach states that a calcining furnace, 39 feet long and 13 feet wide, will treat per hour 0.6 ton of ore which is finer than 4 mm., requiring the labor of two men and the consumption of 0.11 ton of coal per ton of ore.*

* *Journal of the Iron and Steel Institute*, 1887, No. 1, p. 341.

Recent Progress in Concentration.

A prominent English metallurgist has said:

"Whether the magnetic separation of iron from the materials with which it is found associated in nature is at present an assured economic success or not, so much has been actually done in this direction as to justify the hope that we shall see applied to ores *per se* just as effectual treatment with a view to getting rid of phosphorus and other metalloids as we have already seen applied to the treatment of impure iron by the basic process. We have more than passed the threshold of such a devoutly wished-for consummation, and we may well feel hopeful in reference to the future."

Since the writer's presentation of the paper on "The Concentration of Iron-Ore," previously mentioned, a number of plants for concentrating magnetic iron-ores have been built and the erection of others has been begun. Four are located in the State of New York. The one at the Tilly Foster mine has been described by Mr. McDowell. Another is in operation at the Croton mines, two miles from the Tilly Foster mine, where a large vein of lean magnetite is treated first by heating the ore, to facilitate crushing, and then by passing it through rock-breakers, rolls and screens, and feeding it to Buchanan separators and also to a vertical belt-machine designed for the lessee of the mine. Another plant is at Mineville, N. Y., where lean "New Bed" ore from the mines of Witherbees, Sherman & Co. is fed to crushers, rolls and screens (wet fines being dried sufficiently to prevent clogging the screens) and passed over Wenström and Buchanan separators. This plant, now producing 130 tons of concentrates daily, is so arranged that by duplicating the machinery, one-half of it can be operated upon "Old Bed" phosphatic ore—experiments having demonstrated the practicability of largely reducing the phosphorus and the possibility of converting this into a Bessemer ore. Another feature of the mill is an experimental floor for the purpose of making practical tests of various separators.

The fourth plant in New York, which is rapidly approaching completion, is designed to work a large deposit of lean magnetite at the Benson mines, Little River: the ore is to be crushed and dried, after which it will be passed through rolls and screens, and the separation will be made by the Monarch machines, described by Mr. Ball.

A fifth plant will probably soon be erected at a large deposit near West Point, N. Y., where the Edison machine will be employed.

In New Jersey an extensive concentrating-works is practically completed, in which the ore from the Ogden mine will be treated by passing it through crushers, rolls, screens and Edison separators. A smaller plant is being constructed for the Weldon mine, where the separation will be performed by a belt traversing a magnetic drum and passing through a trough of water.

In Pennsylvania a plant was in operation near Bechtelsville, Berks county, but the lean and uncertain character of the deposit did not warrant the continuance of operations. This plant consisted of crushers, rolls, screens and Edison separators; the separator was described in the *Transactions*, vol. xvii., 741.

In addition to the concentrating-works at Michigamme, Michigan, described by Mr. Fowle, there has been one in operation near Humboldt, Michigan, in which the Edison machine was used, but the fine state of subdivision in which the iron occurs with the gangue made the operation both difficult and expensive, and produced material which, up to the time that the plant was destroyed by fire, had been used only for rolling-mill fix. A plant is also in operation, and about being enlarged, at the Cranberry ore-mines, in North Carolina, using Wenström separators.

At the present writing there are, in the United States, nine Wenström machines employed in separating iron-ore, and four that are used in steel-works upon the waste from converters, etc. Four Edison machines are likewise in practical operation, separating iron-ores, and four Buchanan machines are used on iron-ores, others being employed in sorting other magnetic from non-magnetic substances. Three Conkling belt-machines and two other belt-machines are in use, or about being constructed, and six Monarch machines are contracted for and will be in operation by the time this paper is published. In addition to these there are a number of magnetic separators which have not yet developed beyond models and laboratory apparatus. It will thus be seen that the amount of attention devoted to the process has resulted in placing it fairly beyond the experimental stage, and that considerable progress has been made toward practically demonstrating the production of concentrates by magnetic separation. Up to the present time, however, owing to difficulties which have beset the installation and operation of the various works, and in one case owing to the practical failure of a supposed available ore-supply, the whole quantity of concentrated iron-ore has not been sufficient materially to affect the market or draw the general attention of consumers to it. It has, however, been

quite large enough to show that the objection urged against the use of concentrated ore on account of its size, is not confirmed by actual trial; for producers of concentrated ore have found no difficulty in disposing of it in considerable quantities along with other ores, the price being practically unaffected by the comminution of the material. Probably 90,000 tons of separated iron-ore were marketed in the year 1890, the product being nearly equally divided between magnetic and wet-jigging machines. Inexperienced prejudice against the use of finely-comminuted ore will have to be overcome; and a candid investigation will reveal that the fine physical condition of the ore is less of an objection in a blast-furnace than most iron-workers believe. The "dirt-troubles" in blast-furnaces often attributed to the use of fine ore are undoubtedly more *dirt*-troubles than *fine-ore* troubles, and if much of the "dirt" were removed from the ores these annoyances could be largely obviated. Many blast-furnaces use, as a permanent part of their charge, ore that is no larger or coarser than concentrates. The claim that the fine ore will be blown out of the furnace offers no reason why such should be the case. The blast-pressure is intended to permeate all parts of the furnace, but not to cause a current at the tunnel-head that will be strong enough to carry off grains of ore which are specifically four or five times heavier than water. The descent of the fine ore in the blast-furnace is to many another source of anxiety, which undoubtedly will be relieved by the method of charging and operating the furnace.

The amount and character of the gangue of an iron-ore is of great importance to a blast-furnace manager, and one of his constant problems is to flux off the impurities. But if these can be largely eliminated, and, at the same time, the percentage of iron in the ore be increased, there would seem just reason for expecting that a given furnace would require less flux, less fuel and labor per ton of metal, less expense in handling cinder, as well as less anxiety for the manager, while the increased product would reduce the charges for interest, superintendence, repairs, insurance, etc., per ton of iron made.

The statement in the letter above quoted as to the impracticability of working a mine profitably on lean concentrating ore alone, coming from authority which the writer recognizes, cannot be lightly dismissed; yet, judging from what has actually been accomplished, it seems possible, with the aid of rock-drills and high explosives, to reduce the cost of working large deposits of lean ore to such low figures as will in many cases permit mining for the sole purpose of

concentrating the ore. The object of concentration at present, however, is mainly to utilize the lean ore, which can be, or in many cases must be, mined along with the richer ore. Oftentimes the possibility thus offered of securing without absolute loss larger gangways and drifts and more room throughout the mine will be a decided advantage. The instance cited in the letter, where a 38 per cent. ore could be carried to the furnace and smelted more cheaply than it could be concentrated and then smelted, points either to close admixture of gangue and magnetite, imperfect methods, or low cost for fuel and transportation. There are probably few cases where such a lean ore can be smelted to better effect in its natural condition than by first enriching it by concentration, unless the ore possesses in itself other valuable qualities besides the iron. Ore yielding less than 38 per cent. of iron is now concentrated at costs which leave ample margins for mining the crude material, if it occurs under what may be termed "average conditions."

The practical results obtained in concentrating iron-ores by magnetism cannot be expected to harmonize closely with empirical tests which are controlled by special conditions; for no mere experiment of short duration upon small lots of ore will betray the varying characteristics which large quantities of the ore may exhibit. Atmospheric conditions which affect the operation more or less from day to day, and the inevitable derangements of mechanism which arise in practice, also make it impossible to obtain as satisfactory results from work carried on in a large way as may be procured by treating carefully sized samples over and over on the same machine. But the value of concentration that is already indicated by practice should certainly not be underrated. Experience in treating or observing the treatment of a variety of ores, or of the same ore by different machines, shows that where phosphorus as apatite is associated with magnetite, the amount of phosphorus in the concentrate can be invariably reduced if the ore is sufficiently comminuted to permit a mechanical separation. The finer the comminution of the ore and the thinner the layer of crushed material passing through the magnetic field, or the oftener it is acted upon by magnetic influences, the greater will be the reduction of phosphorus. Whether a given non-Bessemer ore can or cannot be converted into a Bessemer concentrate, will depend largely upon the size of the crystals of magnetite and apatite, and the possibility of exposing each particle to unobstructed magnetic influence, or a number of the particles together to repeated magnetic influences. It is not to be expected that, ordinarily,

ores can be comminuted as cheaply as they can be coarsely crushed. One factor, therefore, in determining the practicability of converting a non-Bessemer into a Bessemer ore will be the expense of reducing it to such a size as will permit a substantial elimination of the phosphorus. Another will be the capacity of a plant of given cost, or the maximum rate at which the ore can be fed to the separating machine without impairing effective work. In exceptional cases a fine comminution may interfere with the ready sale of the product, and if the material is made into briquettes, this, like fine crushing, will add to the cost.

Experiments upon magnetite containing sulphur have shown that in some ores, where the sulphur occurs as pyrites, more of it can be eliminated by magnetic concentration than by roasting, but where the sulphur occurs as pyrrhotite, some analyses have indicated a higher percentage of sulphur in the concentrate than in the crude ore.

Little has been done in the treatment of titaniferous ores of late years, but a decade ago considerable attention was paid to them, and, as early as 1870, Dr. T. Sterry Hunt wrote:* "Where, however, the impurity is chiefly titaniferous iron, as in the Moisie sands, the separation may be readily effected by means of magnets, a process which is equally advantageous where magnetic iron-ore is mixed with lighter impurities, as quartz or siliceous minerals." He gives some analyses of iron-sand from the lower St. Lawrence. The composition of the unwashed black sand from Moisie, holding 49.1 per cent. of magnetic grains, is shown in column A of the subjoined table. That of the material from which the magnetic portion had

	A.	B.	C.	D.
Protoxide of iron	70.10	58.20	85.79	56.38
Titanic acid	16.00	30.74	4.15	28.95
Insoluble, chiefly quartz	5.92	6.14	1.95	8.75
Oxide of manganese	.....	.....	.40	1.10
Lime	.....	.....	.90	.95

been removed is given in column B. The same sand, after being washed, contained 52 per cent. of magnetic grains, which, upon analysis, showed the composition marked C, while the non-magnetic

* *Vide* "Notes on Iron and Iron-Ores," prepared for the Geological Survey of Canada.

portion was of the composition marked D. These results indicate that under some circumstances the percentage of titanium can be materially reduced by magnetic separation.

Some inventors who have entered the field with magnetic ore-separators have apparently based their experiments upon the removal of iron-wire from grain, nails from cotton, and iron-filings from brass, etc., ignoring the fact that in most lean magnetites the difficulty lies in so breaking up the material as to have only magnetic and non-magnetic material to handle. However fine the comminution, there are few ores which offer such a simple problem for concentration; were it otherwise, practically pure magnetite and barren tailings would result from any mechanical contrivance which brought the material within the sphere of magnetic influences, and permitted these to act freely on each particle. In a majority of cases, particles of gangue are attached to grains of magnetite, forming an intermediate product, and the amount of this which finds its way either into the concentrate or into tailings, reduces the percentage of iron in the former or augments the loss of iron in the latter.

In converting a phosphatic ore into a product within the Bessemer limit it is well to remember the close approach to perfection which the concentration must reach. If $1\frac{3}{4}$ tons of ore analyzing 40 per cent. iron and 1.2 per cent. phosphorus should produce, by treatment, 1 ton of concentrate yielding 64 per cent. iron and .06 per cent. phosphorus, the amount of phosphorus to be removed in the operation would be shown as follows:

	Tons iron.	Tons phosphorus.
$1\frac{3}{4}$ tons of crude ore contain	.70	.0210
1 ton of concentrate contains	.64	.0006
Leaving in $\frac{3}{4}$ ton of tailings, ...	.06	.0204

Expressing the amount of phosphorus in pounds, the $1\frac{3}{4}$ tons of crude ore contained 47.04 pounds, and the ton of concentrate has only 1.34 pounds. From this it appears that a very satisfactory commercial result might be obtained if 91 out of every 100 pounds of iron in the assumed ore were recovered in the concentrate, while a product of Bessemer grade would only be obtainable if 97 out of every 100 pounds of the phosphorus were removed.

As far as the enrichment of lean magnetite by crushing, sizing, and separating the magnetic from the non-magnetic material is concerned, it may fairly be pronounced a commercial success. The cost

of the process, as in the case of phosphatic ores, depends on the degree of comminution to which the ore must be reduced to permit a separation. It varies greatly for different ores, as explained in the writer's earlier paper; but from the data given by Messrs. McDowell and Fowle and after allowing for the cost of handling and interest on the purchase or royalty for the use of a concentrating apparatus, there are few places where the expense of crushing, sizing, and separating one long ton of crude ore should exceed 50 cents, and in a majority of cases it should be less than this.

Assuming, for the purpose of illustration, that a ton of crude 40 per cent. ore is obtained for $1.00, and allowing a loss of 8 per cent. of iron in the tailings, then the production of 1 ton of 64 per cent. concentrate will require $1\frac{3}{4}$ tons of the crude material, and we have for the cost, $1\frac{3}{4}$ tons of ore at $1.00 per ton, plus 50 cents per ton for concentration, = $2.63, or about 4.1 cents per unit of iron in the concentrate.

If, however, the 40 per cent. ore is raised only to 56 per cent. and at a loss of 14 per cent. of iron in the tailings, the requisite amount of material will be 1.615 tons, costing $2.42 or 4.3 cents per unit.

The first assumption is a closer approach to perfection than has been regularly obtained at any of the plants in operation, but there appears to be no reason for doubting the practicability of such concentration on a large scale. The second supposition represents some of the actual work detailed in the papers that have been considered, from which it would appear that there is an opportunity for dividend as well as interest in producing a concentrate which would find a fair sale. Naturally, any variation in the value of the material or the cost of concentrating will affect these figures, but they are presented as a general suggestive statement from which those who are interested can calculate more closely the possibility of profitable concentration in any particular locality. While dissenting from the broad statement contained in the letter at the beginning of this paper, that deposits of lean magnetite cannot be profitably exploited for concentration only, a hearty endorsement is given to the further assertion that "the fatal element seems to be the actual loss of iron in the tailings."

The following formula, obtained from Mr. H. Helms, of Port Henry, New York, has been found useful by the author in determining the amount of crude ore of a known iron-contents which is required to produce concentrates of various degrees, the iron in the tailings being also known. *The percentage of iron in concentrates*

minus the percentage of iron in tailings, when divided by the percentage of iron in crude ore minus the percentage of iron in tailings, gives as quotient the tons of crude ore necessary to produce one ton of concentrate; thus:

$$\frac{\text{Heads} - \text{Tails}}{\text{Crude} - \text{Tails}} = \text{Tons of crude ore to produce 1 ton of heads.}$$

This formula can be conveniently applied to test the figures of the several papers that have been discussed, as well as to verify the statements of this paper. It has been used by the author to estimate the approximate losses of iron in the tailings. In the absence of knowledge concerning loss in dust and slimes, and the actual weights of the crude ore, concentrates, and tailings, these calculations cannot be expected to agree absolutely, but the figures are sufficiently close to show the comparative loss under different conditions.

The figures of this paper are intended to place the subject of concentrating iron-ores by the use of magnetic separators in such a light as to permit a general application of its possibilities to individual mines. Each mine and each ore is practically a problem of itself, requiring study as to the best means of utilizing the lean material, and liberal discussion should be encouraged. At the same time the question should be approached with candor, unhampered with the traditions of practice, at a time when with less perfect mechanism and less knowledge of the action of magnetic-force, with fewer transportation facilities and inferior mine-working appliances, success was not possible where it now may be assured.

In conclusion, the following excerpt is made from an editorial which appeared in the *Iron Age*, entitled "Magnetic Ore Separation." There is undoubtedly much that is true about the fascination of the process and the tendency to underrate its difficulties. But there is just as undoubtedly a large field in this country for magnetic iron-ore concentration. Like every other industrial field it must be worked intelligently, or satisfactory results cannot be expected.

"To meet a real want in some parts of the country there has been a rush of inventors into the field of magnetic ore-separation. . . . . There has been a tendency to overrate the future of magnetic ore-separation and to underrate its difficulties. . . . . There is something very fascinating in the production from lean ore of a concentrate in which only very few particles of foreign matter can be detected. No one who has approached the subject has escaped the glowing enthusiasm and the

air of triumph of an inventor over such an achievement, particularly if it is coupled with a reduction in the phosphorus. Yet there are very few of the promoters of such work who have an adequate conception of the costs and of the losses involved. They generally begin by assuming that their raw material, the lean ore, costs them nothing. It is true that in some instances there are accumulations of it on the dump piles. Still it costs money to pick it up and convey it to the concentrating mill. Then the ore must be crushed and screened, the mesh depending upon the character of the ore. . . . . To all the cost of the preliminary operations previous to the magnetic separation proper must be added a heavy percentage to allow for loss. For anything like a close-grained ore this loss is not less than 15 per cent. and may rise to 20 per cent.

"We have not seen thus far a single statement of cost in print which would be considered business-like by a manufacturer. Such a statement should include the cost of mining or delivering at the mill enough raw material to produce one ton of concentrates, the cost of drying and crushing, of concentrating proper, and the cost of royalties, repairs, interest and general expenses.

"Our conviction, from what we have learned thus far, is that the total will prove an uncomfortable sum to enthusiasts. We do not mean to convey the impression that magnetic concentration has not a brilliant field before it; but, generally speaking, hopes have been raised on the basis of underestimates of costs, which are sure to lead to disappointment."

THE GENESIS OF THE EDGAR THOMSON BLAST-FURNACES.

BY WILLIAM P. SHINN, PITTSBURGH, PA.

(New York Meeting, September, 1890.*)

MR. GAYLEY'S admirable paper on the "Development of American Blast-Furnaces" has set forth very fully the history of the *development* of the Edgar Thomson furnaces since the construction of Furnaces A and B; but it throws no light on the genesis of these furnaces. That earlier history it is the purpose of the present paper to record.

Mr. Gayley truly says: "A new era in the manufacture of pig-iron began in 1880 with the putting in blast of the Edgar Thomson furnaces." That era had its beginning with the construction of

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* Read by title at the meeting, but not published until after the New York meeting of the Iron and Steel Institute in October, when Mr. Gayley's paper was presented.

Furnace B, which was not in any sense an accident, but the result of most careful design, based upon truly scientific deductions from known facts.

In September, 1878, when the construction of Furnaces A and B was decided upon, the writer was General Manager of the Edgar Thomson Steel Co. (Limited), and the preparations for their construction were in his charge. For some two years I had been very desirous of having the company (in which I had a large interest) erect its own furnaces, and I had been gathering information bearing upon the subject.

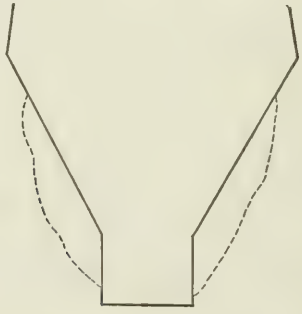
I had visited many furnaces, and invariably was told by the manager, in starting on our inspection of the plant, that "we are a little short of steam this morning" (item—have plenty of boilers); or, "we are short of blast to-day" (item—have greater blowing-capacity); or, "we have been a little short of hot-blast to-day" (item—have a better supply of stoves); or, "we are very short of gas for our boilers" (item—have a ready means of firing in addition to furnace-gas), until I had become convinced that a great improvement could be effected by supplying these "items" and preventing these chronic shortages. Again, I had noticed that when the blowing-engine and hot-blast stoves of two furnaces had been concentrated upon one stack, as had been done in the Shoenberger furnaces in Pittsburgh in 1877, a great increase in output was the result. In the year 1877 Furnace No. 1 was out for repairs, and the whole blowing and blast-heating capacity was used upon No. 2, with the result of increasing its output from 1042 tons per month to 1521 tons. Thus the discovery that increased blast would raise the output was not original with Mr. Julian Kennedy, although he made very good use of it, as described by Mr. Gayley.

While the plans were under consideration, and before any of the details had been determined upon, I was asked by two of the partners, both interested in the Lucy furnace, what action I was taking in regard to the work. I replied that there were two important questions to be decided before plans could be prepared—whether we should have compound engines, and whether we should have iron or fire-brick stoves; and if fire-brick, of which type. The answer was: "We want no compound engines or fire-brick stoves. Build a furnace that will make 100 tons of Bessemer iron per day; the Lucy furnace does it, and you cannot do better than to follow the Lucy"—*i.e.*, copy the design and details. I argued to the contrary, alleging that possibly as much improvement over the Lucy furnace

might be made as was made over the Struthers furnace when the Lucy was built. My view prevailed, and I was authorized to "send out a commission" to investigate and report. That commission consisted of Mr. James Hemphill, of Pittsburgh, the eminent mechanical engineer, member of the firm of Mackintosh, Hemphill & Co.; Mr. H. M. Curry, then manager of the Lucy furnace, and the writer. The commission visited every furnace-plant but one in the United States having fire-brick stoves, and returned to make a unanimous report in favor of the adoption of the Siemens-Cowper fire-brick stove.

That report led to the revolution in blast-furnace construction and practice, which began in 1880, as stated by Mr. Gayley. The report was unanimously adopted by the Board of Directors. As Taws & Hartman, of Philadelphia, were the sole agents for the Siemens-Cowper stoves in the United States, they were employed as the

FIG. 1.



engineers, and were directed to prepare the plans; and to Mr. John M. Hartman, no less than to the writer, should be ascribed whatever credit may be due for the erection of Furnace B.

Another point had caught my attention in observing the results of working furnaces, namely, that when they were blown out for relining, the angle at the top of the "well" had always been cut away from the original form shown by the full lines, Fig. 1, to the form shown by the dotted lines; and there were reasons for believing that this cutting away took place during the first three months of the blast. Hence I was led to query, why put in so many fire-brick where they will simply be melted away, at an expense for both brick and fuel?

Of Furnace A not much more need be said than has been mentioned by Mr. Gayley. Originally a furnace with an 11- or 12-foot

bosh, it was bought to save a debt, and when erected as Furnace A it was made as large as the mantel would permit. The lines shown in Mr. Gayley's Fig. 2 approached very nearly to my conception of what a furnace should be. With an extreme diameter of 13 feet at the bosh, the hearth was 8 feet 6 inches in diameter, or almost 66 per cent. of the bosh diameter. This small furnace of only 13 feet bosh produced in its third month, according to Mr. Gayley, 671 tons of output in a week, or an amount nearly equal to the best output of the Lucy furnace with 20 feet bosh; thus were the large hearth, and straight inwalls fully justified. My intention was to have made the hearth of Furnace B 12 feet in diameter, and the side-line straight from the bosh to the bottom, forming of the lower section a true frustum of a cone; but in some of these details my advice was not followed, particularly after I left the company on October 1, 1879.

In the matter of the blowing capacity, the highest volume of air blown into the Lucy furnace had been 15,500 cubic feet per minute. The engines for Furnace B had been designed for blowing 18,500 feet per minute, and the stoves contained 92,000 square feet of heating-surface. It is stated by Mr. Hartman, in his discussion of Mr. Kennedy's paper on "Hot-Blast Stoves at the Edgar Thomson Furnaces,"* that Mr. Kennedy increased the volume of air blown into Furnace B to 28,000 cubic feet per minute, and the stoves heated that enormous volume to 1050° Fah., which shows that the allowance of power and capacity for heating was very liberal, even for the large volume of air provided for.

The best work of the Lucy furnace on a mixture yielding 65.3 per cent. iron had been 107 tons of pig in twenty-four hours, while Furnace B, working on a mixture yielding only 55 per cent. iron made on the first run:

1880.	Tons.	Average tons per day.
May,	3718	120
June,	4318	144
July,	4345	140
August,	4601	148
September,	4221	140
October,	4722	152

Here was an increase in average output over the Lucy's best work of over 30 per cent., and that on a mixture of ore yielding one-sixth

* *Transactions*, vol. x., 495.

less iron. Truly this indicated a "new era in the manufacture of pig-iron," and pointed the way which has been so skilfully followed by Mr. Kennedy and Mr. Gayley, until the output has been increased to the enormous amount of 330 tons per day. Mr. Gayley truly says, referring to Furnace B: "But while the hearths of various furnaces had been enlarged after they were in blast, yet no American furnace up to that time had been *constructed** with so large a hearth as this one of the Edgar Thomson works. In another respect this furnace was well prepared by its designers for a high productive capacity, viz., in its equipment. *Fire-brick stoves of the most approved type were erected.** Substantially-built blowing-engines were provided, and they were rendered efficient by an ample supply of boilers—*a point in which other furnaces were then sadly lacking.** At the same time all the flues and mains were constructed sufficiently large, and in the most substantial way. In fact, no furnace previously erected had been planned on such a liberal basis; consequently, large yields were to be expected."

I have shown that these "liberal preparations" for a "large yield" were not accidental or fortuitous. They were the result of design, earnestly insisted upon by the writer of this paper, endorsed by the committee, Messrs. Hemphill, Curry and Shinn, and most ably executed by Mr. Hartman.

"Honor to whom honor is due."

THE COPPER-RESOURCES OF THE UNITED STATES.

BY JAMES DOUGLAS, NEW YORK CITY.

(Read Provisionally at the New York Meeting, September, 1890, and Completed with the Statistics for the Current Year.)

THE development of the copper-resources of this country has kept close pace with the unfolding of its geographical area to commerce. In colonial days, when our English ancestors occupied only the Atlantic seaboard, insignificant quantities of copper were mined in Connecticut, New Jersey, and Pennsylvania, and the ore was shipped for treatment to England. The first steam pumping-equipment

* Italicized by the present writer.

erected in this country was probably that used by Josiah Hornblower, in the year 1753, at the copper-mines at Belleville, New Jersey, almost within sight of this city.

That copper in the native state existed on Lake Superior was mentioned in the *Relations des Jésuites* as early as 1659 or 1660. The Rev. Claude Allouez described a stray mass in 1666. But it was not until 1771, a little more than a century later, when this region had passed from the dominion of France to that of Great Britain, that Alexander Henry organized a company in England, with the Duke of Gloucester, Charles Townshend, and other notables as incorporators, to mine copper on the Ontonagon river, the region whence came the stray blocks which had grown so famous by repeated mention in the *Relations* and by the reports of various travellers. Failing, in 1771, to find the copper in place, Henry renewed operations in the following year on Michipicoten Island, on the north shore, but only to score another failure. What little copper-ore he raised was shipped to England, where ship and cargo were seized and sold for the company's debts. Mr. Townshend's losses in his Lake Superior venture may have embittered his feelings toward America, and perchance have been one of the indirect causes of the unreasonable financial policy towards the Colonies, which precipitated the Revolutionary War, and for which he, as President of the Board of Trade and subsequently as Chancellor of the Exchequer, was responsible.

Then stagnation followed for three-quarters of a century. Even if no other cause had interfered, the unsettled ownership of the Lake region must have retarded mining until the date at which it actually commenced, in 1844. For it was not until 1796 that Michigan was ceded by Great Britain to the United States; not till 1837 that Michigan was admitted to statehood, and not until 1842 that the Indian titles to the lands were extinguished. Meanwhile steamers had replaced canoes on the Lake, and in 1841 Dr. Houghton had given to the world the first authentic and scientific account of the geological structure of the Keweenaw series of rocks and of the mode of occurrence of their native copper. In 1844 the Cliff mine was opened, and in 1845 Lake copper appears as an item in the world's production. That in those days our copper enterprises were in an infantile condition, is shown by the fact that although in 1850 the Lake produced only 570 tons, yet that small quantity was over 88 per cent. of the total production of this country. Leplay, Secretary of the Commission of Mining Statistics in France, estimates the world's pro-

duction at that date at 52,400 tons, and assigns to this country a consumption of 6100 tons, which was supplied by about 5500 tons of imported and 600 tons of domestic copper.* After 1850, and until the yield of the Calumet and Hecla mine began to be felt, in 1867, the relative production of the Lake region fell off, owing to the appearance successively of Vermont, the Southern States, and California in the list of producers. Vermont copper came almost entirely from the Ely mine, in the Vershire district. This mine was discovered in 1820, but began to be worked only in 1850, and thenceforward yielded from year to year an increasing quantity, till 1880, when its output amounted to 3,186,175 pounds. It was closed in 1882, but is now being re-opened. By far the largest producer in the Southern States during this period was the Ducktown group of mines, in Tennessee, discovered about 1846. Systematic work was begun in 1850, and before 1854, when smelting-works were erected at the mines, over 14,000 tons of very rich oxides and sulphurets had been shipped. The experience gained there was used in discovering and working similar ore-masses in North Carolina, Virginia, Georgia, and Alabama, most of which yielded at this period no more than trifling quantities, but sufficient in the aggregate to swell the total, which nevertheless remained insignificant. California, during this period, shipped more copper than she has shipped since. Her export of ore in 1864 reached 14,315 tons, chiefly from the Union and Campo Seco mines of Calaveras county, which were opened on lenticular masses of sulphuretted ore, imbedded in talcose slates. The Union mine was closed in 1866, but was re-opened three years ago. Thus it came about that the percentage coming from the Lake region declined from 96 per cent. in 1849 to 68.08 per cent. in 1866. But in 1867 Calumet and Hecla appeared with a shipment of 603 tons, followed in 1868 by 2276 tons, and year after year by an ever-increasing contribution, thus keeping up the Lake percentage to between 80 and 90 per cent. of the total. This continued until 1881, when a sudden fall of the relative Lake Superior production from 82 per cent. to 76.01 per cent. marked the advent of her western rivals.

Up to that date the line of the Union and Central Pacific railroads had offered the only through and direct route across the continent, and it happened that along that route no large copper-deposits existed. But simultaneously, in 1881, the Union Pacific road reached

* Foster and Whitney Report, Part I., Copper Lands, p. 157.

the neighborhood of Butte, Montana, by its Utah and Northern branch, and the Southern Pacific road completed the laying of its track through Arizona and New Mexico. Previous to this, the Longfellow mine at Clifton, Arizona, had been worked, and its copper bullion carted 700 miles to the nearest railroad-terminus in Kansas. The richer argentiferous copper-ores of Butte, Montana, had also been hauled a distance of more than 400 miles to Corinne, on the Central Pacific railroad. But mining operations were necessarily restricted in each district to exceptionally rich ores, until the arrival of the railroad introduced mineral-fuel. Arizona and Montana immediately responded to the stimulus of cheap transportation, and mining and smelting on an extensive scale commenced in the extreme south and the extreme north of our wide western domain. The next event which marked a stride in our progress toward preeminence in the copper world, occurred in 1883. It was the discovery of large bodies of copper-ore in the Anaconda mine, the central and culminating point of the immense Butte lode. With a bound this great mine sprang to second place in the rank of production, and within five years overtook and passed its great competitor, the Calumet and Hecla. Since then the growth of production in both the Lake region and the West has shown no abatement, but the relative increase has been in favor of Butte and against the Lake.

In 1881, therefore, these three sources of supply—namely, the native copper-deposits of the Lake region, the oxidized ores of southern Arizona, and the sulphuretted ores of Montana—constituted our chief resource, and they remain so to-day. In 1889 our total production was divided as follows:

	Per cent.
The Lake region,	35.81
Montana,	43.07
Arizona,	13.34
All other native ores,	5.69
Imported ores,	2.09
Total,	100.00

Lake Superior.

In reviewing Lake Superior mining we find that three groups of mining companies have been successively formed, and that each group began operations on a form of deposit, there being three forms in which copper occurs in the rocks of the Keweenaw series.

The first group of companies worked *veins* in search of mass-copper; the second group worked the *ash-bed* of Keweenaw county and its equivalents, and other *amygdaloid-beds* in the Portage district; and the third group of companies worked the *conglomerate-beds*.

Mass Mines.—The only mine of the first group now producing any considerable quantity of copper is the Central. The Cliff, which was the first mine systematically worked on the Lake, having begun to produce copper in 1844, has yielded very little within the past decade. The same is true of the still more famous Minnesota, opened in 1848 in Ontonagon county on an interbedded vein; of the Phœnix, opened in 1844; the Northwestern and Copper Falls in 1845; the Northwest in 1847; the Ridge in 1850; the National and Evergreen Bluff in 1853; the Mass in 1856, and of a host of small mines which never added much to the copper supply, but did add largely to the assessment-roll. All these companies commenced operations in search of mass-copper either on the transverse veins of Keweenaw county toward the point of the promontory, or on the interbedded veins of the Ontonagon country toward the base of the promontory. The Copper Falls Company, failing in its quest, turned its attention to one of the amygdaloidal beds through which the vein cut, and which at the point of intersection seemed to enrich it. Thus, though the Copper Falls Company remains to-day a small producer, the Central is the only one of the original group of mass-mining companies which still actively works a vein. The other mass-mines yield but a trifle, and this is recovered by tributers. The experience of other mass-mines has been confirmed by the Central, namely, that the masses grow fewer as depth is attained, and that they are distributed in the veins in no regular or assignable order. The Central may not have produced as many large masses as the Cliff and the Minnesota, but it yielded at least one even larger than the famous 500-ton mass of the latter mine. It consisted of a nucleus from which 600 tons of copper were cut, and a number of subordinate masses that were connected with the central mass by distinct filaments. The total amount of copper yielded by this series of related masses was about 1200 tons. Even in its most prosperous days the product of the Central was about evenly divided between mass- and stamp-copper. The vein varies in width from a mere seam to 25 feet, and the productive chute of ore below the 900-foot level does not average more than 300 feet in length. The possibilities for a large production do not, therefore, exist, if operations are confined to this vein alone; but exploration for other veins

is being vigorously pushed. The mine has reached the thirty-first level—a vertical depth of 2900 feet. Above the thirtieth level (2800 feet deep) the vein was shattered in passing through a bed of amygdaloid, and on that level was completely obliterated by an underlying bed of conglomerate. Fears were entertained that the bottom of the mine had been reached, but recently the vein has been cut in sinking the shaft below the 2800-foot level. It was thrown 220 feet to the west. Where rediscovered it is of average size and productiveness. The mine has yielded over 20,000,000 pounds of copper, and has proved so profitable from the start that even the first year's expenditure was repaid by the first year's returns. Though the vein has generally been narrow and the yield on the average only 1.9 per cent., the management has been so excellent, and the cost of extraction so low, that dividends amounting to $2,000,000 have been paid on a capital of $100,000.

Amygdaloid Mines.—The second group of companies was formed to work the ash-beds of amygdaloidal diabase, which, though traceable throughout the whole length of the Keweenaw series of rocks, appear to be richer in copper on the shores of Portage Lake than elsewhere. As already remarked, the Copper Falls (a mass company) diverted its attention early in 1851 to the working of an ash-bed. Although the extravagant hopes which were based upon excessive estimates of the yield of the ash-bed were not realized, this company's endeavor stimulated speculation and honest work on the amygdaloid-beds of Keweenaw county and of Portage Lake district. Many companies were speedily organized for the purpose of developing these properties. Contrary to the fate of the mass companies, most of them have maintained a more or less vigorous existence to this day. Of the companies which appear in the list of producers within the past decade, the following belong to the second group and date back their beginning to the second period of Lake mining, between the years 1850 and 1865:

The Isle Royal Company commenced work in 1852.

The Grand Portage Company commenced work in 1853.

The Sheldon and Columbian Company commenced work in 1853.

The Pewabic Company commenced work in 1853.

The Huron Company commenced work in 1855.

The Quincy Company, though organized in 1848, commenced active work only in 1856.

The Franklin Company commenced work in 1857.

The Albany and Boston Company commenced work in 1860.

The South Pewabic Company (now the Atlantic) commenced work in 1865.

The Kearsarge and Wolverine are the only mines recently opened on an amygdaloid-bed in the Portage district. The former made its first return of 829,125 pounds of copper from a rock yielding 1.02 per cent. in 1888.

Of this group of mines by far the most remunerative has been the Quincy, but the one which exhibits the most extraordinary minimum cost of production is the Atlantic. These mines are situated on opposite sides of Portage Lake. The Quincy crowns the summit of a hill which rises boldly from the lake at the point where the town of Hancock seems with difficulty to be climbing up its steep slope. The Atlantic is on the southern shore of the lake, to the southwest of the Quincy and on a more westerly bed. Both mines are in rock classed as amygdaloid, but the Quincy amygdaloid differs from the Atlantic in appearance, and in the fact that it carries 40 per cent. of its copper in masses and what is known as "barrel-work"—that is, metal in blocks so coarse that the earthy matter can be separated under the hammer without dressing. It can therefore be hand-sorted. Nearly all the Atlantic rock passes without selection through the stamp-mill, and the product is almost exclusively "stamp-work." The Quincy bed, as might be anticipated from the mode of occurrence of its copper-contents, is more variable in size and yield than the Atlantic, which though much poorer, is remarkably uniform in its percentage. Quincy rock, as delivered to the mill, yields on an average about 2 per cent. of metallic copper; the Atlantic only 0.75 per cent., which this year fell to 0.60 per cent. Both mines are worked to a great depth; the Quincy's lowest level is 3900 feet on the incline, and the Atlantic's 1600 feet. In both mines the rock is easily broken, but the character, uniformity and width of the Atlantic bed allow it to be extracted more cheaply than the Quincy. The Quincy has to be sorted at the surface, whereas the Atlantic ore goes without handling to the mill. Both the mining and concentrating of the Atlantic ore can, therefore, be carried on at lower cost, and this partially offsets its greater leanness. Yet nothing but most excellent management has brought down the total cost of mining, raising and concentrating the ore; of delivering the concentrate to the furnace and smelting it; and of transporting to market and selling the refined copper to $1.43 per ton of rock treated. That was the figure in 1888. In 1889 it was $1.53. The Quincy has just completed a new and well-equipped three-stamp

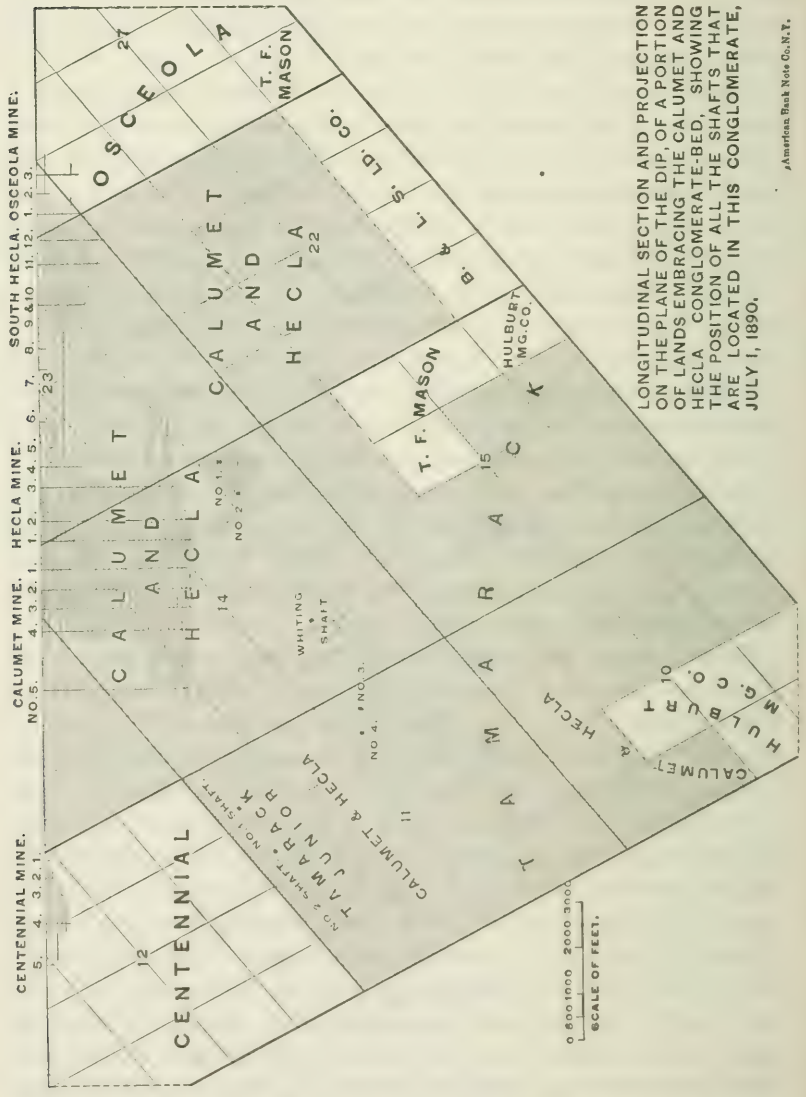
mill with a capacity of 180,000 tons a year, which will enable it to increase its output of copper beyond the 3200 tons limit of 1889. The Atlantic will probably remain steady at about 1800 tons of copper produced from 280,000 tons of ore. I have selected for comparison these two typical amygdaloid mines without disparagement to others, such as the Osceola, which are paying dividends out of rock yielding little more than 1 per cent. of copper.

Conglomerate Mines.—The Albany and Boston forms just such a link between the second and third group of mines as the Copper Falls forms between the first and second groups. After failing to work profitably an amygdaloid-bed, the Albany and Boston made the first experiment in 1864 in working a conglomerate-bed, and thus, though unsuccessful itself, was the forerunner of the most successful and important enterprise on the Lake. The Albany and Boston should therefore appear at the head of the group of conglomerate mines, followed by the Calumet and Hecla in 1865 and by the Allouez, which though organized in 1859, commenced work on the Allouez conglomerate only in 1869. The Osceola appeared in 1873, but after working for a time on so much of the remunerative portion of the Calumet conglomerate as lay within its lines, it transferred its operations to an amygdaloid-bed. The Tamarack, which tapped the Calumet and Hecla conglomerate on the dip of the great ore-chute by a vertical shaft, commenced making copper in 1885, and is the last born of the conglomerate mines.

Several conglomerate-beds alternate throughout the series of the Keweenaw rocks with the unaltered traps and with the copper-bearing amygdaloid. In many places they carry traces of copper, but the only conglomerate-bed that gives assurance of economic value is the Calumet bed where, for about three miles along its strike, it outcrops on Calumet and Hecla ground, on the northern edge of Osceola ground to the south, and on the southern edge of Centennial ground to the north (Fig. 1). In the center of this long stretch Calumet shaft No. 1 was sunk in what has proved to be likewise the center of the most productive area of this vast chute of ore. Hecla shaft No. 1 adjoins it to the south (for the two companies were distinct until amalgamated in 1872). Within the Calumet lines there have been sunk five shafts, from 1 to 5, numbered from south to north, and within the Hecla and South Hecla lines twelve shafts, numbered from north to south. The disastrous fires of late years crippled the deep, central shafts and shut off temporarily all access to the rich ores of the central zone. But previous

to the fire the 700-foot level had been run south to the limit of the Calumet and Hecla property, and had traversed a second chimney of ore in the southern part of the South Hecla, known as the Black

FIG. 1.


LONGITUDINAL SECTION AND PROJECTION ON THE PLANE OF THE DIP, OF A PORTION OF LANDS EMBRACING THE CALUMET AND HECLA CONGLOMERATE-BED, SHOWING THE POSITION OF ALL THE SHAFTS THAT ARE LOCATED IN THIS CONGLOMERATE, JULY 1, 1890.
American Bank Note Co. N.Y.
SOUTH HECLA. OSCEOLA MINE:
HECLA MINE.
CALUMET MINE.
CENTENNIAL MINE.
OSCEOLA
T. F. MASON
C. & H. S. L. CO.
CALUMET AND HECLA
HULBURT MG. CO.
T. F. MASON
BLACK
CALUMET & HECLA
TAMARACK JUNIOR
CENTENNIAL
WHITING SHAFT
NO. 1 SHAFT
NO. 2 SHAFT
NO. 3
NO. 4
HULBURT M. C. O.
SCALE OF FEET.

Hills section of the mine. With extraordinary energy this almost unexplored territory was opened soon after the first fire occurred, and so rapidly was the work of development prosecuted that, although

between the years 1886 and 1887 there was a diminution of product from 22,553 to 20,543 tons of copper, this was due, not to a contraction in the tonnage stamped, but to a decline in the richness of the ore from 4.22 per cent. to 3.52 per cent. A large quantity of lean ore from the Black Hills mine, which was necessarily mixed with the richer ores of the central chute, caused the falling-off. The next year (1888) showed a still further decline to 3.28 per cent. of copper in the rock treated.

The deepest point reached by the Calumet and Hecla is the twenty-ninth level, which is 3750 feet from the surface on the incline of the bed. As the dip is $37\frac{1}{2}°$, it makes the vertical depth 2280 feet. This depth corresponds with the fifth level of the Tamarack mine, but the bottom of No. 4, the deepest Calumet and Hecla shaft, is 4000 feet north of the No. 1 Tamarack shaft. It is therefore assumed that the Tamarack is working on the dip of the South Hecla or Black Hills chimney, and not on the central body. The sinking of the Tamarack vertical shaft to a depth of 2270 feet on the chance of striking the Calumet conglomerate, was regarded from a mining standpoint as an act of audacity, for when it was begun in 1882, the lower drifts of the Calumet and Hecla were far above the level at which the Tamarack shaft was expected to pierce the lode. But ere the goal had been reached, in 1885, the Calumet's deepest workings were approaching the same level; so that it became a foregone conclusion that the Tamarack hopes would be realized. To-day, the Calumet and Hecla has nearly re-equipped the burnt-out portion of its mine; it has also opened up a virtually new mine in the South Hecla; it has nearly reached its fortieth level, or a depth of 4000 feet by its inclined shaft No. 4, and it is sinking a vertical shaft—the Whiting—which in less than two years will touch the central chute of ore at the vertical depth of 3400 feet, or on the sixtieth level. Meanwhile, the Tamarack is sinking a third and a fourth shaft, which should both intersect the central chute of the conglomerate at a depth of about 5000 feet, and Tamarack Junior, a company affiliated in ownership with the Tamarack, is sinking to the north of both the Calumet and Hecla and the Tamarack ground, in the hope of finding good ore below the Centennial property, a hope stimulated by the recent success of that latter company in striking remunerative ground to the north of its older workings.

Within this limited range of three miles, besides the twenty-five inclined shafts of which eight are deeper than 3000 feet, there are sunk, or being sunk, seven vertical shafts, the shallowest of which

is 2270 feet, and the deepest of which will be 5000 feet. By the end of this year, the Calumet and Hecla will be daily extracting from this limited area 3000 tons of ore, and the Tamarack 1000 tons; but these quantities, great as they are, could certainly be more than doubled within a few years, if it should be the policy of these companies to increase their production. And their neighbors, the Osceola, Centennial, Kearsarge, and Tamarack Junior, will contribute in no small degree to swell the grand total. During the next generation, these nine square miles will probably produce more copper than any other equal area on the earth's crust.

It is not within the scope of my present purpose to describe the methods of mining or of concentrating and smelting the ore; suffice it to say, that the ore, though in some cases so very lean, is readily concentrated, by automatic means, into a product carrying from thirty to ninety per cent. of metallic copper, and that this concentrate is smelted and refined at one fusion. Until lately, the smelting was done for the mining-companies by a separate smelting company, under contract, at so much per ton of copper. Now, the Calumet and Hecla company has its own smelting-plant, and as one consequence, it reduces the loss of copper in its tailings by making concentrates of a lower grade.

Arizona.

Over ninety per cent. of all Arizona copper is derived from oxidized ores, which are found in the southern and middle sections of the Territory. Nature has reduced the copper in the Keweenaw rocks of Michigan to metal. In southern Arizona she has assisted the metallurgist by effecting a partial alteration of the ores to oxides and carbonates. In both cases the action has been attended with the elimination of certain elements injurious to copper, which are almost invariably associated with the sulphuretted ores of the metal, and are difficult to extract by the usual smelting methods. Hence, copper made from native Lake ore ranks highest in quality in the markets of the world, and that made from the oxidized ores of Arizona holds an intermediate place between Lake copper and the metal made by the ordinary methods from sulphuretted ores. It can be drawn and rolled, and stamped into intricate shapes, like Lake copper, but it lacks the high electrical conductivity of that exceptionally valuable metal. It possesses the qualities of the copper from the Burra-Burra and Moonta mines of South Australia, which is likewise the product of oxidized ores.

This ore is smelted in water-jacketed cupola-furnaces, by a single fusion, into bars of ninety-six per cent. standard, which are sold by the pound under conditions similar to those which govern the sale of Chili bars.

With insignificant exceptions, all bar-copper comes from the three districts of Clifton, Bisbee and Globe. The principal mines in the Clifton district are owned by the Arizona Copper Company, a Scotch organization, and by the Detroit Copper Company. There are several groups of mines, some miles apart, the members of each group being related to one another, and forming a more or less continuous chain of deposits. The first property to be worked was the Longfellow. Though not itself the most productive mine at present, it lies in the strike of the most important series of ore-masses. At Bisbee, and for miles around it, there are claims capable of producing more or less copper, but no other large ore-deposit has been discovered than that which was opened as the Copper Queen mine in 1880. At Globe, the only large producer has been the old Globe mine, owned by the Old Dominion Copper Company.

The ore occurs in all three districts in, or adjacent to, carboniferous limestone, which has been chemically and mechanically influential in assisting the oxidation of the ore to a very considerable depth; for, through the crevices which intersect the limestone, and which have, in part, been the result of the ore-decay itself, water has filtered down from the surface and decomposed the ore to a depth far below the line of decay of the adjacent feldspathic rocks, where these rocks are not themselves heavily charged with copper. In Bisbee, the ore-beds appear to be confined to the limestone, but follow no regular order in their distribution. In Clifton and Globe, on the other hand, the ore-bodies, though of irregular size and occurring at irregular intervals, are generally found in the plane of contact between the limestone and granite, or the limestone and sandstone. Sometimes, however, and then over considerable areas, the granite and sandstone themselves are replaced by copper and associated ores. But even in these cases, the contiguous limestone has apparently played an essential part in the genesis of the oxidized ore. Sulphuretted ores of copper are found in all of our extensively worked carbonate mines—sometimes in large masses, which for some cause, not always assignable, have escaped decay—occasionally even at a much higher level than that at which oxidized ores occur in the same mine. The average percentage of copper in the ore is difficult to determine, since unassorted ore is never delivered to

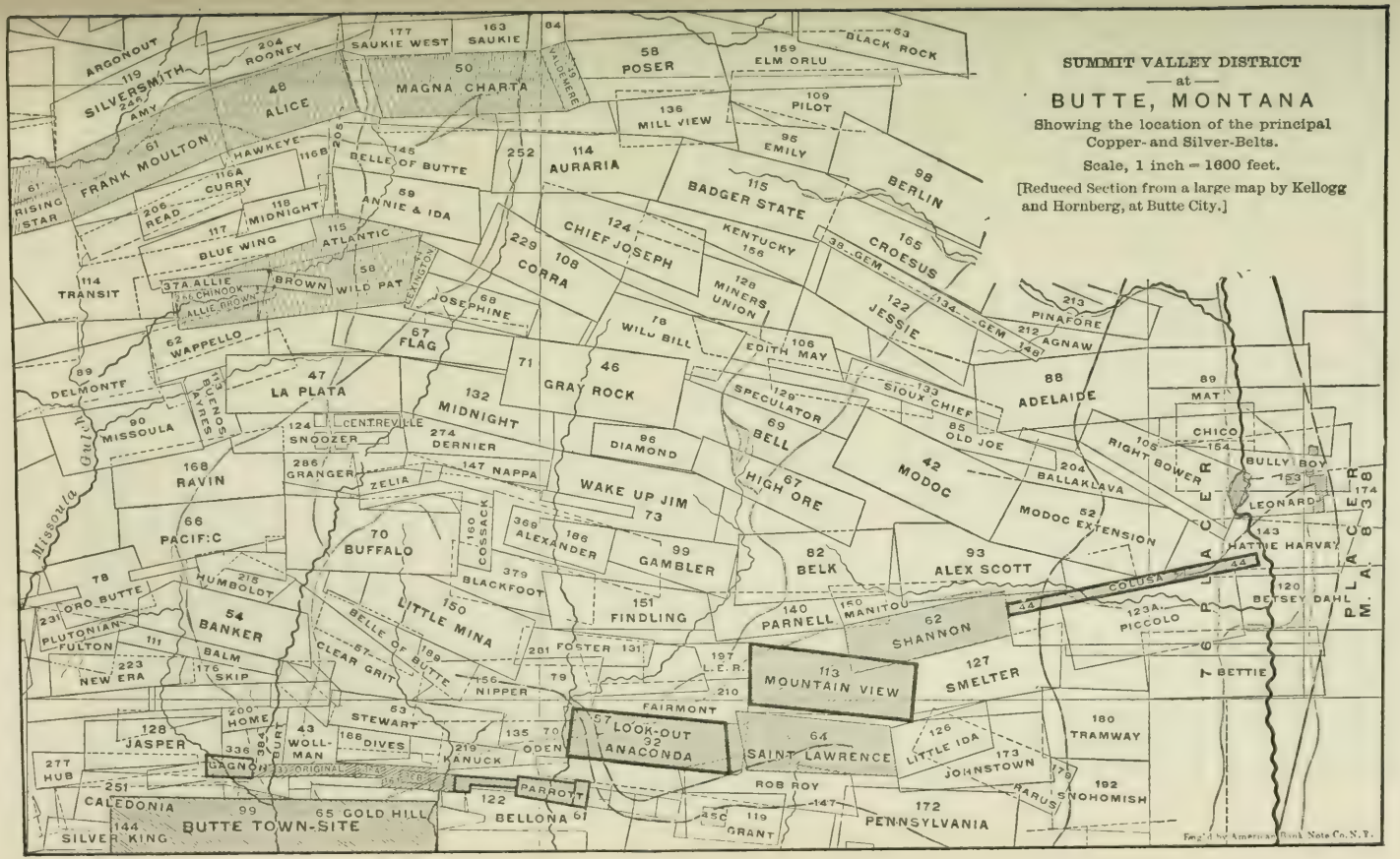
the furnaces, while the grade to which it is selected is dependent in each district upon the cost of fuel and transportation. The furnace-yield of Copper Queen wet ore is about eight per cent., but almost as much very lean ore is stowed away in the stopes as is delivered to the furnace bins. The average ores of the other carbonate districts are probably richer and their furnace-yield is notably higher. The deeper Queen ores consist essentially of ferric oxide associated with cuprous oxide and cupric carbonate, resulting from the oxidation of iron- and copper-pyrites. They are consequently basic. But sufficient siliceous ore can be procured to supply the necessary acid flux for the furnace-mixture. In the other districts, on the contrary, where the ore-gangue consists largely of altered granites and sandstones, the siliceous and aluminous constituents have been imperfectly eliminated, and barren limestone must be added to the furnace-charge. But in none of the large producing mines does the average of the ore reach the high percentage promised to investors in the prospectuses of undeveloped southwestern mines.

It will be seen from this sketch that the Arizona ore-beds are as strikingly irregular in their distribution as those of the Lake are the reverse. The map of a Lake mine is like a chess-board, laid out with mathematical precision; that of the deep Arizona mines is more like a tangle of different colored threads.

The only large copper-mine in the northern half of the territory is the Verde, in Yavapai county. There the copper occurs in lenticular masses imbedded in crystalline schists, probably of Taconic age. Massive carbonates are found at the surface, but at a shallow depth occur altered sulphurets or oxisulphurets which resemble those of the Southern States and of Butte. In this mine, as in all the copper-bearing veins in this section of the territory, notable quantities of precious metals accompany the copper; but the gold and silver are irregularly distributed in the same vein and bear no uniformly definite proportion to the copper.

Montana.

Of far greater moment than the copper-mines of Arizona are those of Butte, Montana. The principal mines have been opened on a lode which has proved to be continuous and productive for over three miles. The principal mines succeed one another from west to east as follows (Fig. 2): Gagnon, Original, Parrott, Anaconda, St. Lawrence, Mountain View, Shannon, Colusa and Hattie Harvey. The gangue of the vein is granitic and softened to a very considerable depth.


SUMMIT VALLEY DISTRICT
— at —
BUTTE, MONTANA
Showing the location of the principal Copper- and Silver-Belts.
Scale, 1 inch = 1600 feet.
[Reduced Section from a large map by Kellogg and Hornberg, at Butte City.]
ARGONOUT
SILVERSMITH
ROONEY
SAUKIE WEST
SAUKIE
MAGNA CHARTA
POSER
ELM ORLU
BLACK ROCK
ALICE
FRANK MOULTON
HAWKEYE
BELLE OF BUTTE
AURARIA
MILL VIEW
PILOT
EMILY
BERLIN
RISING STAR
CURRY
READ
MIDNIGHT
ANNIE & IDA
BLUE WING
ATLANTIC
BADGER STATE
KENTUCKY
CROESUS
GEM
TRANSIT
ALLIE
CHINOOK
BROWN
WILD PAT
CHIEF JOSEPH
CORRA
MINERS UNION
JESSIE
PINAFORE
AGNAW
JOSEPHINE
WAPPELLO
FLAG
WILD BILL
EDITH MAY
DELMONTE
LA PLATA
GRAY ROCK
SPECULATOR
SIOUX CHIEF
ADELAIDE
MISSOULA
CENTREVILLE
SNOOZER
MIDNIGHT
DERNIER
DIAMOND
BELL
OLD JOE
MODOC
RIGHT BOWER
CHICO
BULLY BOY
RAVIN
GRANGER
ZELIA
NAPPA
WAKE UP JIM
HIGH ORE
BALLAKLAVA
LEONARD
PACIFIC
BUFFALO
COSSACK
ALEXANDER
GAMBLER
MODOC EXTENSION
HATTIE HARVEY
ORO BUTTE
HUMBOLDT
BLACKFOOT
BELK
ALEX SCOTT
COLUSA
BETSEY DAHL
PLUTONIAN
FULTON
BANKER
LITTLE MINA
FINDLING
PARNELL
MANITOU
SHANNON
PICCOLO
BELLE OF BUTTE
CLEAR GRIT
FOSTER
NEW ERA
SKIP
NIPPER
MOUNTAIN VIEW
SMELTER
BETTIE
FAIRMONT
JASPER
HOME
STEWART
WOLLMAN
DIVES
LOOK-OUT
ANACONDA
SAINT LAWRENCE
LITTLE IDA
TRAMWAY
HUB
GAGNON
ORIGINAL
KANUCK
ODEN
JOHNSTOWN
CALEDONIA
GOLD HILL
PARROTT
ROB ROY
SNOHOMISH
BUTTE TOWN-SITE
BELLONA
GRANT
PENNSYLVANIA
SILVER KING
Engd. by American Bank Note Co. N.Y.

It contains disseminated particles of ore, but most of the ore is derived from large imbedded masses. One of these in the Anaconda is said to attain a width of 150 feet of solid mineral. Two, or perhaps three, parallel veins, or as many chains of ore-masses in one very wide vein are traceable throughout its extent. Everywhere the copper carries silver in proportions varying from two ounces per unit of copper to less than one-half ounce per unit. The proportion of silver to copper is greater in the western section of the vein than in its central and eastern sections. The ores of the Gagnon mine are smelted into rich argentiferous matte at the Williams works, which ship their product to the Boston and Colorado Smelting Company of Argo, Colorado. The Parrott ores contain about sixty ounces of silver to the ton of copper. The average silver-contents of the Anaconda, Mountain View and Colusa ores is less. Both the Chambers Syndicate and the Boston and Montana Company, however, make rich silver-matte, but not from unmixed ores of the great lode.

In following the lode from the Gagnon eastward, one ascends the western slope of a steep hill, traversing the Parrott and the Anaconda to the Mountain View, which covers the summit. In descending the eastern slope of the hill, one crosses the Shannon and the West and East Colusa. The latter is situated in the trough of a valley and was formerly supposed to cover the eastern limit of the lode. But productive ground has recently been found far to the east of this point, extending the length of this vast lode a mile beyond its formerly supposed confines. The mines which lie at the extremities of the most productive section of the lode, viz.: the Gagnon and Original on the west and the Colusa on the east, are likewise lowest in level. In them, the copper-ore came to the surface, whereas in the Anaconda and the Mountain View, which crown the hill several hundred feet above the terminal mines, the surface-ores to a depth of four hundred feet carry some silver but no notable quantity of copper. At about that depth, however, great bodies of oxisulphurets and erubescite were met with in the Anaconda, from which, in the early days, ores running fifty per cent., and over, in copper were mined by tens of thousands of tons. Subsequently, the Mountain View commenced to yield from about the same depth the rich decomposed ores which are now assisting the large product of the Boston and Montana Copper Company. The same oxisulphurets come to the surface, or are found near the surface, where the lode crops out in the valleys and on the flanks of the hill. But their depth and richness

there are not as great as they are on the summit, where it seems as if the copper, leached out of the four hundred feet of depleted vein, had been concentrated in the underlying ore, and had thus produced a zone of secondary ore about two hundred feet deep, which contains, as might be expected, about thrice its normal copper-contents. In the decuprified surface-ore, silver was retained as an insoluble chloride when the soluble copper was removed. Therefore we find that the proportion of silver to copper is less in the enriched copper zone, where an excess of copper is added to the normal silver contents, than in the unaltered ore beneath. Hence, the proportion of silver to copper increases with the depth. The Anaconda was bought as a silver-mine, and preparations were being made to mill the ore when the shaft entered the copper. The experience of the Parrott and other mines confirms the theoretical anticipation that the lean unaltered ores carry a larger proportion of silver to copper than the overlying altered ores. No massive carbonates have been discovered, but the Gagnon and Parrott in their early days yielded small quantities of an earthy carbonate of copper, rich in silver. The altered ores of both the great lode and the smaller cross- and parallel-lodes consist mainly of a compact purple ore and of soft black ore with a bright metallic streak.

The actual copper-yield of the Boston and Montana is given in the report of the company for the year ending June 30, 1890, at 9.36 per cent. This high percentage is the result of treating a very large proportion of the rich altered ore. Distributing the published yield of the Anaconda and the Chambers Syndicate mines over the number of pounds said to have been transported from the mine to the smelting-works—namely, 61,647,000 pounds of copper over 1,028,000,000 pounds of ore—we obtain, as the yield of the Anaconda, whose reserves of altered ore are not yet exhausted, 5.9 per cent. The ultimate yield of the deep copper-ores of the great Butte lode and its subordinate veins will undoubtedly be lower still. Many small copper-mines are being worked on the parallel veins, and on what appear to be spurs of the great lode. The operators are the Chambers Syndicate, a company closely connected with the Anaconda; the Boston and Montana Company; the Butte and Boston Company and a number of individual owners. Some of the ore-bodies of these subordinate lodes are large and their ores generally carry a profitable percentage of silver. In the aggregate they add largely to the present production of Butte, and, considering the great length of ground which they represent, they insure for Butte

a longer life than would be its lot if all the companies were making their extravagant drafts on the one great lode alone.

Both Arizona and Montana have increased their production from year to year, but in very different proportions, as the following figures, from *The Mineral Resources of the United States*, show:

	Arizona. lbs.	Montana. lbs.
1882,	17,984,415	9,058,284
1883,	23,874,963	24,664,346
1884,	26,734,345	43,093,054
1885,	22,706,366	67,797,864
1886,	15,657,035	57,611,621
1887,	17,720,462	78,699,677
1888,	31,797,300	97,897,968
1889,*	31,600,000	104,200,000

Conclusion.

When we turn from a review of our present sources of production to the consideration of our ultimate resources, we wander at times from the domain of fact into the borderland of fancy.

In the Appalachian chain from Vermont to Georgia there are imbedded in the crystalline schists large masses of pyrites, some consisting of ordinary bisulphide of iron, but most of them of pyrrhotite, and all carrying more or less copper. In the North they are less conspicuous than in the South, where, in the absence of denudation, rock-decay extends to a considerable depth, and the existence of the ore-masses is indicated by an oxidized iron cap. In Vermont, as already mentioned, the Ely mine was an important producer between 1854 and 1882. In Maine, copper-bearing pyrites-beds have been worked in two widely separated districts. New Hampshire has always produced a little copper. The Ore Knob mine in North Carolina yielded largely between 1873 and 1882. In Tennessee, the Ducktown mines produced annually more copper than any single Lake mine between 1852 and the advent of the Calumet and Hecla in 1866. Like all sulphuret mines, they became poorer as depth was attained. The Tallapoosa mine, in Georgia, was worked for several years, shipping its pyrites to phosphate-works in Atlanta, Ga., where the copper was extracted from the cinder in the wet way. The Stone Hill mine in Alabama yielded a much higher grade of copper-ore. There are very large masses of pyrrhotite in Carroll county, Virginia, which, despite its

* The figures for 1889 are from the *Engineering and Mining Journal.*

inferior sulphur-contents, will sooner or later be used for acid-making, and then made to yield up their very small percentage of copper. Considerable quantities of bisulphide of iron are mined near Charlemont, Mass., and in Louisa county, Va., but neither deposit carries any appreciable amount of copper. The Liberty and other mines in Maryland, as well as the copper-bearing slates and sandstones of Pennsylvania and New Jersey, are lying idle. At the present moment, therefore, the Atlantic States yield virtually no copper, and there is no reason to suppose that they will ever become large producers, inasmuch as they contain no bodies of pyrites comparable in size or in composition to even the smaller Iberian ore-masses.

Turning to the Lake region, the case is very different. The larger of the existing mines will, doubtlessly, extend their capacity for production, and new mines will probably be opened more rapidly than the old ones will decay. There is but one mine in the country whose actual contents can be even guessed at. I refer, of course, to the Calumet and Hecla. Its ground has been explored laterally almost to its limits, and by its deepest workings, as well as by the exploration of the Tamarack Company, the value of its principal block of ground on the dip has been determined. Of course, there may be large tracts of unproductive vein within the area thus partially explored, which may, to some extent, impugn Mr. Agassiz's assumption that there exists within the Calumet and Hecla territory sufficient copper to last for forty years at twice the present rate of production. At what rate the administration may see fit to extract that definite quantity, depends simply on the use which they wish to make of money, and of the appliances which money can furnish for extracting, transporting and refining their ores. The Tamarack has no known means of ascertaining to what depth this same rich chute extends, but it is reasonable to assume that the depth of the ore-bodies included in the Keweenaw rocks bears some relation to their length, and is, therefore, not illimitable. Outside of the Calumet and Hecla ore-chute, the Quincy is the only mine whose ore is rich enough sensibly to affect future production in case the administration should deem it wise to increase the capacity of its new mill. But the Keweenaw rocks extend beyond Michigan through the northwestern part of Wisconsin into Minnesota, where I have seen them carrying small quantities of metallic copper, associated, strange to say, with sulphides of iron and copper. Metallic copper has been mined, though unprofitably, in Douglas

county, Wisconsin, and salable quantities of copper have been found as float in the drift in Wisconsin. But the outcrop of the Keweenaw rocks, both in Wisconsin and Minnesota, is generally so heavily covered with forest and swamp, that the discovery of copper in those regions will obviously be slow. In Michigan itself there is an almost boundless area of very lean amygdaloid- and conglomerate-rock, which will yield about one-half per cent. of copper, and will therefore attract mining only when a high range of prices is assured, for the starting of a Lake mine and the erection of the plant necessary to the economical treatment of the ore is a costly enterprise.

Thus we may count on a gradual but considerable increase of copper from the larger existing Lake mines. It is also reasonable to suppose that new mines will be opened, and that if the price be tempting, abandoned mines will be revived. At present, the price of the metal determines the activity of most of the smaller mines. During the late depression the cost of production in many cases hovered dangerously near the figures at which copper was sold.

Beyond the Lakes lies the West—a wide and but partially explored domain. Looking at the existing mines, it is hardly probable that those of Arizona will greatly extend their production, though in none is there any sign of exhaustion. They do not grow richer with growing depth, and it is safe to presume that when the oxidized ores, including the oxisulphurets, give place to unaltered sulphurets, the latter will be leaner than the superjacent layers. In Butte the extensive surface-ground owned by the Chambers Syndicate, and the Boston and Montana, and Butte and Boston companies probably covers undeveloped bodies of rich surface-ores. When the ravages of the fire in the Anaconda have been repaired, she will resume her former production; and the new work of the Boston and Montana Company at Great Falls, Missouri, will enable that company to increase its output. For a time, therefore, we may look for augmented production from Butte. But sooner or later even her output will decline by reason of the notable debasement which the deeper Butte ores undergo in common with the sulphuret of all copper-veins. If a given standard of production is to be maintained, a corresponding expansion of plant will be required, and this in time will create unwieldy and unprofitable establishments. The resources of Butte are enormous, but they are drawn upon with lavish prodigality, and like everything else on the earth, even including copper-mines, they are not inexhaustible.

So much for the principal centers of active mining. As for the undeveloped riches of our West, it is to be borne in mind that although this wide expense of territory has not been thoroughly explored, there are few districts which have not been superficially prospected. For forty years a large army of active and sanguine men has been marching back and forth over every obstacle, and in every season, traversing the Rocky Mountains from east to west, and from north to south, in search of hidden treasure. That their search long ago was very thorough is being revealed ever-more clearly by the rarity of the occasions on which fresh discoveries of large bodies of any metalliferous mineral are made at the present day. And copper is a metal whose ores betray their presence by such characteristic and brilliant colors that they are not easily overlooked, especially in a country whose naked mountain-ranges are riven and deeply scored by the elements. Some new discoveries of large deposits are sure to be made, but they will be infrequent. It must be remembered that ten years have elapsed since the railroads gave access to Montana and Arizona, and led to the active development of the large mines in those sections that had been already discovered; and that despite the construction in the interval, of other through lines and ramifying branches, no large deposit that was then unknown has been opened since. But while the copper world is not destined to be startled often by the apparition of an Anaconda mine, we have no cause to fear a dearth of copper even when the large mines now known have been exhausted. In every State and Territory of the Rocky Mountains we know of an abundance of low-grade ores which will help us to maintain our supremacy in the ranks of producers when they become available by the growth of population, the cheapening of fuel and labor, and the application of still better metallurgical methods.

There extends through northwestern Texas, northern New Mexico and northern Arizona a belt of Permian sandstone carrying nodules of rich ore and immense quantities of low-grade ore, which at some future day will be utilized. The San Pedro mine in that section of New Mexico has figured at intervals as an important producer. In the Santa Rita district of southern New Mexico large deposits of ore were once extensively worked. In the Clifton district, as indeed throughout extensive tracts of southwestern Arizona, the rocks are tinted green with copper; but the copper is associated with so much alumina and silica, and is so lean, that to-day it cannot be profitably treated. Nor are these the only undeveloped deposits of Arizona which simply await favorable conditions for exploitation. In the

State of California large beds of low-grade, sulphuret ores occur in Calaveras and Nevada counties, which beside promising a valuable yield of copper, will prove a most desirable auxiliary to California's future chemical industry. Passing to Nevada there are large deposits of low-grade ores in Esmeralda county, and ores of higher grade are reported to exist north of the line of the Central Pacific Railroad. Utah now contributes some copper, and Idaho will undoubtedly contribute much more than at present. She has copper-deposits on the Lost river, and still more extensive deposits in the Seven Devils district on the Snake river, which if nearer the railroad, would now be furnishing their quotum to the common stock. Oregon has unworked copper-mines. The copper of Montana is not confined to Butte, nor are the States of Wyoming and Colorado barren. Taking into account the wide area over which we find copper, and the casualties of mining, which if they generally bring misfortune, yet occasionally bless us with unexpected luck, we may allay all alarm of famine while dispelling any apprehension of a deluge. We should bear in mind that the great bulk of our undeveloped copper cannot be made available without a favorable price, and yet we believe that only by drawing on these reserves can our own and the world's future requirements be met. A greatly increased production, either from existing mines or from new ones, will be necessary to supply the world's ravenous demand for copper—a demand that springs from many new applications of the metal as well as from increased consumption for old uses.

The following table exhibits the rate of copper-consumption in the United States since 1850:

Year.	Population.	Consumption in Tons of 2000 lbs.	Consumption per Head.
1850	23,191,876	6,710	0.550 lbs.
1860	31,443,321	7,116	0.405 "
1870	38,558,371	12,342	0.603 "
1880	50,155,703	26,799	1.006 "
1889	61,000,000	34,800	2.736 "

The *per capita* decline between 1850 and 1860 is probably attributable in part to the introduction of iron in ship-building and the consequent abandonment of copper-sheathing, and partly to the financial disturbance which reached its extreme depression toward

the close of the decade. Electricity may be credited with creating an extraordinary demand within the last few years, but it had no part in raising the *per capita* consumption so steadily between 1860 and 1880. This was due to the growing wealth of the people which stimulated the demand for fine metal-work in architectural construction and decoration. A *pro rata* increase of consumption during the coming decade may not be maintained, though no good reason for a falling off can be alleged. The present applications of electricity are few compared with its future uses, and although we may expect to see copper economized by the discovery of new methods for reducing the size of conductors, there is no probability that any other metal will supplant copper in the construction of dynamos or in the distribution of electrical energy. The necessity for sheathing iron vessels is most keenly felt now that speed is becoming the supreme consideration both in mercantile and naval marines, and we may therefore almost count upon the discovery of a successful method for effecting it. Such a discovery would naturally exert a potent influence on the balance between supply and demand. For ordinary household-purposes copper will certainly hold its own, for even though aluminum should be produced as cheaply as copper, it would not displace bronze and brass, with their capacity for artistic treatment in color and form. If the growth of our population during the next ten years continues at the same rate as for the past decade, and the *per capita* increase of consumption is maintained, it will require in the year 1900 over 300,000 short tons of copper to satisfy the demands of this country alone, or about seventy-five per cent. of the world's present annual production.

The importance of the copper interests of the country as industrial enterprises is much greater than is ordinarily suppposed. Without pretending that they rival the iron- and steel-industries as a national necessity, they bear no insignificant comparison with them. If we assume that the average of all copper-ores treated carries $3\frac{1}{2}$ per cent. of metal (and this is above rather than below the true average), and that our production of metallic copper in 1889 was 241,600,000 pounds, this would represent the mining of 3,451,000 tons of copper-ore, as against an annual production of 12,000,000 tons of iron-ore, or about 28 per cent. The metallurgical treatment of copper, and more especially of the sulphuretted ores, involves heavier costs in labor, fuel and material than the treatment of iron, and most of the subsequent manipulation of the metal in converting it into manufactured forms is more intricate. The comparative importance of the

two metals in the national balance-sheet, therefore, is not to be estimated so much by the respective weight of the finished products as by the relative amount of raw material and labor used in creating them.

STATISTICS.

TABLE I.—*Copper-Product of the United States from* 1845 *to* 1890. *Showing Relative Proportion of the Lake Superior Product.*

[From *Mineral Resources of the United States*, for 1888, published by the U.S. Geological Survey.]

Year.	Total Production.	Lake Superior.	Calumet and Hecla.	Percentage of L. Superior of total product.	Year.	Total Production.	Lake Superior.	Calumet and Hecla.	Percentage of L. Superior of total product.
	Long Tons.	Long Tons.	Long Tons.			Long Tons.	Long Tons.	Long Tons.	
1845	100	12	.....	12.0	1868	11,600	9,346	2,276	80.6
1846	150	26	.....	17.0	1869	12,500	11,886	5,497	95.1
1847	300	213	.....	71.0	1870	12,600	10,992	6,277	87.2
1848	500	461	.....	92.5	1871	13,000	11,942	7,242	91.9
1849	700	672	.....	96.0	1872	12,500	10,961	7,215	95.7
1850	650	572	.....	88.0	1873	15,500	13,433	8,414	87.3
1851	900	779	.....	86.6	1874	17,500	15,327	8,984	87.6
1852	1,100	792	.....	72.0	1875	18,000	16,089	9,586	89.4
1853	2,000	1,297	.....	64.9	1876	19,000	17,085	9,683	88.9
1854	2,250	1,819	.....	71.1	1877	21,000	17,422	10,075	82.9
1855	3,000	2,593	.....	86.4	1878	21,500	17,719	11,272	82.4
1856	4,000	3,666	.....	91.6	1879	23,000	19,129	11,728	83.2
1857	4,800	4,255	.....	88.7	1880	27,000	22,204	14,140	82.2
1858	5,500	4,088	.....	74.3	1881	32,000	24,363	14,000	76.1
1859	6,300	3,985	.....	63.3	1882	40,467	25,439	14,309	62.1
1860	7,200	5,388	.....	74.8	1883	51,574	26,653	14,788	50.1
1861	7,500	6,713	.....	89.1	1884	63,555	30,916	17,812	48.4
1862	9,000	6,065	.....	67.4	1885	74,053	32,206	21,093	43.5
1863	8,500	5,797	.....	67.0	1886	69,971	35,666	25,259	50.1
1864	8,000	5,576	.....	69.7	1887	80,768	33,693	20,543	41.7
1865	8,500	6,410	.....	75.4	1888	103,245	38,604	22,453	38.2
1866	8,900	6,138	.....	68.8	*1889	106,507	38,679	21,714	35.8
1867	10,000	7,824	603	78.2	*1890	101,861	44,451	26,539	38.2

* From *Engineering and Mining Journal.*

TABLE II.—*Copper-Product of the United States, by States and Territories.*

[Compiled from *Engineering and Mining Journal.*]

	1882.	1883.	1884.	1885.	1886.	1887.	1888.	1889.	1890.	Percentage of Total Product, 1890.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	
Michigan (Lake Superior)	57,155,991	59,702,404	69,353,202	72,147,880	79,918,460	76,028,697	86,587,424	87,504,103	99,570,000	38.30
Arizona	17,984,415	23,874,963	26,734,345	22,706,366	16,000,000	17,790,000	33,200,000	32,933,000	35,720,000	13.32
Montana	9,058,284	24,664,346	43,093,054	67,798,864	57,611,485	78,699,677	98,500,000	104,539,353	111,010,000	42.67
New Mexico	869,489	823,511	59,450	79,839	558,385	283,664	1,631,271	3,400,000	870,000	.40
California	826,695	1,600,862	876,166	496,028	430,210	1,600,000	1,570,021	1,700,000	2,140,000	.84
Colorado	1,494,000	1,152,652	2,013,125	1,146,460	409,306	2,012,027	1,621,100	3,100,000	6,760,000	2.66
Utah	605,880	341,885	265,526	166,199	500,000	2,500,000	2,131,047	2,400,000	600,000	.23
Wyoming	100,000	962,468	.....	.....	.....	.....				
Nevada	350,000	288,077	100,000	8,871	50,000	.....				
Idaho	.....	.....	46,667	40,381	.....	.....				
Middle States	294,695	324,706	232,114	190,000	.....	.....	3,240,725	3,000,000	3,900,000	1.56
New England States	1,555,000	612,124	904,423	211,602	315,719	200,000				
Southern States	400,000	395,175	317,711	40,199	29,811	.....				
Lead refiners	125,000	782,880	950,870	910,144	1,282,496	2,432,801				
Domestic product	90,819,449	115,526,653	144,946,653	165,942,842	157,105,872	181,546,869	228,481,588	238,576,456	260,570,000	99.98
Imported ores	1,000,000	1,625,742	2,858,754	6,056,841	4,500,000	3,750,000	5,000,000	5,100,000	6,100,000	
Total product	91,819,449	117,151,795	147,805,407	171,999,683	161,605,872	185,296,869	233,481,588	243,676,456	266,670,000	

TABLE III.—*Copper-Product of Lake Superior.*

[Compiled from *Engineering and Mining Journal.*]

Name of Mine.	1882.	1883.	1884.	1885.	1886.	1887.	1888.	1889.	1890.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.
Calumet and Hecla	32,053,539	33,125,045	40,473,585	47,247,990	50,518,222	46,016,123	50,295,720	48,640,029	59,000,000
Quincy	5,665,796	6,012,259	5,650,436	5,848,530	5,888,517	5,603,691	6,367,809	6,405,686	8,050,000
Osceola	4,176,782	4,256,409	4,247,630	1,945,208	3,560,786	3,574,972	4,134,320	4,534,127	5,196,950
Franklin	3,264,120	3,488,708	3,748,652	4,007,105	4,264,297	3,915,838	3,655,751	4,336,000	5,368,506
Allouez	1,683,557	1,751,377	1,928,171	2,170,476	1,725,463	885,010	314,198	1,762,816	1,447,396
Atlantic	2,631,708	2,682,197	3,163,585	3,582,633	3,503,670	3,641,865	3,974,972	3,698,837	3,644,403
Pewabic	1,482,666	1,171,847	227,834	.....	.....	.....	.....	.....	.....
Central	1,353,597	1,268,506	1,446,747	2,157,408	2,512,886	2,199,133	1,817,023	1,270,592	1,366,011
Grand Portage	757,080	735,598	255,860	.....	.....	.....	.....	.....	.....
Conglomerate	734,249	222,117	1,198,691	.....	.....	.....	.....	.....	.....
Mass	737,440	659,474	481,396	663,500	247,179	.....	71,090	76,775	61,500
Copper Falls	587,300	804,000	891,168	1,150,538	1,378,679	719,150	1,199,950	720,000	650,000
Phœnix	537,177	512,291	631,004	341,355	1,101,804	11,000	.....	.....	.....
Hancock	540,575	484,906	562,636	203,037	150,000	.....	.....	.....	.....
Huron	364,579	720,213	1,927,660	2,271,163	1,992,695	1,881,760	2,370,857	2,215,000	1,758,748
Ridge	102,936	60,155	74,030	63,390	158,272	84,902	50,924	31,969	21,569
St. Clair	87,126	125,225	139,407	.....	.....	.....	.....	.....	.....
Cliff	66,053	10,374	28,225	.....	22,342	.....	.....	.....	.....
Wolverine	25,623	699,622	751,763	328,610	3,125	2,330	.....	.....	.....
Nonesuch	46,450	.....	23,867	28,484	.....	.....	.....	.....	.....
Isle Royal	35,447	.....	16,074	.....	.....	.....	.....	.....	.....
Minong	21,580	3,582	.....	.....	.....	.....	.....	.....	.....
National	17,060	26,006	87,368	162,252	184,706	25,187	.....	495,000	123,879
Minnesota	10,672	6,226	1,144	12,608	.....	.....	.....	.....	.....
Belt	5,625	16,402	130,851	27,433	7,300	.....	.....	.....	.....
Sheldon and Columbia	3,299	.....	9,828	.....	.....	.....	.....	.....	.....
Knowlton	.....	.....	.....	.....	.....	.....	3,300	2,000	20,000
Adventure	429	.....	4,333	4,000	1,000	.....	.....	692	15,485
Peninsula	.....	849,400	1,225,981	.....	.....	.....	.....	736,507	1,106,529
Tamarack	.....	7,435	.....	181,669	3,646,517	7,396,529	11,411,325	10,605,451	10,091,064
Ogima	4,207	3,000	1,106	12,000	.....	.....	.....	.....	.....
Kearsarge	.....	.....	.....	.....	.....	21,237	829,185	1,918,849	1,616,556
Evergreen Bluff	.....	.....	954	1,500	1,000	.....	41,000	28,773	25,000
Centennial	83,534	.....	.....	.....	.....	.....	.....	.....	.....
Other mines and tributers	75,765	.....	23,213	31,000	50,000	50,000	50,000	25,000	36,491
Total	57,155,991	59,702,404	69,353,202	72,447,889	80,918,460	75,928,727	86,587,424	87,504,103	99,570,000

TABLE IV.—*Copper-Product of Arizona.*

[Compiled from *Engineering and Mining Journal.*]

Mine.	1884.	1885.	1886.	1887.	1888.	1889.	1890.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.
Copper Queen	7,700,000	6,721,535	3,800,000	5,945,550	9,379,949	9,408,000	9,408,000
Holbrook & Cave	.....	.....	.....	.....	3,042,468	2,561,144	3,047,310
Old Dominion	7,400,000	4,688,640	4,567,665	1,441,770	4,870,000	5,923,289	7,782,190
Arizona Copper	3,760,000	6,832,880	5,250,000	5,714,000	7,133,188	7,600,000	4,662,281
Detroit	2,940,000	3,456,000	2,135,000	4,404,321	5,420,224	5,076,890	4,976,890
United Verde	3,680,000	....	.....	272,124	3,200,000	1,923,738	5,675,602
Other mines	1,254,345	1,007,301	247,335	12,235	154,171	440,000	167,727
Total	26,734,345	22,706,356	16,000,000	17,790,000	33,200,000	32,933,061	35,720,000

TABLE V.—*Copper-Product of Montana.*

[Compiled from *Engineering and Mining Journal.*]

Mine.	1884.	1885.	1886.	1887.	1888.	1889.	1890.
	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.	Pounds.
Anaconda	23,000,000	36,000,000	33,267,864	57,000,000	63,245,473	61,647,000	64,046,812
Parrott	9,300,000	9,809,000	10,000,000	10,000,000	10,750,000	9,500,000	9,000,000
Boston & Mont.	6,600,000	7,500,000	2,000,000	1,500,000	18,278,667	26,425,228	26,822,804
Clark's Colusa	600,000	10,000,000	7,000,000	7,100,000	700,000		
Butte Red. Wks.	2,000,000	2,500,000	1,700,000	1,565,000	3,521,556	2,560,000	3,300,000
Col. S. & M	....	1,200,000	2,000,000	1,500,000	1,488,000	2,951,000	2,320,000
Butte & Boston	.....	.....	.....	.....	.....	1,103,125	5,357,723
All others	4,593,054	789,864	1,643,621	34,677	521,295	400,000	162,661
Total	46,093,054	67,798,864	57,611,485	78,699,677	98,504,991	104,589,353	111,010,000

TABLE VI.—*Copper-Product of the World, by Continents, in Tons of 2240 Pounds.*

[COMPILED BY H. R. MERTON & CO., LONDON.]

COUNTRIES.	1879.	1880.	1881.	1882.	1883.	1884.	1885.	1886.	1887.	1888.	1889.	Percentage of Total Product, 1889.
EUROPE.												
Great Britain	3,462	3,662	3,875	3,464	2,620	3,350	2,773	1,471	1,500	1,456	1,500	.568
Spain and Portugal:												
Rio Tinto	13,751	16,215	16,666	17,389	20,472	21,564	23,484	24,700	28,500	29,700	*32,000	12.509
Tharsis	11,324	9,151	10,203	9,000	9,800	10,800	11,500	11,000	11,000	11,000	*11,000	4.165
Mason and Barry	4,692	6,603	8,170	8,000	8,000	7,500	7,000	7,000	7,000	*7,000	*5,250	1.99
Sevilla	1,360	1,705	1,340	1,885	2,026	2,000	1,800	2,135	2,300	1,709	1,350	.511
Portugueza	770	1,000	1,410	1,700	2,357	2,300	1,665	1,258	856	*900	*900	.341
Other mines	800	800	800	800	1,000	2,251	2,424	3,560	4,400	*7,000	*6,500	2.463
Germany:												
Mansfeld	8,400	9,800	10,999	11,536	12,634	12,582	12,450	12,595	13,025	13,380	15,506	5.877
OtherGerm'n	600	1,000	1,743	3,552	3,568	2,200	2,800	1,870	1,850	*1,850	*1,850	.701
Austria	255	500	474	474	572	670	585	550	700	1,010	800	.307
Hungary	1,019	900	800	661	661	614	504	500	500	858	*300	.113
Sweden	800	1,074	995	798	732	662	775	600	500	1,036	1,000	.382
Norway	2,412	2,426	2,640	2,590	2,630	2,706	2,560	2,220	1,450	1,570	1,257	.476
Italy	1,140	1,380	1,480	1,400	1,600	1,325	835	900	2,500	3,500	*3,500	1.326
Russia	3,081	3,154	3,411	3,537	3,500	4,700	5,100	4,875	5,000	4,700	4,070	1.542
Total Europe	53,866	59,370	65,006	66,786	72,172	75,224	76,255	75,234	81,086	†86,669	86,783	33.271
N. AMERICA.												
United States	23,000	27,000	32,000	40,467	51,574	63,555	74,053	69,971	78,571	103,245	106,507	40.373
Canada	50	50	500	500	1,055	236	2,500	1,440	1,400	*2,250	*2,500	.947
Newfoundland												
Bett's Cove	1,500	1,500	1,718	1,500	1,053	668	778	1,125	1.180	1,300	1,115	.422
Tilt Cove										700	750	.284
Mexico:												
Boleo Co.									1,950	2,566	3,280	1.243
Other Mexican	400	400	333	401	489	291	375	850	100	200	500	.189
T'l N.America	24,950	28,950	34,551	42,868	54,171	64,750	77,706	73,386	83,201	110,261	114,652	43.458
S. AMERICA.												
Chili	49,318	42,916	37,989	42,909	41,099	41,648	38,500	35,025	29,150	31,240	24,250	9.188
Bolivia:												
Corocora	2,000	2,000	2,655	3,259	1,680	1,500	1,500	1,100	1,300	1,450	*1,200	.454
Peru	600	600	615	440	395	362	229	75	50	250	275	.104
Venezuela:												
Quebrada	1,597	1,800	2,823	3,700	4,018	4,600	4,111	3,708	2,900	4,000	5,563	2.108
Argentine Rep.	300	300	307	800	293	159	233	180	170	150	190	.071
T'l S. America	53,815	47,616	44,389	51,108	47,485	48,269	44,573	40,088	33,570	37,090	31,478	11.925
AFRICA.												
Algiers	500	500	600	600	600	260	250	110	150	50	190	.072
Cape of Good Hope	4,328	4,739	3,467	5,716	5,975	5,000	5,450	6,015	7,250	7,500	7,400	2.805
Total Africa	4,828	5,239	4,067	6,316	6,575	5,260	5,700	6,125	7,400	7,550	7,590	2.877
ASIA.												
Japan	3,900	3,900	3,900	4,800	7,600	10,000	10,000	10,000	11,000	11,600	15,000	5.686
Total Asia	3,900	3,900	3,900	4,800	7,600	10,000	10,000	10,000	11,000	11,600	15,000	
AUSTRALIA.												
Australia	9,500	9,700	10,000	8,512	12,271	14,100	11,400	9,700	7,700	7,450	8,300	3.146
Totals	150,859	154,775	161,913	180,390	200,274	217,603	225,634	214,533	223,957	260,620	263,803	100.363

* Estimated. † Figure adopted by the United States Geological Survey.

The paper was illustrated by the author with a series of lantern-views, showing the details of important mines, mining- and concentrating-plants and smelting-works.

Discussion.

F. M. F. Cazin, New York city (communication to the Secretary): The mention of the San Pedro mine in New Mexico, on page 697, of Mr. Douglas's paper, might be construed as placing that mine in the belt of Permian sandstone described in the sentence which immediately precedes. Having worked the Nacimiento (one of the mines in the sandstone) and the San Pedro, I am positive that there is no similarity between them, the latter being in Archæan limestone, both fissured and interstratified with petro-silex. The copper-deposit at the San Pedro, so far as hitherto worked, has been in seams parallel with the bedding; but it has recently been shown that a fissure, faulting the lime for a few feet only, also carries copper. The San Pedro ore is chalcopyrite in continuous seams, whereas the ore at the Nacimiento is copper-glance, giving evidence of its origin through the agency of organic matter in a thin coating of coal or graphite surrounding every nodule and facilitating its separation from the sandstone when mined.

NOTES ON THE EXCAVATION OF THE NEW CROTON AQUEDUCT.

BY J. P. CARSON, NEW YORK CITY.

(New York Meeting, September, 1890.)

The Croton water-shed furnishes the source and storage of water-supply to both the old and the new aqueduct. The Croton river rises in the southern part of Dutchess county, about 68 miles from the lower end of New York city. Its three main branches, called the East, Middle and West branches, flow southward across Putnam county, joining, near its southern boundary, to form the Croton, which continues thence in a general southwest direction across Westchester county to the Hudson river at Croton Point, about 35 miles north of the city. The water-shed above the proposed site of the Quaker Bridge dam extends about 33 miles north and south, with an average width of 11 miles, making in all about 362 square miles, from which, by complete storage, an average daily supply of 250,000,000 gallons may be collected—sufficient for the needs of the city, probably, for the next fifteen years. The water-shed above the Croton dam is 23 square miles smaller, and furnishes the present daily supply of 100,000,000 gallons.

By referring to the map it will be seen that the shortest approximately straight line between the proposed site of Quaker Bridge

dam on the Croton and High Bridge on the Harlem river passes for the greater part down the valley of the Hudson. It was at first contemplated to construct the aqueduct in open cut, but investigation showed that the deep cuttings required would cost as much as a tunnel. This fact, and the consideration of the more permanent character of the work, caused the tunnel to be adopted. Although the line down the Hudson was the shortest, questions of land-damages arose, so that it was decided to go further back from the river, where land was cheaper, and at the same time the necessary blow-offs could be easily constructed. After much deliberation, the location of the present line was adopted, commencing at Croton dam and running down the Saw Mill valley to High Bridge.

Fig. 1 shows the line of the aqueduct, and Fig. 2 its profile. The following data from the hand-book of information of the chief engineer give a digest of the extent and general features of the work.

Total area of water-shed of Croton river above Croton dam = 216,845 acres, or 338.82 square miles.

Total area of water-shed of Croton river above Quaker Bridge = 231,565 acres, or 361.82 square miles.

RESERVOIRS.

Now Existing.

	Contents in million gallons.	Height of dam in feet.	Length of dam in feet.	Elevation of crest of dam.
Croton Dam Reservoir,	2,000	50	270	167
Boyd's Corners Reservoir, on West Branch of Croton river,	2,700	78*	670	600
Middle Branch Reservoir,	4,000	88†	980	380

Now Being Constructed.

	Contents in million gallons.	Height of dam in feet.	Length of dam in feet.	Elevation of crest of dam.
Double Reservoir "I," consisting of Sodom Dam Reservoir, East Branch of Croton river and Bog Brook Reservoir,	9,000	78	500	415

Proposed.

	Contents in million gallons.	Height of dam in feet.	Length of dam in feet.	Elevation of crest of dam.
Reservoir "A," on Muscoot river,	7,000	90‡	1400	410
Reservoir "D," on West Branch of Croton river,	9,000			
Reservoir "M," on Titicus river,	6,000	100	1200	325
Quaker Bridge Reservoir, on the Croton river,	30,000	264	1500	200
Total,	69,700			

* Fifty-six feet from bottom of pipe to overflow.

† Sixty-three feet from bottom of tunnels to top of overflow.

‡ Seventy-five feet from bottom of tunnels to top of overflow.

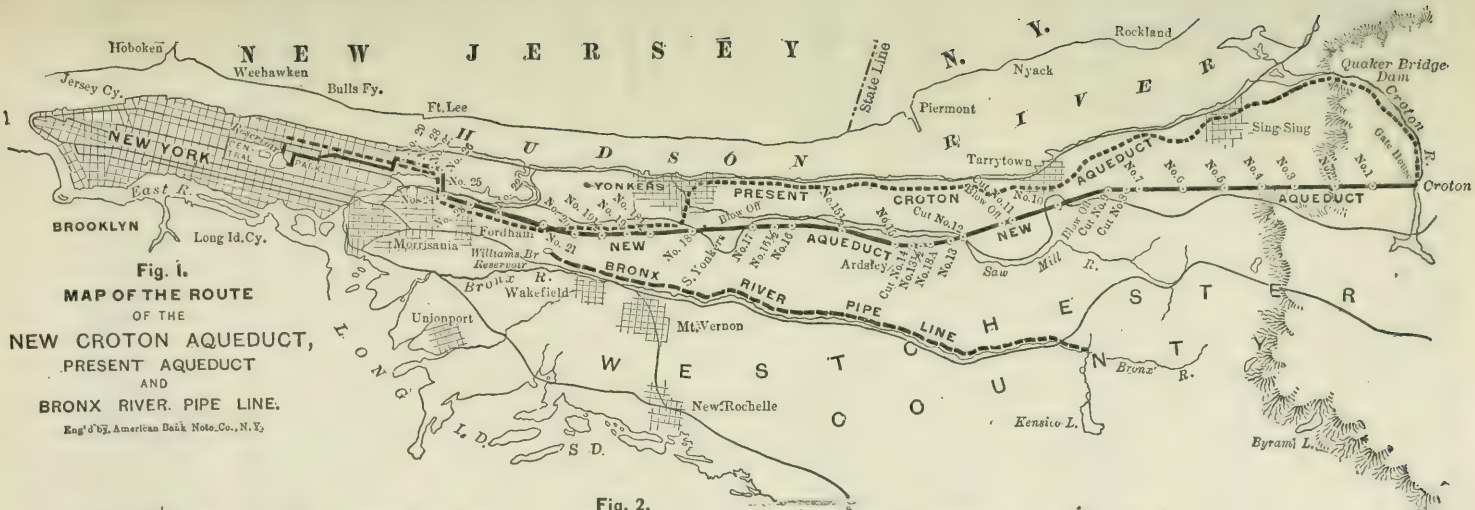
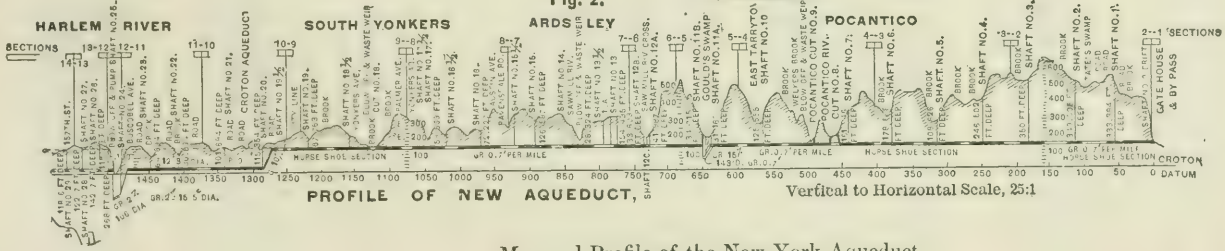


Fig. 1.

MAP OF THE ROUTE OF THE NEW CROTON AQUEDUCT, PRESENT AQUEDUCT AND BRONX RIVER PIPE LINE.

Eng'd by American Bank Note Co., N.Y.

Fig. 2.



PROFILE OF NEW AQUEDUCT,

Vertical to Horizontal Scale, 25:1

Map and Profile of the New York Aqueduct.

DIMENSIONS OF NEW AQUEDUCT.

	Feet.	Miles.
Total length of tunnel not under pressure (horse-shoe), . .	120,370.6	22.80
Total length of tunnel under pressure (circular), . . .	36,079.8	6.83
Total tunnel, including siphon at Gould's Swamp, . . .	156,450.4	29.63
Total length of aqueduct in open trench, ...	5,926.7	1.12
Total of aqueduct from Croton Lake gate-house to One-hundred-and-thirty-fifth Street gate-house, ...	162,377.1	30.75
Total length of pipe-line, ...	12,516.0	2.37
Total length from Croton Lake gate-house to Central Park gate-house, ...	174,893.1	33.12

Total fall from Croton gate-house (elevation, 153) to One-hundred-and-thirty-fifth Street gate-house (elevation of water, 128), 25 feet.

Total fall from Croton gate-house (elevation, 153) to Central Park reservoir (elevation, 118.61), 34.39 feet.

Aqueduct under Harlem River.

Depth of center-line below mean high water, 307 feet.

Maximum inward pressure on aqueduct when empty, 128.74 lbs. per square inch.

Maximum pressure on aqueduct when full, 55.29 lbs. per square inch.

Horse-shoe shape: height, 13 feet $6\frac{36}{100}$ inches; width, 13 feet $7\frac{20}{100}$ inches; inside area, 155.573 square feet = a circle 14 feet 1 inch in diameter.

Prevailing circular shape: diameter, 12 feet 3 inches; inside area, 117.86 square feet.

Circular shape under Gould's swamp: diameter, 14 feet 3 inches; inside area, 159.48 square feet.

Circular shape under Harlem river: diameter, 10 feet 6 inches; inside area, 86.59 square feet.

Flowing capacity of aqueduct above Shaft No. 20 (not under pressure), 318,000,000 gallons per twenty-four hours.

Flowing capacity of aqueduct below Shaft No. 20 (under pressure), 250,000,000 gallons per twenty-four hours.

NOTE.—Difference, 68,000,000 gallons, expected to be distributed in annexed district through a proposed intermediate reservoir.

Flowing capacity of pipe-line (8 pipes of 48 inches diameter each) from 135th Street gate-house, 250,000,000 gallons (approximated) per twenty-four hours.

NOTE.—Four of these pipes deliver 125,000,000 gallons into the Central Park Reservoir; the balance of 125,000,000 gallons is taken into city distribution through the other four pipes.

Gate-Houses.

Croton Lake gate-house.—Built in view of the construction of the Quaker Bridge dam.

One-Hundred-and-Thirty-fifth Street gate-house.—At end of the aqueduct and beginning of the pipe line.

Central Park gate-house.—At the end of the pipe line and at the head of Central Park reservoir.

Blow-offs.

Pocantico blow-off: distance from Croton gate-house, $9\frac{29}{100}$ miles; empties into Pocantico river and Hudson river through Sing Sing.

Ardsley blow-off: distance from Croton gate-house, $15\frac{65}{100}$ miles; empties into Saw Mill river through Yonkers.

South Yonkers blow-off: distance from Croton gate-house, $21\frac{151}{100}$ miles: empties into Tibbett's brook and Harlem river through Van Courtland lake.

Shaft No. 25: distance from Croton gate-house, $28\frac{38}{100}$ miles; empties into Harlem river.

Overflow at Shaft No. 26: distance from Croton gate-house, $28\frac{48}{100}$ miles; empties into Harlem river.

Shafts and Open-Cuts.

Total number of principal shafts, 32; total number of auxiliary shafts, 10; total number of shafts to be left open, 28.

Deepest shaft is No. 25; depth, $419\frac{7}{10}$ feet. Shallowest shaft is No. $17\frac{1}{2}$; depth, 22 feet. Average depth of shafts, $122\frac{1}{10}$ feet.

Total number of portals, 9.

Total number of open-cuts, 6, as follows: Open-cut 8A, length 84 feet; open-cut 8B, length 733 feet; open-cut 9, length 1312 feet; open-cut 12, length 123 feet; open-cut 14, length $459\frac{2}{10}$ feet; open-cut 18, length $3215\frac{4}{10}$ feet.

MISCELLANEOUS INFORMATION.

Brick-work.

The amount of brick-work laid in the New Croton Aqueduct is estimated at 312,258 cubic yards, or about 163,000,000 of bricks, which would build thirty-three structures the size of the *Tribune* building.

The total area of the inside surface of the aqueduct from Croton lake to One-hundred-and-thirty-fifth Street gate-house is equal to 7,092,823 square feet, or 162.83 acres, or about equal to $\frac{1}{5}$ the area of Central Park.

Excavated Material.

The total excavated material of the entire aqueduct exceeds 2,800,000 cubic yards.

The total excavated material plus the masonry placed exceeds 3,250,000 cubic yards. This is equivalent to 83 per cent. of the volume of the great pyramid of Cheops. This material would be sufficient to build a wall 10 feet thick and 55 feet high around Manhattan Island, 30 miles in length on the water front.

The amount of dynamite used in blasting on the aqueduct, exclusive of the amount used in sinking the shafts, was over 5,800,000 lbs., or over 2900 tons.

Elevations.

Maximum level of present Croton lake, . . . 167 feet.
Level of water at One-hundred-and-thirty-fifth Street gate-house, ... 128 "
Level of water at Central Park gate-house, . . $118\frac{61}{100}$ " (reservoir full).

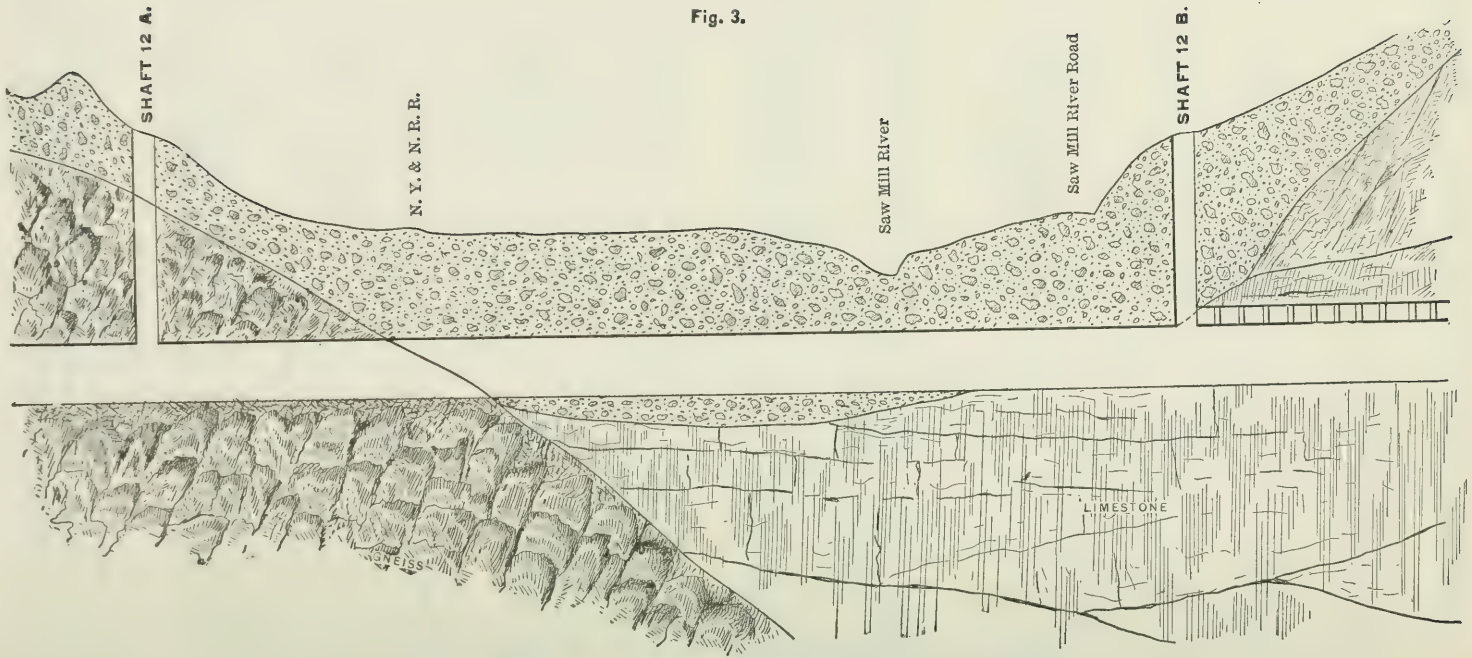
All elevations are referred to the Croton datum, which is $3\frac{61}{100}$ feet below mean high water of Harlem river or city datum.

The flowing capacity of the aqueduct, 318,000,000 gallons per twenty-four hours, is equivalent to a stream 50 feet wide and 10 feet deep flowing 59.1 feet per minute, or about 1 foot per second, or about $\frac{7}{10}$ths of a mile per hour.

Geology.

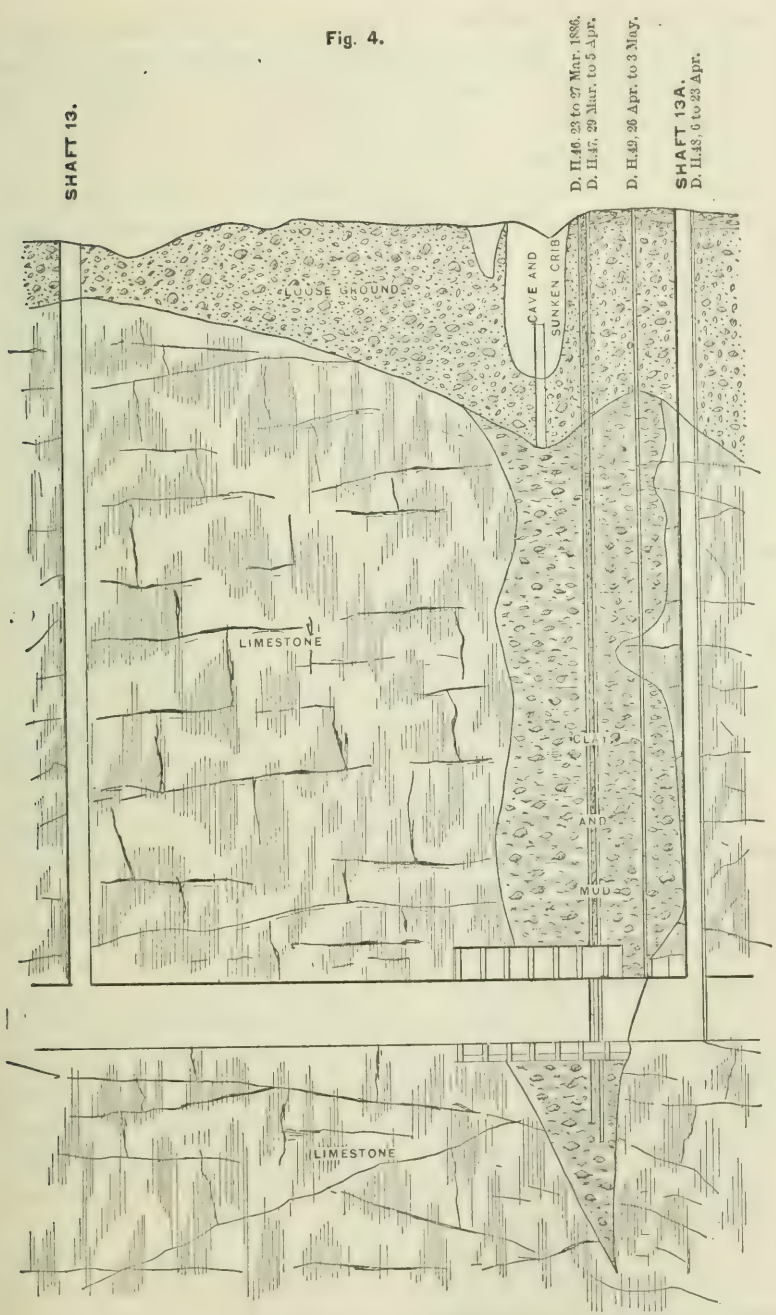
The rocks are metamorphic, principally gneiss, which varies from firm granite to ordinary mica-schist. The first 14 miles from Croton dam to the crossing of the Saw Mill river are through this material,

Fig. 3.



Longitudinal Section of Aqueduct from Shaft 12 A to Shaft 12 B.

Fig. 4.



Section showing Bad Ground near Shaft 13 A.

and it constitutes about 90 per cent. of all the rock passed through to Harlem river. The general strike is N. 20 E., and the dip varies from 20° to 85° E. It was generally observed that where a brook occurs on the surface the stratification of the rock below has undergone great disturbance. This is characterized by faults, a jointed structure, and seams of variable size filled with talc, decomposing feldspar and clay. This faulty ground, after a few weeks' exposure to the air, is liable to fall down, by reason of the slacking or swelling of the seams.

The only interesting mineral specimens found were a few insignificant crystals of garnet and pyrite, and some fine pieces of fibrous asbestos—the latter in the tunnel under the Harlem river.

At the Saw Mill river a mass of siliceous dolomitic limestone was for the first time encountered, striking N. 46° E., and dipping 60° to 70° S.E. Near the surface it was decomposed, but gradually became harder, and with an occasional small pot-hole or clay-seam, continued quite regularly for 4200 feet. It then gradually became decomposed (260 feet) and finally the soft ground of Shaft No. 13 South Heading was reached. This extended practically for 110 feet, and the driving of the tunnel through it caused more anxiety, trouble, and expense, than any other similar work in this country. The predominant material was a hard, compressed, yellow mud, composed of fine particles of mica, sand and clay. Through the mass there were vugs of white and yellow clay, and several strong, water-bearing seams, filled with white sand and gravel. The water from these seams, when they were cut, formed with the mud a sticky mixture of the consistency of pea-soup. This mixture, being strained or allowed to settle, gave a sandy residue, upon which one could stand without sinking. A lump of the mud readily dried and hardened in the open air, but dissolved immediately in water. It was equivalent to a quicksand.

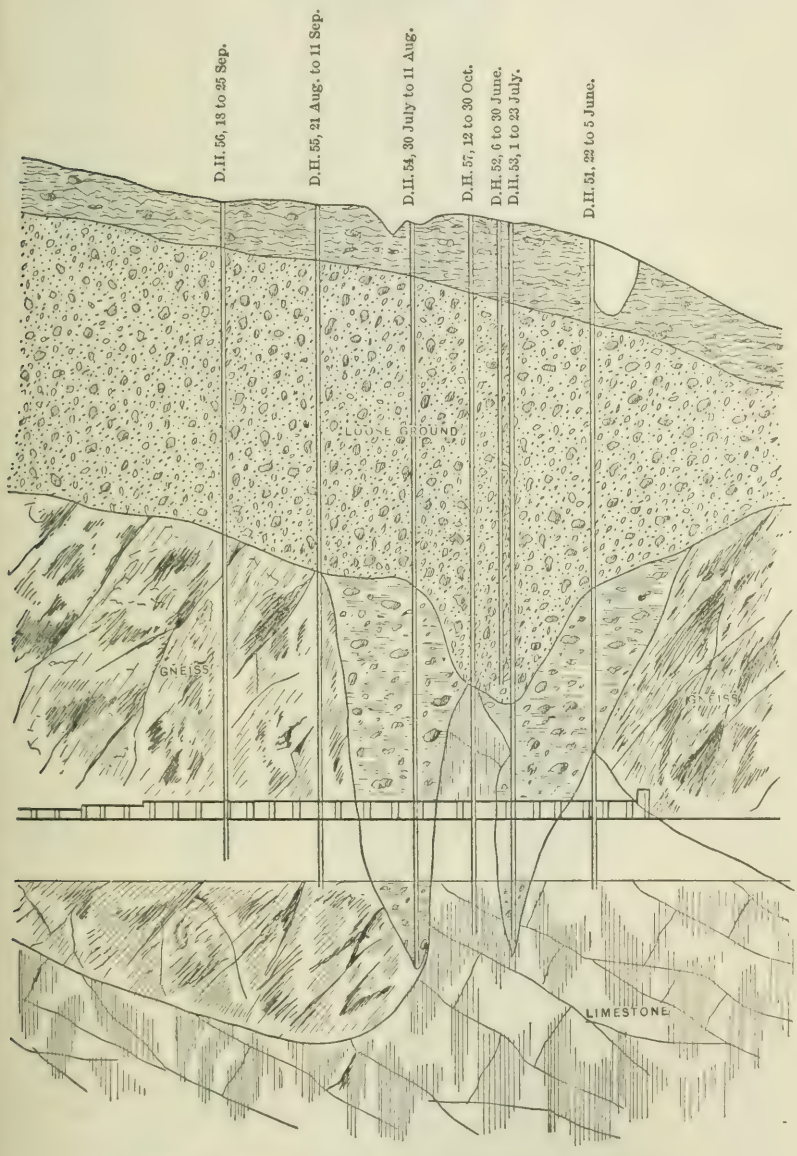
South of this pocket, the limestone again occurs for 2230 feet, then decomposed gneiss for 640 feet, then two small peaks of limestone, with irregular seams of feldspar, clay, and boulders, which, with a cave from the surface, caused some trouble. Then came two pockets of clay, each extending about 60 feet. At the southern limit, the limestone and gneiss merged into each other in wavy lines, without a distinct dividing seam.

This bed of limestone may be said to extend 6800 feet. It is to be regretted that the tunnel-notes on the dip and strike are not as complete as would be desirable.

Figs. 3, 4, and 5, are longitudinal sections on the line of the tunnel, and Fig. 6 is a cross-section taken in the excavation for the

blow-off at Ardsley. The longitudinal sections south of Shaft 13 were determined by the diamond drill.

Fig. 5.


D.H. 56, 13 to 25 Sep.
D.H. 55, 21 Aug. to 11 Sep.
D.H. 54, 30 July to 11 Aug.
D.H. 57, 12 to 30 Oct.
D.H. 52, 6 to 30 June.
D.H. 53, 1 to 23 July.
D.H. 51, 22 to 5 June.
LOOSE GROUND
GNEISS
GNEISS
LIMESTONE

Section between Shafts 13 and 14, as shown by Drill-holes.

For the next 4.87 miles the rock is mica-schist, with the occasional jointed structure. Then, in Shaft 17, occurred for 265 feet a pocket

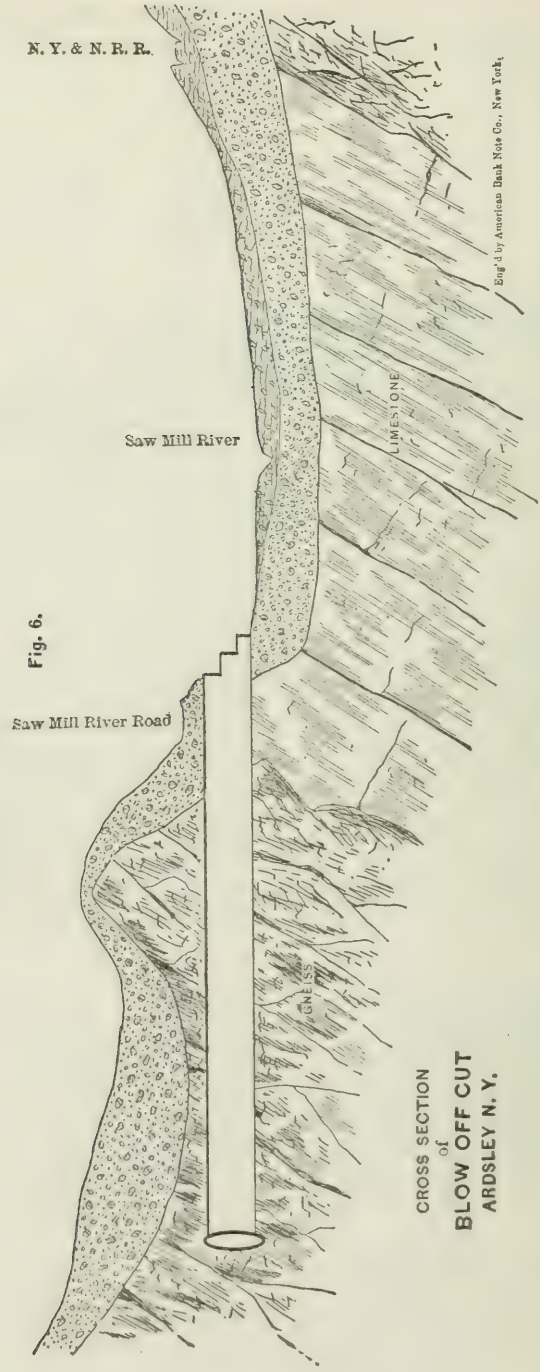
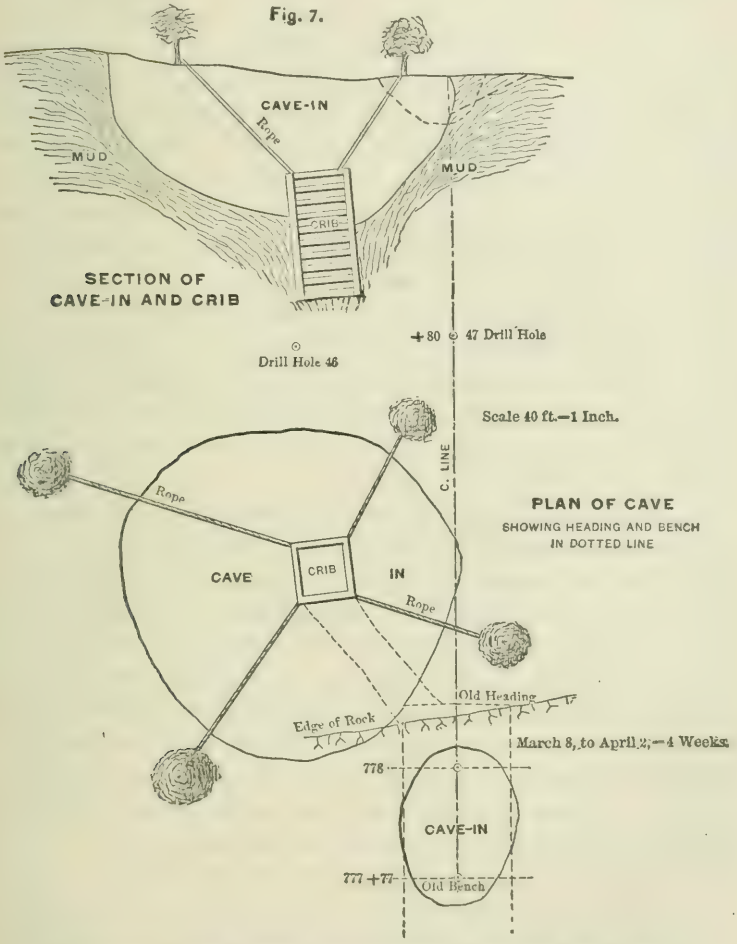


Fig. 6.

New York Aqueduct.

of sand and drift-boulders, with decomposed rock for 60 feet on either side. The distance below the surface was 150 feet.

For the next 3.39 miles, with the exception of 3200 feet of open-cut, the mica-schist is again found. The tunnel then approaches



Section and Plan of Cave. Shaft 13, South Heading (See Fig. 4).

within 30 to 40 feet of the surface, and passes through 1730 feet of sand and boulders. It then soon follows down the incline and enters the gneiss, which, with the exception of a pear-shaped horse of greenstone, about 50 feet long, occurring at Shaft 20, continues 4.34 miles to the Harlem river.

Recapitulation: Croton Dam to Harlem River.

Gneiss,	25.54 miles.
Clay and limestone,	1.29 "
Sand,	0.38 "
Open-cut,	1.12 "
Total,	28.33 miles.

The Excavation.

It was not contemplated to line the entire tunnel with brick, but the developments of the nature of the rock soon demonstrated the necessity for doing so. The contract-price for tunnel-excavation was $7.00 per cubic yard, or $52.88 per running foot. This included everything. But in case of excessive excavation, the vacant space between the rock and the brickwork was to be filled with broken stone packed dry, at the expense of the contractor.

After the order had been given to line the entire tunnel with brick, it was further ordered to fill the vacant space with rubble, built in radial lines, so as to strengthen the arch. The simple excavation then became complicated with the question of "making rubble," which was a very great opportunity for those who "would not play false, and yet would wrongly win."

The following notes are from personal observations, and data compiled from the records of Sections 7 and 8, Division 3.

The American center-cut system was used. The plant is described below. In driving the heading, the usual difficulty was experienced in keeping the floor at or about the grade of the spring-line. It invariably crept up one or two feet (the roof following suit); the alleged reason being that this was "on account of the drainage," or "on account of the very bad rock, which breaks high." The truth was that the frequency of drill-holes showed the rock to be very good. In practice the ordinary rock could be broken as desired by placing the drill-holes about a foot within the line.

The area of the horse-shoe section with 12-inch lining was:

				Per cent.
Above spring-line,	73 sq. ft.	Average as excavated,	105 sq. ft.	Excess, 44
Below "	131 "	" "	178 "	" 35
Total,	204		283	38

The areas of the heading and bench as excavated are in the ratio of 3 : 5. The following table shows the average weekly progress and the proportion of time lost when working through different kinds of material, and by different methods.

In this table, the column headed "Per cent. no progress" shows the proportion of time during which the work stood still.

TABLE I.

Average Weekly Progress of Tunnel on Division 3 as Worked by Different Systems through Different Materials.

CHARACTER OF MATERIAL.	HEADING AND BENCH WORKED TOGETHER.							HEADING WORKED ALONE.							BENCH WORKED ALONE.						
	Number of working points.	Weeks worked.		Area, square feet.	Linear feet.			Number of working points.	Weeks worked.		Area, square feet.	Linear feet.			Number of working points.	Weeks worked.		Area, square feet.	Linear feet.		
		Total.	Per cent. no progress.		Total.	Per week.	Cubic yards per week.		Per cent. no progress.	Total.		Total.	Per week.	Cubic yards per week.		Per cent. no progress.	Total.		Total.	Per week.	Cubic yards per week.
Soft ground, clay, sand and mica	2	128.0	28	507.7	110	1.4*															
Sand and boulders	1	62.3	40	490.	265	4.25															
Decomposed rock, limestone, feldspar and mica-schist	7	152.0	31	361.	1,487	9.8	131.	1	12.5	8	165	188	23.5	143.6	2	42	19	196	230	12.1	87.7
Limestone	3	95.5	24	283.	2,148	22.4	235.	3	13.4	52	105	1752	33.8	131.0	3	3	57	178	1531	27.0	178.0
Mica-schist	8	427.0	4.4	283.	13,832	30.0	314.	8	0.6	164	105	7711	47.0	183.0	8	2.4	247	178	8913	36.1	238.0
								At foot of shaft.							At foot of shaft.						
								10	20	54	...	528	9.8	...	6	35	23	...	1.91	8.3	...
															Sinking shaft.						
															7	5	98	200	658	6.7	49.1

* This figure is the average, not of the whole period, but of the last stage (and best practice).

THE DIFFERENT SYSTEMS OF EXCAVATION CONSIDERED IN RELATION TO THE SUBSEQUENT BRICKWORK.

For 16 months, or until Sept. 1886, the headings and bench were worked together, being kept as near together as possible (the average distance between them being about 50 to 75 feet) and the tunnels had advanced about 1500 feet from the shaft.

It was then decided to change this procedure and drive the headings first. This was done, by constructing an inclined plane about 400 feet long from the grade of the tunnel to the grade of the heading.

The following reasons were advanced for abandoning the old system:

1. Because progress was delayed by the difficulty of ventilation; whereas by first connecting the headings, the air would be improved and the work really advanced. This is in part true.

2. Because the length of haul caused much difficulty in moving the "runs," and in getting rid of the broken stone. Moreover it was desirable to start the masonry. In taking out the smaller heading both these operations would be impeded to a less extent than by excavating the enlarged heading at once.

3. As the rock from the heading was too much shattered to be fit for rubble, it would prove more advantageous to leave the bench to be subsequently excavated, and save the re-handling of this much of the broken stone by using it directly for the masonry.

Under certain local conditions, these two latter reasons may be good, but a consideration of facts connected with this work will serve the double purpose of showing their unsoundness here, and of illustrating the manner of conducting the work.

The "runs" were movable bars, placed across the tunnel, to support the continuation of the track from the heading, so that the muck could be directly dumped from the cars above to those below. The labor was not increased on this account; hence it may be regarded as an incidental detail. The transportation was done in cars holding about a cubic yard of broken stone, equivalent to half a yard of solid rock. One mule hauled two and sometimes three cars at a trip. Up to a distance of 1500 feet, 6 to 8 cars were allowed to each heading. It is easy to show that a third more would suffice for an increased haul up to 4500 feet, which was greater than ever occurred.

The average daily excavation of the bench and heading was 53 cubic yards of solid rock and of the bench alone 40 cubic yards. The transportation of this material would require trips of two cars, to be made respectively every 24 and 30 minutes. The relative difference of interruption was therefore practically the same in both cases.

The subsequent excavation of the bench involves the following questions:

1. The quantity of rubble required;
2. The rate of construction of the masonry; and rate of excavation of the bench.

1. *The Quantity of Rubble Required.*—The rock from the heading was too much shattered for use as rubble, and the same applies to about 25 per cent. of that from the bench.

Requirements of Rubble.

	Average daily progress, feet.	Per running foot.	Cubic yards of rubble used per day.
Side-walls, . . .	40	0.78	17
Arch, ...	20	1.43	29
Invert, ...	...	0.40	...
Total, . . .	...	2.61	46
Bench, ...	6	6.6	40 = 64 broken.

The masonry was built in the above order, and required 1.74 cubic yards of solid rock per running foot. The bench contained 6.6 cubic yards, or after allowing for waste, sufficient for three running feet of masonry. It would then appear proper to leave one-third of the bench in the tunnel to be used as needed. Often more than one-half was left. In practice the stone was used by the masons direct from the car—whenever possible—but even under the most favorable circumstances about a third more was daily excavated than could be so employed. The broken stone was piled up at a convenient place in the tunnel, about 7 feet high and 6 feet wide—a quantity about sufficient for one running foot of arch. It had to be rehandled; so that it would generally have been more convenient to have piled it on the surface. The cost of hoisting it to and lowering it from the surface was saved, which amounted to 17 cents per cubic yard, while the increased cost of working the heading and bench separately (supposing work in both tunnels) was, as will be subsequently shown, $2.10 per cubic yard.

2. *The Question of Time.*—By referring to Table I., it will be seen that with mica-schist the average progress was as follows:

When heading and bench were worked together, 30 feet per week.

When heading was worked alone, 47 feet per week.

When bench was worked alone, 36.1 feet per week.

It therefore required 1.3 week to excavate a length of bench equal to the length of heading excavated in 1 week. Hence, 2.3 weeks were required to excavate 47 feet of complete enlarged tunnel-section. The weekly progress was therefore 20.43 feet, or 32 per cent. less,

working separately, than when working together both heading and bench.

As the relative daily advance of the masonry to the excavation (done separately) was as 20 to 3.4, obviously one would overtake the other and would have to stop. So no time was saved by rushing the masonry, as its completion was limited by that of the excavation.

The tunnel and brickwork can be finished practically at the same time, without delay to either: *first*, by carrying on the brickwork without rushing, at a reasonable distance behind the excavation, say 500 feet, and at the same rate of advance, say 5 feet per day; *second*, by starting the brickwork after 80 per cent. of the excavation is completed, and continuing the excavation of the full section.

By excavating the heading and bench separately, the time for the completion of the work is increased 10 per cent.

At Shaft 15, out of 7000 feet in length, 3400 feet or 49 per cent. of heading and bench were excavated separately.

The lower portion of the aqueduct, Shaft 11 B, Station 687 + 00, to Shaft 24, Station 1489 + 88, after deducting 3798 feet of open-cut, consisted of 76,490 feet of tunnel. The heading and bench were excavated separately in 18,026 feet of this length, or 23 per cent. The delay was 11.5 per cent.; but as frequently both headings were not driven at the same time, this percentage should be doubled; and, from the fact that some shafts were idle for over two months, the completion of the excavation took 45 months instead of 33, the increase of time being 36 per cent. A part of this delay was reasonable, on account of the bad ground in Shaft 13, but there was no good cause why the work should have been delayed so long as 4 months after No. 13 was finished. The approximate increase of cost of driving heading and bench separately, as compared with driving them together, is shown in the transcripts from actual average practice, given at the close of this paper under "Cost of Driving."

One might infer from the foregoing, that the contractors were unmindful of their own interests—which, in the light of subsequent events, would be a great mistake. The most probable reason of their choice of method was that there was not sufficient boiler-power, and they did not wish to incur the expense of further additions.

To each shaft there were two tubular boilers, 75 horse-power. As many of these boilers had been burned out soon after the work started, their actual power was probably reduced to about 60 horse-power. As the work advanced, this power was not sufficient; the ventilation became fouler, and the receipts consequently fell off; to

keep them up, the sub-contractors were forced to start the masonry, and rush it regardless of all principle.

On the northern portion of the work, from station 0 to 687 + 00, the length of the tunnel, after deducting the open-cuts, was 66,571 feet. Of this, 3550 feet of bench and 2030 feet of heading were driven separately, or 8 per cent. of the whole tunnel, as against 23 per cent. done in this way on the southern portion. The reason is, that the work was not sub-let; the management was concentrated, and it was cheaper to drive the full section.

Lighting and Ventilation.

The Lighting.—For the first two years, gasoline torches, miners' lamps, a few candles, and electric lights to some extent, were used. The two former predominated. The air finally became so foul, that an order was issued forbidding the use of torches or lamps, and after a time the torches were abandoned. In the construction of the masonry only candles and electric lights were employed.

The Means of Ventilation.—On the northern portion of the work, the arrangement for improving the ventilation was by means of what the men termed, very properly, "smoke-boxes." These consisted of wooden flues, varying in section from 10 inches by 24 inches to 18 inches by 36 inches, placed continuously along the side on the floor of the tunnel. They were too small. They should have been air-tight, but they were continuously being broken or damaged by falling stones and other accidents. Air-jets were inserted at intervals, which had a cooling and consequently detrimental effect. The automatic action of the flue depends upon the difference of density between the air within and the air outside the tunnel. As the average temperature inside was about 60°, the desired action could only occur during a few months of winter when the outside temperature was low. Had a sufficiently large flue been placed along the roof and connected with a fire-place, the arrangement, though clumsy and expensive, would have been rational, whereas the system as employed was a miserable farce.

For the ventilation of the southern portion, Baker blowers No. 4½ were used. They were run by independent 10 horse-power engines at from 70 to 150 revolutions per minute—80 revolutions being a fair average. According to the manufacturers' circular, the capacity was 16½ cubic feet per revolution. The air was conveyed by spiral-riveted 12-inch iron pipe with ends fitting into each other, and suspended on the side of the tunnel near the spring-line. It was

carried to within 100 or 200 feet of the blasting operations. The effectiveness of this system depended on the manner in which it was carried out, as details, which will be given later, will show.

The air was usually forced. The reasons given for this procedure were that the men liked it better, and that the eccentric of the engine was set to work that way. At Shaft 24, however, a Sturtevant fan was arranged for both forcing and suction.

Shaft 16 was provided with two blowers, but they were seldom used, because it was considered too much trouble to keep the pipes in repair.

At Shaft 15, the volume of air delivered in each heading and the loss by leakage was as follows—the velocity being determined by a standard Casella anemometer:

Date.	Number of Experiments.	Distance from Blower. Feet.	Revolutions of Blower per minute.	Theoretical Volume. Cubic feet.	Actual Volume. Cubic feet.	Per cent. lost by friction and leaks.	Volume delivered to each heading. Cu. ft.
18–22 Dec., '86.....	7	2350	113	1854	942	54	471
7–21 March, '87.....	4	3000	113	1854	870	56	435
31 Mch. and 4 Apl., '87.....	2	3400	110	1815	320	82	160
						Average,	355

To the above must be added the air from each drill, 65 cubic feet a minute; but as the drilling occupies but 7 hours, the average quantity delivered during the ten-hour shift is only 45 cubic feet per minute.

The Standard of Purity of Air.—Pure mountain air in all parts of the world has been found to contain 3 to 4 parts of carbonic acid in 10,000. When people speak of good ventilation, they mean air which contains less than 0.07 per cent. of carbonic acid. Sanitarians declare that the limit should not exceed 0.06 per cent. In tunnels, 0.35 per cent. is allowed. Experience shows that an atmosphere containing 0.10 per cent. of this gas is not only offensive but unwholesome; 10 per cent. is deadly.

Pure carbonic acid, in small proportions, is not necessarily injurious. Men frequently work without ill effects in mineral-water factories, certain rooms in breweries and champagne-bottling vaults, where it may be present in the air to the proportion of 1.5 per

cent. But in questions of ventilation great importance is given to the results of chemical tests of air in reference to this gas, because it is usually accompanied by vapors and suspended organic particles that are both offensive and dangerous. Variations in the amount of the gas to the extent of 3 or 4 parts in 10,000, indicate a corresponding variation in the extent of the vitiation. The quantity of carbonic acid can be determined with comparative ease by several methods, while there are no such tests for the accompanying impurities. For convenience the gas is measured, and thus is obtained a comparative measure of the extent to which the ventilation is being affected.

The following table shows the composition by volume of inspired and expired air:

	Pure air inspired.	Air expired.
	(A. Smith.)	(Huxley.)
Nitrogen	79.000	79.000
Oxygen	20.965	16.300
Carbonic acid	0.035	4.700
	100.000	100.000

It is seen that air which has been once breathed, contains a hundred times more carbonic acid than pure air; or, speaking roughly, it may be said to contain 5 per cent. while the oxygen is 5 per cent. less and the nitrogen remains unchanged.

From experiments conducted in a closed lead chamber, the following effects were observed by Prof. Angus Smith.

	Volume, per cent.	
	O	CO_2
Instantaneous headache occurred,	20.19	3.84
Decidedly close,	19.61	1.26
Candles went out,	18 80	2.28
Spirit-lamps went out in 150 minutes,	18.40	2.45
Difficult to remain in many moments,	17.20	

In a mine in England, the worst specimen of air examined by the same authority gave:

	Volume.
In one case minimum oxygen,	18.27 per cent.
In another case maximum carbonic acid,	2 50 "

The average of 339 analyses of air in mines in England gave:

	Volume.
Oxygen,	20.260 per cent.
Carbonic acid,	0.785 "

Other analyses showed:

	Volume, per cent.
In the worst parts of theatres in London, carbonic acid,	0.32
In workshops, carbonic acid,	0.30
Smoking cars in United States, carbonic acid,	0.098 to 0.37

In Germany,* in a school-room of 252 cubic meters capacity and containing 64 children, a series of 14 experiments was made in one day. The carbonic acid contained in the air of the room varied from 0.221 per cent. to 0.936 per cent.

Air Required for Respiration.—A man inhales at least 22 cubic inches at each breath, which, at 20 respirations a minute, gives 15 cubic feet as the quantity passed through the lungs in an hour. The air exhaled contains from 4 to 5 per cent. of carbonic acid; so that 0.6 or 0.7 of a cubic foot of this gas is given off in this time, supposing the man to be awake and at rest. The amount of carbon eliminated in 24 hours is represented by a piece of charcoal weighing 8 ounces. When at work, the respirations will be at least four times faster and the air required will be 60 cubic feet an hour, and the carbonic acid expelled will be 2.40 cubic feet.

As to the quantity of cutaneous excretions, few authorities give available figures, and those who do, differ widely. Huxley states that a man exhales from the lungs 12,000 grains of carbonic acid in 24 hours, and in the same time, he throws off from the skin 400 grains, also 300 grains of solid organic matter.

Dalton states that in 24 hours the quantity of carbonic acid amounts to 0.5 per cent. of that expelled from the lungs; and from figures given, it is deduced that the solid organic matter is 590 grains. The average per hour may be assumed to be: carbonic acid, 17 grains or 3.5 per cent. of that expelled by breathing; solid organic matter, 15 grains or 3.0 per cent. of that expelled by breathing.

	Per hour.
Carbonic acid expelled from lungs while working,	2.40 cubic feet.
Carbonic acid expelled from skin while working, 3.5 per cent.	.08 "
Total carbonic acid expelled from lungs and skin while working,	2.48
Solid organic matter expelled from lungs and skin while working,	60 grains.

It is assumed that the proportion of carbonic acid in the air of the tunnel should not exceed 0.35 per cent. In order to keep it at this standard a man will require:

* *American Chemist*, vol. i., p. 158.

	700 cubic feet of air per hour.		
Or,	11.6	"	" per minute.
For 1 grain of organic matter. .	1.5	"	" "
Total required for respiration, .	13.1	"	" "
For half a lamp, ...	7.1	"	" "
Total, ...	20.2	"	" "

Or, the air required for a man and half a lamp is, in round numbers, 20 cubic feet a minute. When the electric arc-light was used, it was frequently found necessary to employ one oil-lamp for two men. This would be a fair allowance in estimating air for ventilation.

A mule is supposed to require six times as much air as a man, or 78.6 cubic feet. Since each mule carried a lamp, the total air required for a mule was 92.8 cubic feet per minute.

The Gasoline Torch.—On the northern portion of the work, three or four of these torches were used in each heading and sometimes as many as six or eight by the bricklayers; on an average, one torch served for five men. On the southern portion of the aqueduct, they were not so extensively used.

The torch held about $3\frac{1}{2}$ gallons of gasoline or naphtha (25 pounds) and was refilled every half-shift; so it consumed about 5 pounds of gasoline per hour. In the combustion of gasoline the following reaction takes place:

$C_5H_{12} + 16\ O = 5\ CO_2 + 6\ H_2O.$

$72 + 256 = 220 + 108.$ Then,

72 : 256 : : 1 lb. gasoline burned : 3.56 lbs. O required.

72 : 220 : : 1 lb. gasoline burned : 3.05 lbs. CO_2 produced.

72 : 108 : : 1 lb. gasoline burned : 1.5 lbs. H_2O (aqueous vapor).

One cubic foot of air at 60° weighs 0.0763 pound. The oxygen in it $= 0.0763 \times 23\% = 0.0176$ pound. Hence the quantity of air required to furnish 1 pound of oxygen $= \frac{1}{0.0176} = 57$ cubic feet.

One cubic foot of carbonic acid at 60° = 0.1167 pounds; or 1 pound carbonic acid = 8.57 cubic feet. Hence for the combustion of 1 pound of gasoline, there is required, 3.57 pounds $O \times 57 = 203.5$ cubic feet of air. As 5 pounds are burned per hour, 17 cubic feet of air are required each minute. The carbonic acid produced would be: 3.05 pounds $CO_2 \times 8.57 = 26.1$ cubic feet, or, per minute, 2.20 cubic feet. To dilute this to the standard would require 630 cubic feet of air per minute, or almost sufficient for 32 men with 16

lamps. One torch served for 5 men, therefore its effective light or work was one-sixth that furnished by the miners' lamps.

The gasoline torch may be most excellent for street-peddlers, but is a fiendish device for use in a tunnel.

The Smoke and Carbonic Acid Produced by a Miner's Lamp.— The attempt was made to estimate approximately the amount of smoke produced. As this depends upon the many variable conditions under which the lamps may be burned, the results of a rough average determination will answer the purpose as well as a more complete analysis. The lamp was weighed and placed in a large china plate, and enclosed by a cylinder of stiff paper, 24 inches high, so arranged as to admit variable quantities of air. Another large plate was suspended from above. Of course, much of the smoke escaped, but that deposited on the paper and plate was collected and weighed. The test was made with the ordinary 150° kerosene oil used in the tunnel.

The average of six trials gave :

Soot collected per hour, ...	23.1 grains,	2.81 per cent.
Oil completely burned per hour, . .	800.1 "	97.19 "
Oil used per hour, ...	823.2	100.00 "

In other words, a miner's lamp used about 2 ounces of oil an hour, of which 3 per cent. went off as smoke.

The following is the reaction occurring when kerosene oil is completely burned: $C_6H_{14} + 19O = 6CO_2 + 7H_2O$; or $86 + 304 = 264 + 126$. Then, 86 : 304 : : 1 pound of oil burned : 3.53 oxygen required; 86 : 264 : : 1 pound of oil burned : 3.07 CO_2 produced; 86 : 126 : : 1 pound of oil burned : 1.46 H_2O produced. Hence the air required for the combustion of 1 pound of kerosene is $3.53 \times 57 = 201.2$ cubic feet; but the lamp uses per hour 823 grains less 3 per cent. in smoke $= 0.114$ pound; the requirement of air per hour is therefore $201.2 \times 0.114 = 23$ cubic feet.

The carbonic acid produced per hour is 3.07 pounds $\times 8.57 \times 0.114 = 3.00$ cubic feet. To dilute this to the standard will require 854 cubic feet of fresh air, or 14 cubic feet a minute. The smoke amounts to 23 grains an hour, or about 0.4 grain a minute. It is assumed that the deleterious effects of organic impurities are one-third greater than those of the inorganic. Then the air required to counteract the presence of each grain of solid inorganic matter should be 0.5 cubic foot a minute. Therefore, the total air required

for each lamp will be 14.2 cubic feet. A candle requires 12 cubic feet per minute.

Gas and Dust Formed by the Explosion of Dynamite.—It is estimated that, per pound of nitro-glycerine exploded, 4 cubic feet of carbonic acid are produced.

The quantity of explosive used during a shift was 32 pounds of 60 per cent. dynamite in the heading and 38 pounds of 40 per cent. on the bench, or in round numbers 7 pounds of 50 per cent. dynamite per hour. Therefore, the air required to dilute to the standard proportion the 2 cubic feet of carbonic acid formed from the explosion of 1 pound of 50 per cent. dynamite is 570 cubic feet per hour, or 9.5 cubic feet per minute.

The absorbent was composed of wood-pulp, 32.5 per cent.; sulphur, 5 per cent.; nitrate of soda, 62.5 per cent. This of itself forms a gunpowder, and produces about as much carbonic acid as the glycerine. The smoke and gases vary with different absorbents. When infusorial earth was used, it formed about 80 per cent. of the dope, and upon explosion was diffused as an impalpable powder. The dust formed from this source was 0.4 pound, or 2800 grains per hour, or 50 grains a minute. According to the absorbent used, we have then the following air-requirements per minute:

	Cubic feet of air with: Infusorial earth.		Wood-pulp, etc.
For carbonic acid, ...	9.50		9.50
For 50 grains of absorbent dust, .	25.00		9.50
Total cubic feet of air per minute, .	34.50	or	19.00

The local conditions affecting an explosion we will not discuss. But an allowance must be made for the dust derived from the rock after an explosion. This will vary with the character of the rock, and whether it is wet or dry. It cannot be even approximated. So for ordinary rock it would seem proper to multiply the above figures by 3 or 5, and, in cases where the material is very dry, to multiply by 6 or 10. Assuming the gases to be uniformly distributed during ten hours for each pound of 50 per cent. dynamite burned per shift, there should be allowed 100 cubic feet of fresh air per minute, and when the rock is very dry, 200 cubic feet per minute.

Dust Formed during Drilling.—Usually, in driving a heading, at least four holes are drilled, inclining upward, so that water cannot be poured into them. From these dry holes the dust is derived. The holes are 8 feet x $1\frac{3}{4}$ inches diameter; the contents, 231 cubic

inches, or 0.13 cubic foot = 22 pounds. Assuming that one-quarter of this amount is pounded to an impalpable powder, then during the seven hours of drilling 100 grains of dust are derived from each hole per minute. Allowing 0.5 cubic foot of air for each grain of dust, the air required per minute will be 50 cubic feet.

The probable quantity of dust a man inhales during a shift, working in an ordinary heading, may be stated as follows:

There is produced—

From 20 miners' lamps, say,	. . .	8 grains per minute.
" 2 pounds of dynamite,	. . .	350 "
" 4 dry-holes,	. . . .	400 "
Total dust per minute,	. . .	758

The working-space may be calculated as follows:

Heading area,	. . . .	20 sq. ft. × 50 = 1,000 cubic feet.
Enlarged section,	. . .	280 " × 50 = 14,000 "

or, in round numbers, 19,000 cubic feet. Hence each cubic foot of air-space contained about .04 or $\frac{1}{25}$th grain of dust.

Supposing a man at each breath to receive only half of a full draw, after ten hours he would have inhaled into his lungs nearly 15 grains of solid matter. Physiologists do not state how much a man can stand, but say that "the sub-pointed particles are drawn into the air-passages at each inhalation, and there find lodgment upon the delicate mucous surface with which they come in contact. The irritation there set up disturbs the working of the lungs, and, if maintained, eventually ends in organic disease."

Mr. C. L. Kalbfuss made the following experiments in the tunnel:

Into a beer-bottle, the bottom of which was broken off, he put a piece of sponge moistened with glycerine, about the size of a man's fist. Into the neck of the bottle he inserted the nozzle of an ordinary house bellows, each stroke of which delivered about a quarter of a cubic foot of air. After 24 strokes of the bellows, or 6 cubic feet of air, had been delivered, the sponge was completely blackened and impregnated with smoke and dust.

By the foregoing computations, based on the arbitrary assumption of uniformly active vitiating causes, we have found the quantity of fresh air necessary per minute to dilute the carbonic acid to the required standard, and to mitigate the presence of dust

Assuming a heading and bench to be worked together with lamps, and neglecting for a moment the explosions and the dust, we have the following air-requirements per minute:*

	In Heading. Cu. ft.	On Bench. Cu. ft.	Total Cu. ft.
17 men at 20 cubic feet of air. . . .	340	18 men, 360	700
1 mule at 92 cubic feet of air. . . .	92	92	184
	432	452	884
Deduct 3 drills at 45 cubic feet of air, .	135	1 drill, 45	180
Total, ...	297	407	704

It would be very convenient to assume that the gases produced by the explosions are distributed throughout the shift, and determine the air required, but unfortunately the smoke and dust formed are too variable to permit any estimates of practical value.

The blasting was generally done at the end of each shift, so that an hour intervened before the work was resumed. During this time the smoke filled the entire tunnel. Occasionally the compressed-air valve was opened, and the heading thus cleared in a short time. The persistence of the smoke in the tunnel depended on the atmospheric conditions. It often happened that on a rainy day, or after a rain, the tunnel was comparatively free, while on clear warm days it was so smoky that a lamp could scarcely be distinguished at a distance of 50 feet. Doubtless the moisture of the air was precipitated as fog, and much of the trouble attributed to the smoke was due to this cause.

Ordinarily, a tunnel 3000 feet long by 300 feet in cross-section would by natural ventilation become practically clear, so that a light could be plainly distinguished at a distance of 150 feet, about ten hours after a blast. The smoke may, therefore, be said to move at the rate of 5 feet a minute.

In artificial ventilation of mines the air-current is necessarily continuous, and the allowed velocity is from 2 to 6 feet per second; but as active ventilation in a tunnel is only required immediately after a blast, it would obviously be unnecessary to establish a plant of such power. It is believed that a fan capable of removing half of the smoke, and with it the dust of the tunnel, two hours after a blast, is all that is necessary. Under these conditions, with a tunnel of the above dimensions, the volume of air required to be removed per minute is 3800 cubic feet; and, as the natural ventilation

* For fuller information on this subject the reader is referred to André on "Coal Mining," and Angus Smith on "Air and Rain."

removes 1500 cubic feet per minute, the capacity of the fan should be 2300 cubic feet per minute.

The two blowers employed, when working at 145 revolutions per minute, and with all the connections in good order, were capable of doing the above work, or sufficient for one heading. But, as only one blower was usually provided for each shaft, the ventilating capacity was just half of what was required. According to the Miners' Act of Pennsylvania and the order of the Board of Health of the State of New York, 60 cubic feet of fresh air are required for each man. For mines this is barely sufficient; for a tunnel after the smoke is removed it is too liberal by one-half.

Analysis of the Air in the Tunnel.—The first apparatus used was Owen's carbonator. The principle of this apparatus is to use a variable quantity of air with a standard solution of hydrate of barium in connection with an indicator. The apparatus is fully described in the *Sanitary Engineer*, vol. ix., p. 479. On account of the poor and variable lights it was difficult to determine the change of color in the tunnel; each experiment consumed more than an hour, and the apparatus was too complicated to handle, so after a time it was discarded.

Pettenkofer's process was then used. The principle is that a known volume of air is made to act upon a definite quantity of standard baryta-water (standardized by oxalic acid solution) in such a manner that the carbonic acid is completely fixed by the baryta. The baryta-water is then poured out into a cylinder and allowed to deposit with exclusion of air; a part of the clear fluid is then removed, and the baryta remaining in solution is determined. The difference between the oxalic acid required for a certain quantity of baryta-water before and after the action of the air gives a basis for the calculation of the amount of carbonic acid present.*

The apparatus consisted of a Mohr's burette with float, a measuring-glass, some small flasks, and a gas-jar with a tight cover. An anatomical jar was used, the convexity of the cover was filled with cement, and to protect it from breakage and give facility of transportation, it was packed in an open wooden box with rope handles.

The analysis was conducted as follows: the jar filled with clean water was taken to the place in the tunnel where the air was to be examined, the water was emptied, the jar wiped out, 45 cubic centimeters of previously standardized baryta-water poured in, and the

* For details see Fresenius' *Quantitative Analysis.*

cap secured. The operation occupied but a few minutes. The jar was then brought to the office and the solution tested.

Table II.

Analyses of Air of the Tunnel Taken in the Headings.

No. of observation.	Date.	Outside. Deg. Fah.	Outside. Condition.	Shaft.	Dist. from Shaft.	Dist. from Heading.	Number. Lamps.	Number. Candles.	Number. Men.	Number. Drills.	Ther. Deg. Fah.	Blower. Revolut'ns.	Blower. Cubic feet.	Hours after blast.	Per cent. CO_2.	Condition of ventilation.	Remarks.
1	Dec. 18, '86.	30	Clear.	15N	2102						55	115	1222				Damper south closed.
2	"	30	"	" S	2156						55	115	498				Both open.
3	" 20, "	28	"	" N	2102						54	110	401				" "
4	"	30	"	" S	2156						55	110	496				" "
5	" 21, "	29	"	" N	2102						55	115	743				South closed.
6	" 22, "	32	"	" S	2173						58	115	897				North "
	Mean.....				2350							113	471				The blower 158 ft. above tunnel.
7	Feb. 25, '87.	30	Clear.	15 N	2618	60	11		12		3 55	100	440	2	.138	Very good.	Carbonator used.
8	Mch. 1, "	30	"	" N	2648	20	3		4		3 62	90		8	.165	" "	" "
9	" 7, "	32	"	" S	2994	80	4	11	16		3 60	150	584	36	.113	" bad.	" "
10	" 12, "	38	"	" N	2690	45	4		8		3 59	100	387	6	.142	Fair.	" "
11	" 14, "	41	"	" S	3111	90	10	4	16		3 58	100	535	30		Good.	" "
12	" 21, "			" S	2955							100	262				" broken during big run.
13	" 31, "	40	"	" S	3249	78	13		19		3 60	120	172	10	.126	Good.	Blow-pipe broken.
14	Apr. 12, "	40	"	" S	3331	80	5	10	20		3 60	100	148	6	.121	Very bad.	Carbonator used, blow-pipe broken.
15	June 17, "	91	Fair.	" N	2927	45					60	120		7	.124	Bad.	Pettenkofer used.
16	" 30, "	90	"	" N	3000	41	6		3 14		3 60	100		8	.182	Very bad.	" "
17	July 18, "	90	"	" N	3017	150					60	150		4	.159	Bad.	" "
	Average.....				2832	69	7	7	12		3 59	114	446	8	.141		
18	July 7, '87.	90	Sultry.	16N	3775	50	3		8	17	2 62	0		3	.394	Very bad.	Blower occasionally running.
19	" 19, "	90	Clear.	" N	3925	50	12		6	30	2 65	150		3	.471	" "	Blower occas'ly running dur'g big run.
20	" 22, "	90	"	" N	3845	50	12		5	30	2 63	175		2	.577	" "	"
21	July 20, "	90	Clear.	3S	2830	70	18		*5	37	3 62	0		6	.579	Very bad.	Shaft 350 feet deep, smoke-boxes used.

* Torches.

Referring to the above analyses taken in the heading of Shaft 15, which may be considered as representing a fair average of the southern portion of the work, it is seen that often, when the ventilation was apparently very bad, the amount of carbonic acid was very low, and *vice versa*.

With the admission of 450 feet of air per minute, or nearly twice the quantity required for the number of men present, the proportion of carbonic acid was diluted to 0.141 per cent, which is about two-and-one-half times better than the assumed standard.

At Shaft 16, with partial ventilation, the carbonic acid present was 0.394 per cent., or greater than the worst recorded condition of a smoking-car. At this shaft, when the blower was not running, and on the northern portion of the work, where natural ventilation was practically depended upon, the carbonic acid was 0.578—much worse than any smoking-car, but nearly a quarter less than the average composition of the air of mines in England. It is to be regretted that there was no opportunity to make more tests.

General Conclusions.—From the foregoing it is seen that the vitiation of the air due to the respiration of the men and the combustion of the lights could be reasonably estimated and provided for. The same applies to the gases from the dynamite, but these were accompanied by the smoke, which varied according to the quantity of wood-pulp or other carbonaceous matter used with the absorbent, and even according to the loading and firing of the charge. The dust, of course, was a variable quantity, depending on the character of the rock.

It is believed that a ventilating-power sufficient to remove half the volume of air contained in the tunnel within two hours after an explosion would have satisfied practical conditions. After the smoke was removed, 30 cubic feet per minute for each man employed was sufficient. Chemically considered, the air in the tunnel, though it could easily have been kept purer, was no worse than in some public schools, and was not so poisonous as some persons asserted. The real trouble was from the smoke and dust. When artificial ventilation was used, the shafts, with the exception of No. 16, were provided with only one blower each, or half the necessary means; and of this, through carelessness and indifference, only about 50 per cent. of the useful effect was utilized. The additional expenditure at each of these shafts of the comparatively insignificant sum of $1500, and the exercise of a little care and humanity, would have insured decent ventilation.

An Example of Slow Work—The Excavation of the Tunnel through the Soft Ground of Shaft 13 South Heading, Station 773 + 52.

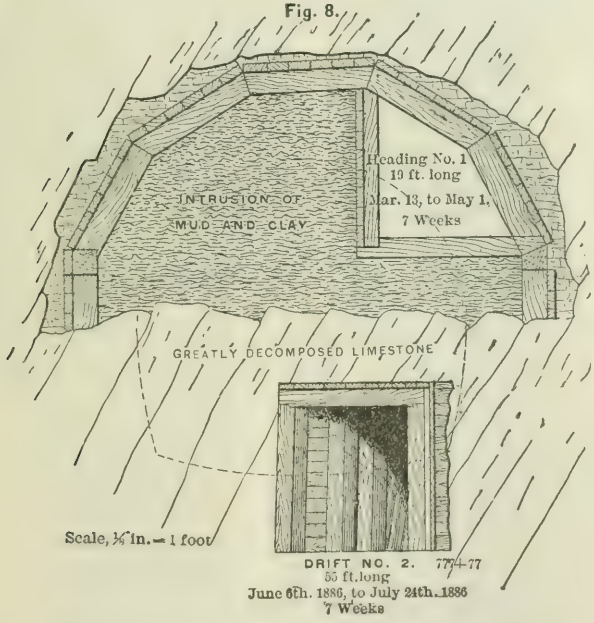
First or Development Stage.

On June 1st, 1885, the tunnel at the foot of the shaft was commenced, and advanced at the rate of 80 feet a month for 392 feet, through hard dolomitic limestone, which then became softer. Fif-

teen feet further, at station 777 + 59, on the east side of the heading, a fissure was cut on December 9th, and the trouble began. There poured out a mixture of decomposed limestone, clay, sand and dirty water, which soon partially filled the tunnel for 125 feet, and amounted to about 100 cubic yards. After three days the water became clear, the fissure was plugged with straw, and the heading advanced 20 feet further. Then, on December 22d, without warning, another outpour three times greater than the first occurred. The men were driven out, and everything in the heading for 50 feet was covered out of sight.

Timbering of the form shown in Fig. 8, allowing for 20-inch brickwork, was then commenced at the soft rock. At the end of two months the heading had been cautiously advanced 34 feet, to 778 + 13, and seventeen sets of timber placed, the last two being

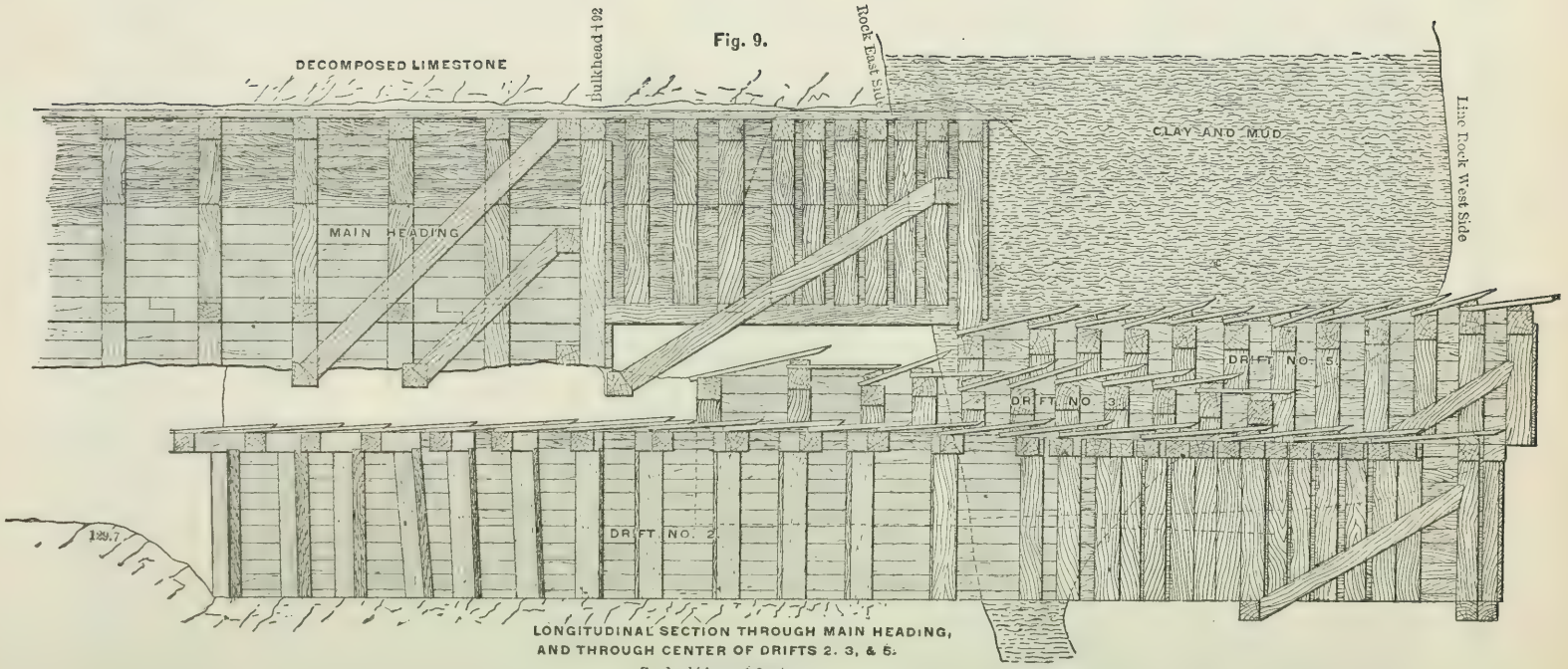
Fig. 8.



First Stage of Work in Bad Ground. Shaft 13, South Heading.

re-enforced with heavy hemlock spreaders. A bulk-head was then built at the end of the timbers, at 778 + 09, this being the edge of the soft ground on the east side. The bench during this time had been brought up, and was 32 feet behind, at 777 + 77.

Fig. 9.



Scale, ½ in. = 1 foot

Drifts and Timbering in Bad Ground. Shaft 13, South Heading.

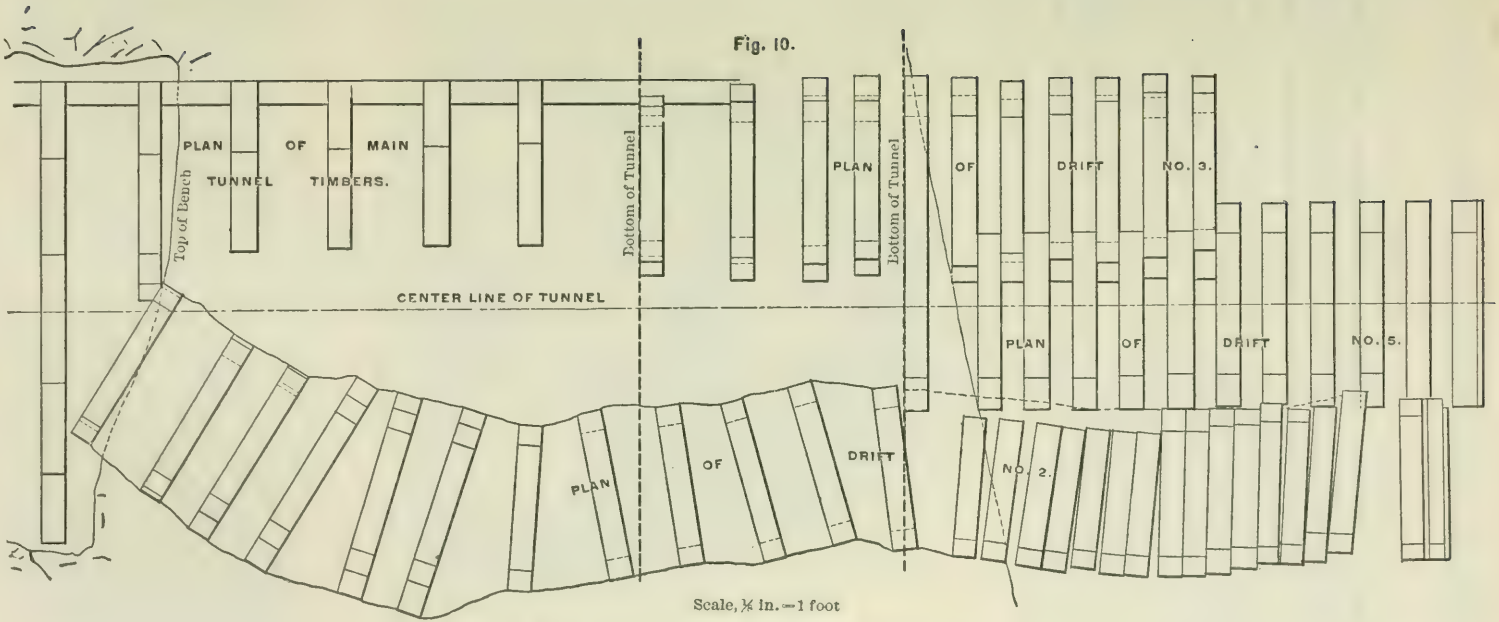
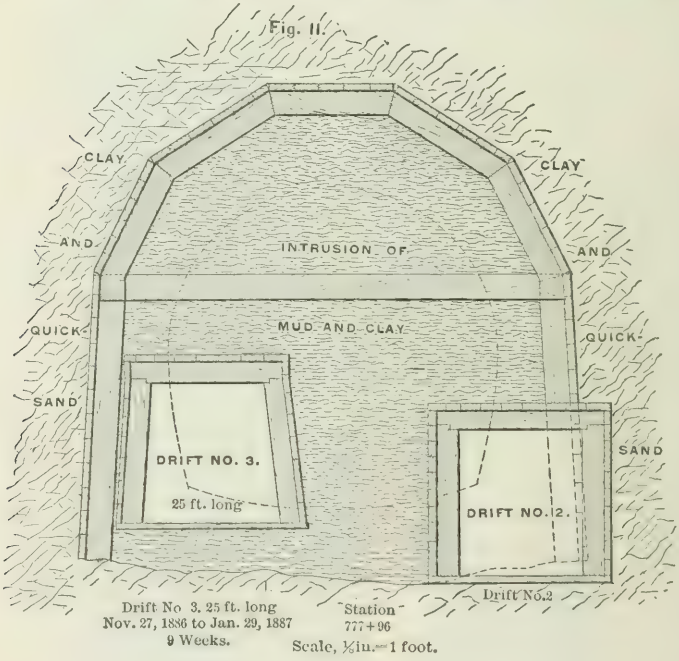


Fig. 10.
PLAN OF MAIN TUNNEL TIMBERS.
Top of Bench
Bottom of Tunnel
PLAN OF DRIFT NO. 3.
Bottom of Tunnel
CENTER LINE OF TUNNEL
PLAN OF DRIFT NO. 5.
PLAN OF DRIFT NO. 2.
Scale, ¼ in. = 1 foot

Plan of Fig. 9, Drifts and Timbering in Bad Ground.

Second or Experimental Stage.

For several days the clay and mud in the face of the heading were continually forced inwards. Some idea of the pressure may be formed from the fact that the 24-inch oak logs used as rakers at the bulkhead became so crushed after twenty-four hours that they had to be continually renewed. A small cave on the surface then occurred; finally, on February 20th, this was followed by a much larger one, about 25 feet east of the center-line. It was directly in line of a small stream which partly filled it with water. The stream was immediately diverted, and a pump rigged to remove the water from the hole. This cave was supposed to be the mouth of the original fissure, and an abortive attempt was made to clog it up by throw-



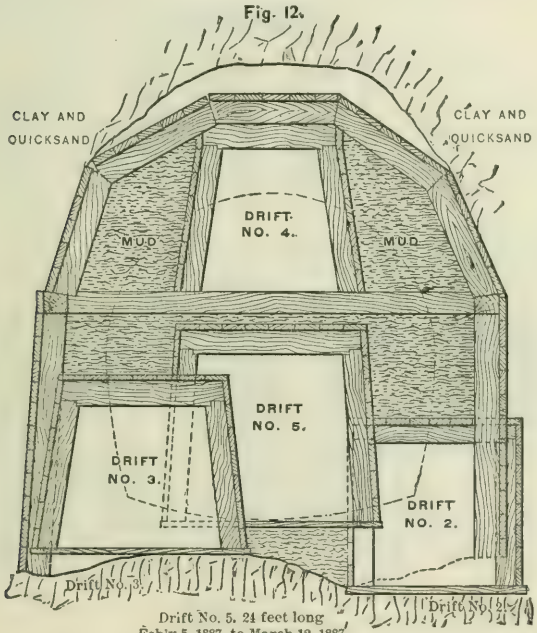
Section Showing Drift No. 3.

ing in bales of hay, cedar-brush, logs and stumps. Either from the fact that the course of the brook had not been properly diverted, or by reason of a sudden thaw, or local conditions, this débris gently glided down into the tunnel, completely filling it for 70 feet, and gradually sloping down to a depth of 1.5 feet for 312 feet, or to within 75 feet of the foot of the shaft. It amounted to about 1189 cubic yards.

The Crib.—Faith in the theory of a fissure in the limestone led

to the plan of a crib (Fig. 7, page 715). It was aimed to strike the fissure, which was to be arched over, or otherwise closed. The dimensions were 12 x 12 feet on the outside. It was built in the hole, and consisted of wall-plates of 12-inch timber jointed at the ends, and held apart by vertical posts, 2 feet 10 inches long, placed at the corners, the whole system being held together by planks spiked on the outside. It was sunk to a depth of 50 feet, when, hard-pan occurring on the west side, it began to sink irregularly. Sheet-piling was then driven on the east side, and finally, to keep the crib from toppling over, ropes were attached to the top and bottom sets, and secured to trees and posts on the surface.

Fig. 12.



Successive Drifts in Bad Ground, Shaft 13, South Heading.

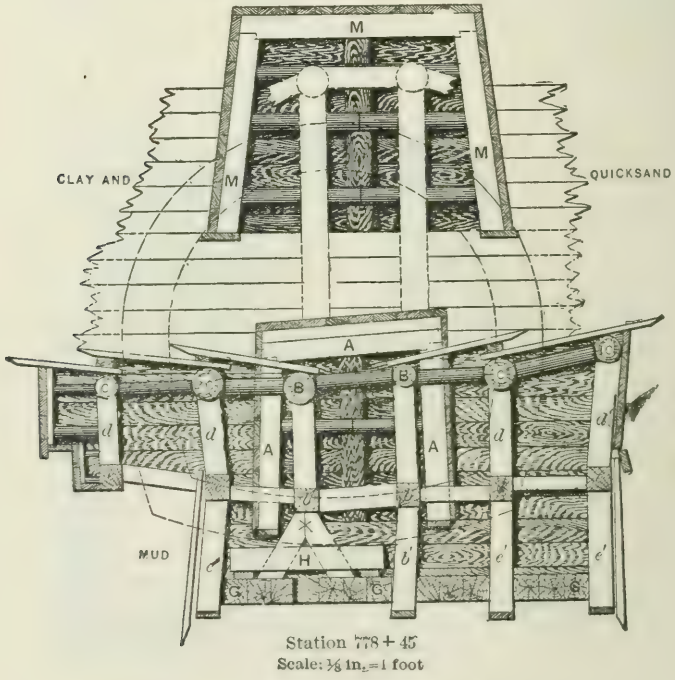
After three weeks' work and the expenditure of about $900 the contractors concluded that they had expended the amount allowed "for any work they should do on the surface, to be paid after the tunnel was through," and stopped.

At this time the diamond-drill was brought up, and hole No. 46, station 778 + 87.33 east of center-line, was sunk 191 feet; 79 feet through gravel and boulders, and 112 feet in one day through clay

and mud. By reason of these developments the work was ordered to be stopped. Two years later, the hole was filled up with 2533 cubic yards of earth; the contract-price of $5000 had to be paid, and, to the relief of all concerned, the crib was buried out of sight.

At this time several plans were proposed. Among the most feasible were the following: Mr. Charles M. Kalmbach, a well-known tunnel-expert, proposed to drive a heading below sub-grade and fill with concrete; then gradually to widen out, and thus obtain a foundation. The division engineer in charge, Mr. Alfred Craven, pro-

Fig. 13.



Successful advance of Drift by English Method.

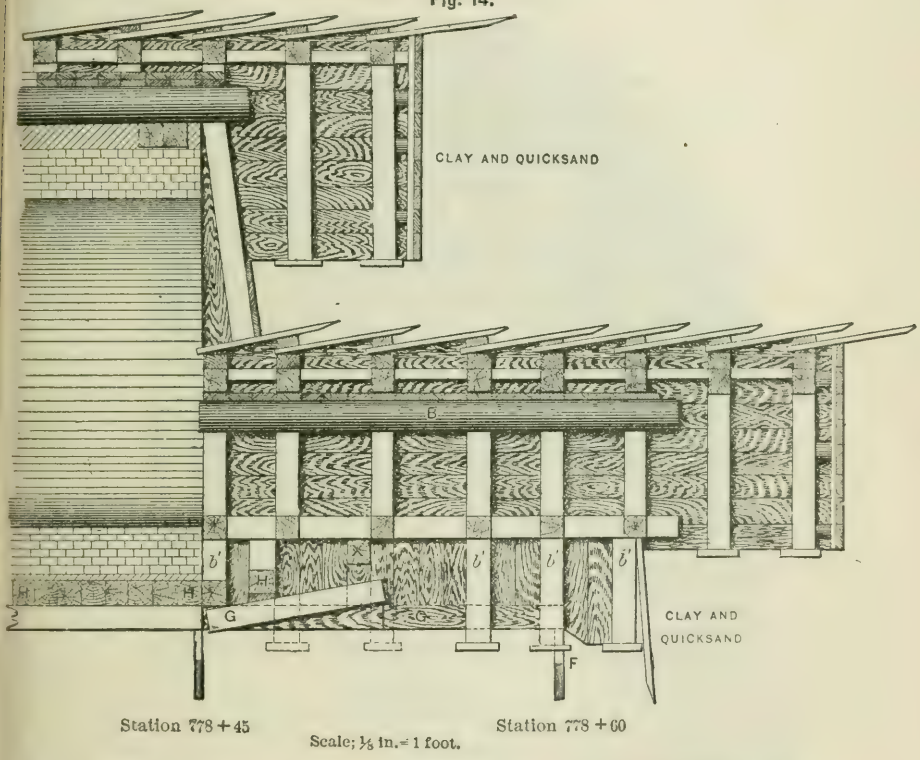
posed instead, a timber platform of oak or Georgia pine, upon which the concrete should be laid, and to change the section from the horseshoe to the circle. This was generally conceded to be right, but in the following August it was decided to use white oak for the platform, and return to the horseshoe-section; and this plan was formally approved January 10, 1887, but not fully adopted by the contractors until the year after. The functions of the engineers were confined to giving lines and grades, and suggesting plans to

the contractors, who executed the details of the work in their own way, and assumed the responsibility of any miscarriage.

For the next year the work was carried on with a series of experimental drifts, conforming to no regular system of timbering, and the means adopted were entirely inadequate to the work in hand.

The First Drift.—This was started at 777 + 77, and in two weeks progressed 18 feet. It was driven within the original heading, on the west side. It advanced 20 feet, when the sand rushed in through

Fig. 14.



Progressive Timbering by English Method. Longitudinal Section of Fig. 15.

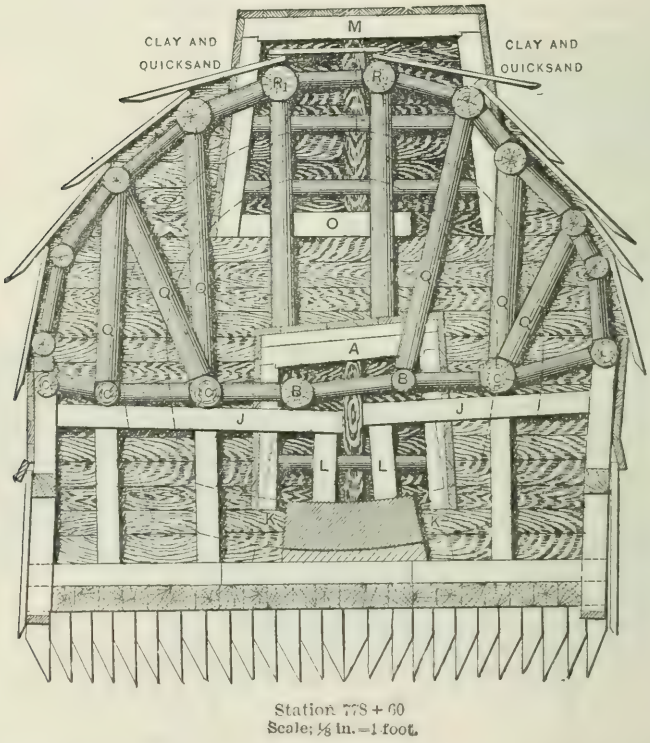
a 12-inch opening, and filled it up for 12 feet. A bulkhead was then built across the entire heading, and the mud removed, which occupied three weeks longer. (Figs. 8, 9 and 10, pages 733, 734 and 735.)

About May 1st, Mr. Charles Sooysmith visited the tunnel with a view to trying the freezing-process, and it is reported that he offered to contract to drive the tunnel for $1000 a foot.

The next experiment was to drive eight 2-inch pipes with perforated ends 30 feet into the sand and mud, in order to drain it. This was not a success.

The Second Drift.—Forty feet north a sump was sunk 8 feet deep on the east side, and a ditch carried 4½ feet below invert-grade, to the bench at + 77. Here a bottom drift was started, on the west

Fig. 15.



Station 77S + 60
Scale; ⅛ in. = 1 foot.

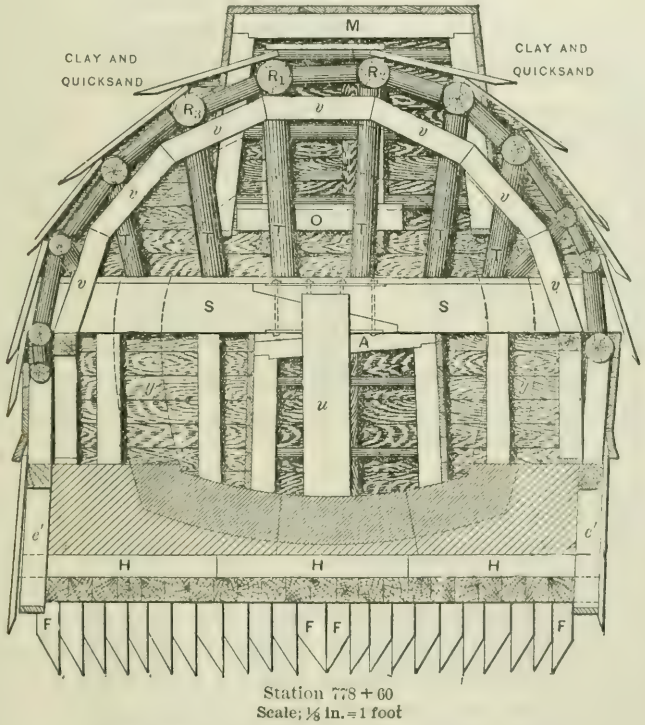
Progressive Timbering by English Method. Cross-section through Fig. 14.

side. It was 5 feet in the clear, and was timbered with square sets placed 2 to 3 feet apart; nineteen regular sets and ten intermediate sets were put in. It was 54 feet long, 35 feet through decomposed rock, the remaining 19 feet through heavy ground which caused the timbers to crack. Finally, mud ran in as fast as it could be excavated, so that the drift was abandoned after eleven weeks' work. Figs. 8, 9 and 10, pages 733, 734 and 735, show it in section, elevation and plan.

The great trouble was to drain this ground. The water amounted to 160 gallons per minute. After ten months it had not sensibly diminished, and, in fact, it never did diminish. At first sight, one might suggest driving a drift to the Saw Mill river, but unfortunately the bed of the river was higher than the grade of the tunnel.

There was some talk of diverting the line of the tunnel to the valley on the west, and making an open cut; and with this view a topographical survey was made of the ground between Shafts 13 and 14. This was found to be impracticable.

Fig. 16.



Station 778 + 60
Scale; ⅛ in. = 1 foot

Completion of Timbering and Setting of Invert.

The work was then abandoned for ten weeks. Then the excavation was resumed and the masonry commenced. After seven weeks the tunnel was sufficiently widened, and on 4th December the arch reached 777 + 95, just 18 feet short of where the heading had been nearly a year before.

During the next four months, or until April 2d, 1887, the following drifts were run (Figs. 9, 10, 11 and 12):

Drift No. 3.—East side at grade, nine weeks, 27 feet to + 22.

Drift No. 4.—Top-center, one week, 8 feet to + 13.

Drift No. 5.—Center at grade, 6 weeks, 24 feet to + 32.

These drifts were no more successful than the others; however, they were widened out, the arch was carried to 778 + 05, and the invert and side walls to + 08, or the edge of the soft ground. The extent to which it had been honeycombed by successive drifts, is shown in Figs. 9, 10 and 12.

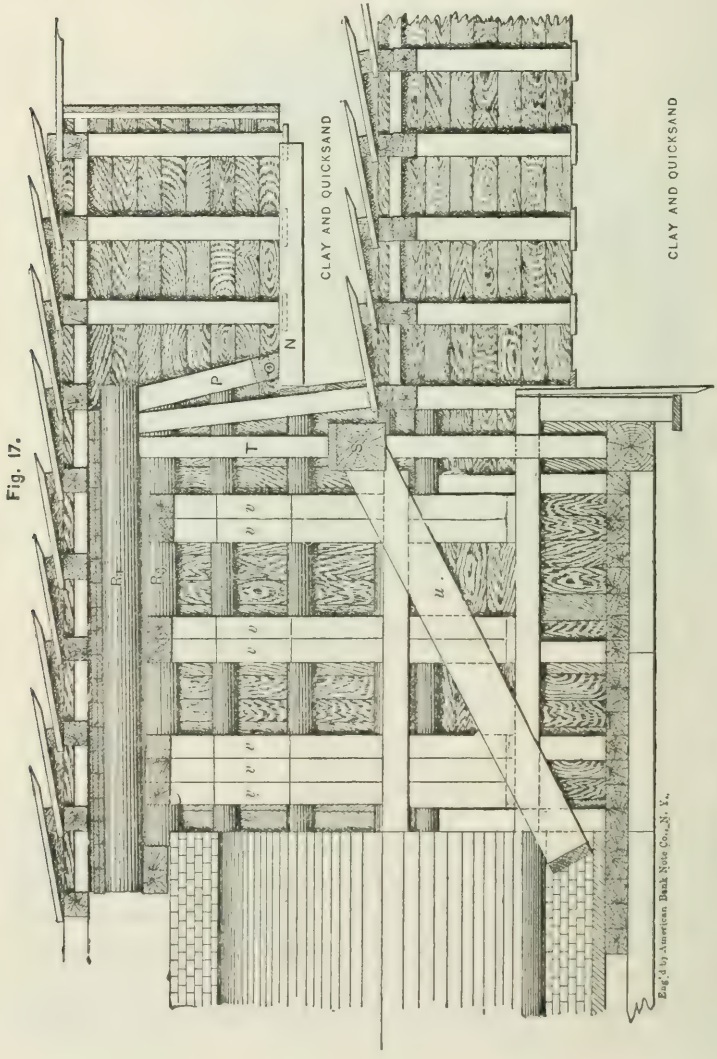


Fig. 17. Completion of Timbering and Setting of Invert. Longitudinal Section through Fig. 16.

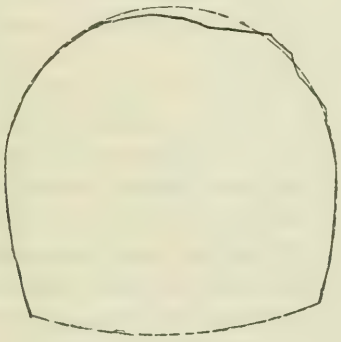
Third or Systematic Stage.

In March, there occurred an entire change in the management, and on April 2d, 1887, M. Nolan was appointed foreman in charge. The English system was introduced, and the work was completed, without serious interruption, in a year and two months.

The entire excavation made was allowed within the limit of 676 square feet, or 26 x 26 feet. This permitted the placing of a 2-foot timber platform and 24 inches of brick masonry. The average area excavated was 507.7 square feet.

The first two stretches of 13 and 12 feet occupied respectively twenty and twelve weeks. Against advice, a top-central drift was first run about 18 feet, and, as the rock was found on the west side, the widening was done from west to east; about 15 crown-bars were

Fig. 18.



Crushing of the Arch.
Station 778 + 16
Scale; ⅛ in.=1 foot.

placed, and a bulkhead built. The bench was then removed to 4.2 feet below grade. A platform 2 feet thick, formed of cross and longitudinal timbers 12 x 12 inches, was built. Upon this the invert was laid, and the masonry completed.

In widening out the second stretch, several of the crown-bars on the east side broke, and the weight of the ends resting on the masonry arch at + 18 caused it to crack for 8 feet. This crack extended irregularly down the east haunch and side-wall. Fig. 18 shows the extent of this defect. This difficulty was overcome by supporting the arch with oak centering, and also by placing a large 24-inch longitudinal timber in the center, supported by posts about 6 feet long from the invert and mud-sills. From this, radial pieces were placed to the crown-bars above. Finally, the bottom

was removed and platform placed. The cracked portion of the arch was subsequently rebuilt.

Top-drifts were still persisted in, both here and in the heading from Shaft 13 A, which had now reached the other end of the soft ground; but after two weeks' further trial the foreman was finally forced by the mud, sand and water to abandon them.

As already observed, the English system was finally adopted, and the following is a description of its application:

The beginning was made 4 feet north of the soft ground.

The first 54 feet (778 + 05 to 778 + 45, 40 feet, and 779 + 19 to 779 + 05, 14 feet), as before mentioned, were worked from the top, down; the remaining 60 feet from the bottom up.

Referring to Shaft 13: on January 28th, 1888, the excavation and masonry had reached 778 + 45, and a small preliminary bottom drift, 2 feet by 3 feet, was driven through, connecting both headings, and the top drift from the driving of the previous section, was at Station 778 + 55.

The First Operation.—The lower drift was enlarged to 6 feet by 8 feet for a distance of 25 feet. (A, Fig. 13, page 738.)

The Second Operation.—Widening out of the drift. Bearing-bars, B (Fig. 13), 20 feet long, were placed under the caps, and were supported by posts, resting on longitudinal sills, *bb*. The operation of widening out, on either side as most convenient, was then done, and the bars, posts and sills (*c, c, c, d, d, d* and *e, e, e*) placed. The process of widening is shown on the left of the drawing, Fig. 13.

The Third Operation.—The placing of the timber platform. The previous operation being well advanced or completed, sheet-piling was driven down, 5 or 6 feet outside of the sills *b, b, e, e*, etc., and the enclosed spaces were excavated in pockets to a depth of 4.5 feet below invert-grade, for a length of 18 feet, and as wide as possible, generally 10 or 15 feet. Posts *b'*, *e'*, etc., placed at intervals measuring 7 feet, Figs. 13 and 14, resting on foot-boards in the mud, now support the system. Sheet-piling (F, Fig. 14), 3 or 4 feet long, was then driven across the bottom of the excavation about 15 feet in advance of the completed invert. This prevented the mud from being pumped out from beneath the platform during the construction of the succeeding section. The longitudinal sills, G, in variable lengths up to 15 feet, were then placed in the excavated spaces between the posts, *b' b' b' e'*, etc. Underpinning was then employed in order to get the longitudinal sill in the space interrupted by the line of posts. Immediately behind the first post, a cross-sill (H,

Fig. 13) 6 feet long was placed on blocks 6 inches above the sills in place. The weight was transferred to this and the first post removed. The second post was removed in a similar way. The third post was supported by struts (X, Figs. 13 and 14). The sill, G, in the vacant space was in half-lengths and was slipped into place as shown in Fig. 14. The cross-sills, H, were then placed. The sills put in were as long as circumstances would allow, and arranged to break joints. The posts, *e'* (Fig. 16), on the outside, could seldom be removed; and longitudinal pieces were placed between them. The excavated space ahead of the platform was used as a sump for pumping.

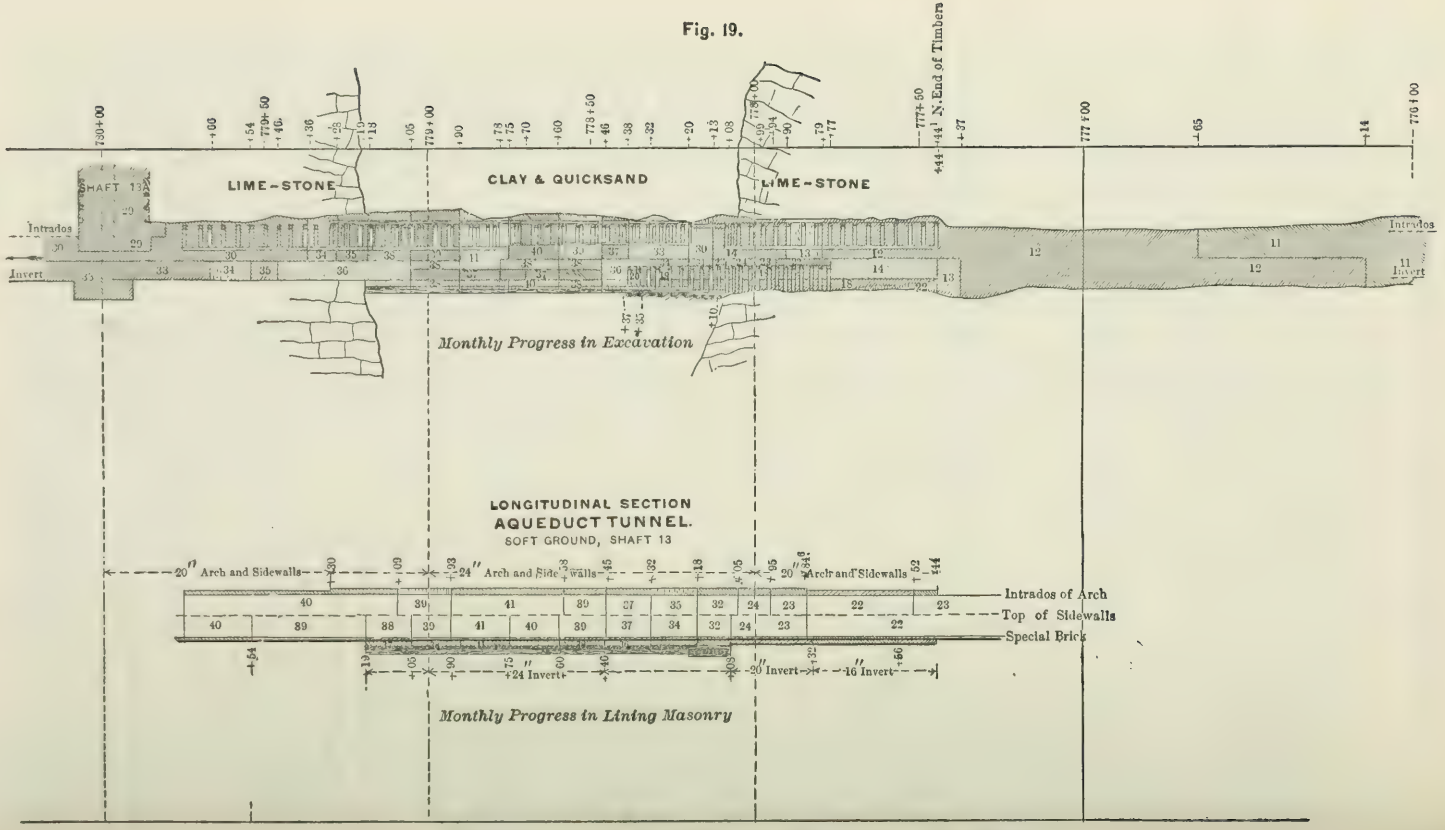
The Fourth Operation.—The platform being completed, the cross-beams, J (Fig. 15), supported by the inclined posts, *k*, *k*, were placed about 3 feet between centers under the bearing-bars, B and C, and well wedged up. The posts and sills, *b*, *b*, *d*, *e*, and *e'* were removed, and the central portion of the invert built between K K, which were replaced by L L (Fig. 15), and the remainder of the invert and 2 feet of side-wall and backing was built (Fig. 16, page 741).

While the masonry was being built, the top-heading, M (Fig. 15), was advanced 15 feet to + 70.

The Fifth Operation.—The crown-bars, R_1 R_2, etc. (Figs. 15 and 16), were placed. The back ends rested on the arch already built, and the heading-ends of the first two were supported by the posts, P (Fig. 17), resting on the cross-sill, O, and two longitudinal sills, N, placed in the top-heading (Figs. 15, 16 and 17). The excavation was carried down and out at the same time, and the other bars were supported by the struts, Q, resting on the bearing-bars, B, C, etc. (Fig. 15). The large beam, S (Figs. 16 and 17), 24 inches by 24 inches, 26 feet long, was then placed ahead of the invert, and supported by posts resting on sills in the bottom, and strutted back to the invert by the rakers, *u*. Posts, T, were placed under the crown-bars and set upon this sill, which also served as a bearing for the bulkhead against the mud in the face. When the section was completed, the sill was cut off and removed, the ends being built in. To support and strengthen the crown-bars, the segmental timbers, *v*, supported by posts resting on foot-blocks on the masonry already built, were put in, one or two feet apart as necessary. The sidewalls and the arch were finally constructed. No attempt was made to withdraw the crown-bars.

This was the general order of operations which were varied in their successive stages according to circumstances.

Fig. 19.



Sections showing Rate of Progress, compiled by R. Turnbull.

Fig. 20.

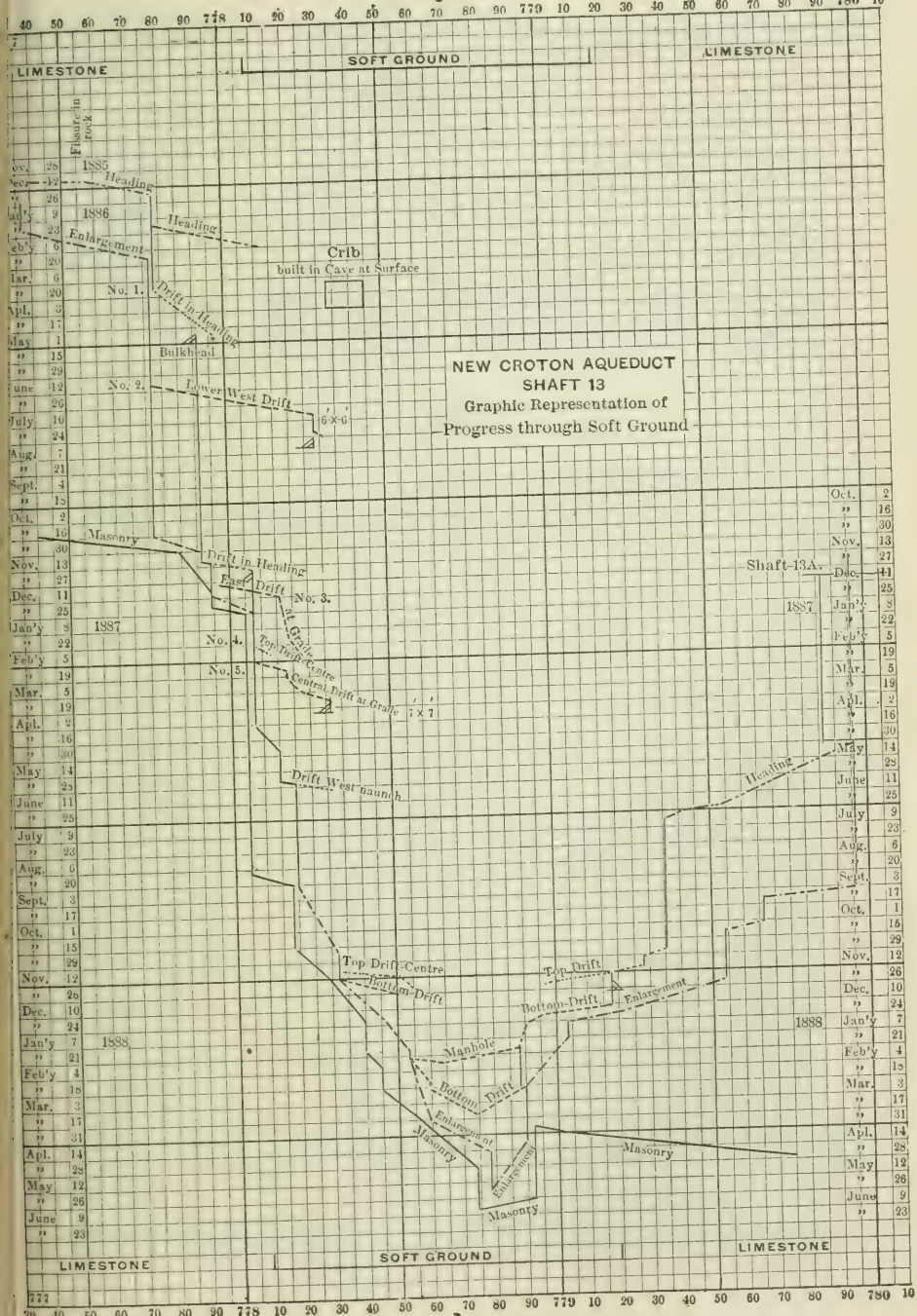

NEW CROTON AQUEDUCT
SHAFT 13
Graphic Representation of
Progress through Soft Ground
LIMESTONE
SOFT GROUND
LIMESTONE
Fissure in rock
1885
Heading
1886
Heading
Enlargement
Crib
built in Cave at Surface
No. 1.
Drift in Heading
Bulkhead
No. 2.
Lower West Drift
6'×6'
Masonry
Drift in Heading
East Drift
No. 3.
1st Grade
1887
No. 4.
Top Drift Centre
No. 5.
Central Drift at Grade
7'×7'
Drift West haunch
Shaft 13A.
1887
Heading
Top Drift Centre
Bottom Drift
Top Drift
Bottom Drift
Enlargement
1888
Manhole
Bottom Drift
Enlargement
Masonry
Masonry
Masonry
LIMESTONE
SOFT GROUND

Table III.

Showing in Detail the Time Occupied by the Different Operations, as Above Described.

Date.	Weeks.	Work Below Spring-Line.	Work Above Spring-Line.	Masonry.
1888.				
Jan. 28...	....	Drift 2 ft. x 3 ft., 778+75	Heading .....778+55	..... 778+45
Feb. 4...	1	Drift widened, 6 ft. x 8 ft., 778+70	....	.....
" 18...	2	Platform built, 778+60	.....	.....
" 25...	1	.....	Drift.....778+62	Invert completed...778+60
Mar. 3...	.....	.....	" .....778+70	....
" 3...	1	.....	" widened ...778+50	.....
" 17...	2	.....	" " & comp.778+60	....
" 24...	1	.....	....	Sidewalls and arch 778+60
	8			
" 31...	1	Widening drift.....	Drift.....778+75	.....
Apl. 14...	2	Platf'm finish'd, +75	Widening out.....	Center of invert built.....
" 21...	1	.....	" "	Inv't & 2 ft. S. wall.778+75
" 28...	1	.....	Cr'n bars placed, " +78	.....
May 5...	1	.....	.....	Arch & S. wall comp...+75
	6			

There is no published record of a tunnel-foundation having been built as described above.

At first some difficulty was caused by carelessly removing the sheeting in front of the invert and allowing some of the material beneath to be pumped out, thus undermining it. Some of the crown-bars on the east side were broken, and some placed too low. They were cut out and replaced by segmental arches of oak, and finally these arches were always used to relieve the weight of the crown-bars on the brick arch; and as a precautionary measure, the last five stretches, one of them in 13 A, were built in this manner. The heading being opened through to 13 A for four months, obviously drained and facilitated the latter portion of the work.

The last stretch was finished at 6 P.M., Friday, June 1, 1888.

According to a most liberal estimate, including the cost of sinking Shaft 13 A, the cost for 110 feet was at the rate of $416.00 per foot. The amount paid for this work was at the rate of $539.00 per foot.

Fig. 19 shows a general longitudinal section, and the progress of the work; and Fig. 20 is a graphic record.

Table IV.

Progress through Soft Ground, Shaft 13, South Heading. (*Sta.* 773 + 52.)

Date.	Weeks worked. No Progress.	Weeks worked. Progress.	Weeks worked. Total.	Headings. Sta.	Lineal feet. Progress.	Lineal feet. Dist. from Shaft.	Bench. Sta.	Bench. Progress.	Remarks.
FIRST, OR DEVELOPMENT STAGE.									
Dec. 9, '85	.....	.....	.....	777+44	2	392	.....	.....	Soft limestone begins.
" 12, '85	.....	.....	.....	+59	15	407	.....	.....	Fissure 2 feet wide on east side.
" 19, '85	.....	1	1	+66	7	414	776+91	.....	
Feb. 13, '86	4	4	8	+79	13	427	777+37	.....	Outpour of mud and water. Men driven out.
Totals.....	4	5	9	778+13	34	461	777+77	.....	Timbered from 777 + 44 to end of heading.
SECOND, OR EXPERIMENTAL STAGE.									
Feb. 13, '86									
Apl. 17, '86	9	.....	9	777+77	0	.....	.....	.....	Building crib and removing mud.
May 1, '86	.....	2	2	777+95	18	.....	.....	.....	Drift No. 1. Top drift west side. McLear, Supt.
" 29, '86	3	.....	3	.....	.....	.....	.....	.....	Driven out, and bulkhead built at + 92. Cleaning out mud and trying driven wells.
Aug. 7, '86	.....	11	11	778+31	54	.....	Arch.	.....	Drift No. 2. Lower west drift. Abandoned.
Oct. 16, '86	10	.....	10	.....	0	.....	777+45	0	No work of excavation. Masonry commenced.
Dec. 4, '86	.....	7	7	.....	.....	443	+95	50	Bulkhead removed and arch extended.
				777+95	.....				
Jan. 29, '87	.....	7	7	778+22	27	.....	778+05	10	Drift No. 3. East side at grade. Abandoned.
" 22, '87	.....	.....	.....	778+05	.....	453	.....	.....	Drift No. 4. Top central.
" 29, '87	.....	.....	.....	778+13	8	461	.....	.....	Abandoned.
Feb. 1, '87	.....	.....	.....	.....	.....	.....	.....	.....	Maddox took charge.
" 12, '87	2	.....	2	.....	.....	.....	.....	.....	Cleaning out.
Mar. 19, '87	6	.....	6	.....	24	.....	.....	.....	Drift No. 5 too high, and no benefit. Abandoned.
Apr. 2, '87	2	.....	2	.....	.....	.....	.....	.....	
Totals,	32	27	59						
THIRD, OR SYSTEMATIC STAGE.									
									Foreman Nolan took charge.
Apr. 2, '87	.....	.....	.....	778+05	.....	453			..... Invert was 3 feet in advance.
Aug. 20, '87	.....	.....	20	+18	13	466			..... Top heading driven fir t.
Oct. 12, '87	.....	.....	12	+30	12	478			..... " " "
									Bottom heading driven first.
Jan. 28, '88	.....	.....	11	+45	15	493			..... Small drift (2 ft. x 3 ft.) connects shafts.
Mar. 24, '88	.....	.....	8	+60	15	508			..... Bottom heading driven first.
May 5, '88	.....	.....	5	+75	15	523			..... " " " "
June 1, '88	.....	.....	4	+90	15	538			..... " " " "
Totals... ..	.....	.....	60		85				Average progress per week, 1.4 ft.
Dec. 4, '86	.....	.....	.....	.....	.....	.....			..... Commenced to sink shaft 13A. (Sta. 779 + 96).
" 17, '87	.....	...	.....	779+19	.....	77			.....
Feb. 25, '88	.....	10	10	+05	14	91			..... Top heading driven.
Mar. 24, '88	.....	4	4	778+90	15	106			..... Bottom heading driven.
Totals... ..	.....	.....	14		114				Average progress per week, 2.07 ft.

Masonry and excavation Progressed simultaneously.

From the foregoing table we see that the first or development stage, December 9th, 1885, to February 13th, 1886, occupied 9 weeks; the second or experimental stage, February 13th, 1886, to April 2d, 1887, 59 weeks; the third or systematic stage, April 2d, 1887, to June 1st, 1888, 60 weeks, for 14 weeks of which the work was prosecuted at both ends.

Notes on the Time and Cost of Drilling through Different Materials with the Diamond Drill.

This work was done for the purpose of exploration on the line, between Shafts 13 and 14. The materials passed through were clay, gravel, boulders, decomposed and hard gneiss and limestone.

There were fourteen different holes bored, the aggregate depth of which was 2084 feet. A daily record was made of the material passed through, as shown by the following table:

TABLE V.

No. of hole.	Date.	Foreman.	Men.	Clay & Gravel.		Boulders.		Soft Rock.		Hard Rock.		Days Moving.	Days Accidents.	Total feet.	Total days.
				Feet.	Days.	Feet.	Days.	Feet.	Days.	Feet.	Days.				
46	March 23–27	1	4	29	1	21	2	141	2	.....	.....	.....	.....	191	5
47	" 29 to April 5	1	4	18	1	26	2	151	4	.....	.....	.....	.....	195	7
48	April 6–23	1	4	29	1	21	2	23	2	101	7	1	3	174	16
49	" 26 to May 3	1	4	33	1	48	2	79	3	.....	.....	1	1	160	8
50	May 6–15	1	4	14	1	18	1	28	1	77	6	2	1	137	12
51	" 19 to June 5	1	4	44	1	33	3	22	1	36	3	2	5	135	15
52	June 6–30	1	4	24	1	65	6	.....	.....	10	1	1	13	99	22*
53	July 1–23	1	4	22	1	77	12	55	6	.....	.....	.....	.....	154	19*
54	July 27 to Aug. 11	1	4	24	1	59	11	73	1	.....	.....	2	.....	156	15
	Aug. 12–19 (7 days)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
55	Aug. 20 to Sept. 10	1	5	19	1	71	8	47	1	6	1	.....	8	143	19
56	Sept. 13–21	1	5	34	1	40	3	98	4	.....	.....	1	.....	172	9
54	" 22 to Oct. 9	1	5	.....	.....	.....	.....	13	1	1	1	1	13	14	16
57	Oct. 11–30	1	5	27	1	70	7	.....	.....	52	5	1	4	149	18
	Nov. 1–12 (11 days)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
58	Nov. 13–29	1	5	16	1	23	4	47	1	53	4	2	2	139	14
59	Nov. 30 to Dec. 9	1	4	18	1	.....	.....	37	2	11	1	1	4	66	9
	Totals			351	14	572	63	814	29	347	29	15	54	2084	204

* Water short.

The work was commenced March 23d, 1886, and continued until December 19th, 1886. This time was distributed as follows:

	Days.	
Actually working,	189	
Moving drill,	15	
		204
Idle,		18
Holidays,		2
Sundays,		37
Total,		261

Table VI.

Compiled from the Daily Reports—the Time Occupied by the Different Operations being Actually or Proportionally Charged to the Quantity of Material passed through.

Material.	Work performed.				Time consumed, days.					Av. no. ft. per day.	
	Holes.			Per cent. of Material.	Moving.	Drilling.	Casing, etc.	Total.	Per cent.	Drilling Alone.	Drilling and Delays.
	Number.	Average Depth.	Total ft.								
Clay and gravel	.....	.....	351	17	.....	14	.....	.....	.....	25	.....
Gravel and boulders	.....	.....	572	27	6.5	63	54	137.5	68	9	6.7
Decomposed gneiss	.....	.....	814	39	6.0	29	.....	35.0	17	28	23.1
Hard gneiss	.....	.....	347	17	2.5	29	.....	31.5	15	12	11.1
	14	149	2084	100	15.0	135	54	204.0	100	.....	10.2

From the above the following is deduced:

	Daily Progress. Ratio.	Feet.
Hard gneiss (taken as standard unit),	1	11 to 12
Decomposed gneiss,	2 to 2.3	23.1 to 28
Clay, gravel, and boulders,	0.6 to 0.75	6.7 to 9
Clay and gravel,	2.1	25

COST PER FOOT.

Labor, Teams and Coal.			
1 foreman,	243 days, 8.1 months, at $125,	$1,012 50	
1 assist. foreman,	" " " " " 70,	567 00	
4 men,	" " " " " 65,	2,206 00	
Teams, etc., moving,		80 00	
			$3,865 50
Coal, 189 days drilling and casing, 66.72 tons,			359 50
Supplies and Repairs,			
Am. Diamond Rock Boring Co.,		471 75	
Foundry,		291 50	
Lumber,		22 50	
Rope,		22 50	
Lard oil,		8 05	
			816 30
Interest on plant, $6,000 at 12 per cent., 8.1 months,			486 00
Total,			5,527 30
Renewing diamonds,		250 00	
Diamond drill-bit lost,		300 00	
			550 00
Total, 204 days in use,			$6,077 30
Average per day,			29 79
Average per foot,			2 91

Table VII.

Cost per Foot of Drilling in Different Materials.

Material.			Total cost.		No. of feet.	Cost per foot.
Clay, gravel, and boulders,	68%	× 5,527.30 =	$3,758.56		÷ 923 =	$4.07
Decomposed gneiss,	17%	× 5,527.30 =	939.64		÷ 814 =	1.15
Hard gneiss and limestone,	15%	× 5,527.30 =	829.10	$1,379.10	÷ 347 =	3.97
Diamonds and bit lost,			550.00			
			$6,077.30			

The items that make up the cost per foot may be divided as follows:

	Per cent.
Labor,	70
Coal,	7
Repairs and supplies,	15
Interest,	8
	100

As, in general practice, the holes would be probably much deeper, and the cost of labor about 20 per cent. less, the above figures may be accepted as having a liberal margin of excess.

RECAPITULATION.

Taking hard gneiss as a standard, calling it unity, we have:

	Daily Progress.	Cost per foot.
Hard gneiss,	1 = 11 to 12 ft.	1 = $3.97
Decomposed gneiss,	2 to 2.3 = 23.1 to 28. ft.	0.3 = 1.15
Clay, gravel and boulders,	0.6 to 0.75 = 6.7 to 9 ft.	1.03 = 4.07
Average,		$3.06 per foot.

The Big Run in Shaft 15, South Heading.

In the north heading, during 9 weeks previous to this run, 567 feet were made, and some timbering was done also.

The south heading, during 10 weeks, had been advanced 730 feet, and runs of 90, 83 and 82 feet per week were made. The subsequent removal of the bench for this distance required 21 weeks. The rock was a hard, compact gneiss, with close seams and occasional streaks of quartz; the general stratification was along the axis of the tunnel. The men of their own accord decided to break the record.

No apparent preparation was made beyond obtaining a larger supply of powder. The work in the north heading had been discontinued two weeks before. The point selected for the trial was 3193 feet from the shaft and about 3400 feet from the compressor.

The plant consisted of one duplex Rand compressor, Class "B," supposed to deliver 1325 cubic feet per minute, at 80 pounds pressure. The air was conveyed through 5-inch and 4-inch pipe; the loss of pressure due to leaks, friction, etc., was about 15 pounds. Three Slugger drills were used.

The work was carried on continuously in nominally 10-hour shifts, with the same gang of men. One shift was lost by the men being disabled by powder-smoke.

Ordinarily the work of a 10-hour shift was divided as follows:

Mucking out, 7 to 9.30 A.M.,	2½ hours.
Drilling from 9.30 to 4.30 P.M.,	6 "
Charging holes from 4.30 to 5 P.M.,	½ hour.
Firing from 5 to 6 P.M.,	1 "
	10 hours.

The following is the record:

1887, March 19th to 26th. Station 922 + 44 to 923 + 46.1 = 102.1 feet; area, 145.5 square feet, making the total excavation 550 cubic yards. Number of drills, 3 Sluggers.

	Feet.
Depth of holes, 8 center-cut, 10 feet deep,	80
12 side-cut, 8 feet deep,	96
Total feet drilled for each blast,	176

Number of blasts,	13
Total feet drilled in all,	2288
Feet drilled per cubic yard,	4.16
Pounds of powder used,	2200
Pounds of powder used per hole,	8.46
Pounds of powder used per cubic yard,	4
Average advance per blast,	7.86

About 25 per cent. of the muck was left in the tunnel.

The credit of organizing and executing this work is due to the men themselves, who received no extra pay. The contractors claim to have lost money by the operation. The following is a statement of cost:

Average Cost of the Driving.

Inside labor:	Per shift.	Total for 13 shifts.	Per cubic yard.
Drilling and blasting:			
1 heading-boss,	$3.25		
3 drillers,	7.50		
3 helpers,	5.25		
1 nipper,	1.00		
1 powderman,	1.00		
	$18.00	$234.00	$0.42
Mucking and handling:			
1 muck-boss,	$2.50		
9 muckers,	13.50		
	16.00	208.00	0.50
Carried forward,		$442.00	$0.92

	Per shift.	Total for 13 shifts.	Per cubic yard.
Brought forward,		$442.00	$0.92
Transportation:			
3 drivers and mules,	6.75		
1 trackman,	1.50		
2 sumpmen,	1.50		
	9.75	126.75	0.31
Outside Labor:			
Shaft:			
2 bellmen,	1.50		
2 topmen,	1.50		
	3.00	39.00	0.09
Power, etc.:			
1 engineer, per week, $15.00			
1 engine-driver, " 9.00			
1 fireman, " 12.25		36.25	0.07
1 machinist, half-week, 7.00			
1 blacksmith, per week, 15.00			
1 blacksmith's helper, " 10.00			
1 carpenter, half-week, 7.50			
1 time-keeper, per week, 14.00			
1 electrician, half-week, 6.00		59.50	0.11
General foreman, per week, $35.00		35.00	0.06
Total labor,		$738.50	$1.56
Sundries:			
5 tons of coal, at $5.00, 7 days,	$175.00		
Oil and candles,	12.50		
Steel,	5.00	192.50	0.35
Powder:			
2100 pounds rackarock, at 16½ cts.,	$346.50		
100 " J. L. aqueduct powder, at 16½ cts.,	16.50	363.00	0.65
Total labor and supplies,		$1294.00	$2.56

Recapitulation: Cost per Cubic Yard.

Inside labor:			
Drilling and blasting,	$0.42		
Mucking,	0.50		
Transportation,	0.31	$1.23	
Outside labor:			
Shaft,	0.09		
Engineers, etc.,	0.07		
Incidentals,	0.11		
General foreman,	0.06	0.33	$1.56
Fuel, etc.,		0.35	
Powder,		0.65	
Interest, etc., on plant,		0.37	1.37
Total			$2.93

Table VIII.

Log, Supplied by the Contractors at Shaft No. 15, somewhat Extended for Comparison.

Length, 102.1 linear feet. Area, 145.5 square feet. Rock broken, 550 cubic yards.

Date.	No. of Shot.	Commenced Firing.	Firing.	Drilling, etc.	Total Time between Blasts.	No. Cars of Rock.
			Hours.	Hours.	Hours.	
March 19, 1887.....	1.	5.30 P.M.....	0.50	10.30	12.00	50
" 20, "	2	6.10 A.M.....	0.50	11.10	12.00	50
" 20, "	3	4.50 P.M.....	1.50	9.50	14.00	57
" 21, "	4	6.30 A.M.....	1.10	11.30	13.00	62
" 21, "	5	6.00 P.M.....	0.30	11.00	13.00	60
" 22, "	6	6.30 A.M.....	0.40	11.30	13.00	54
" 22, "	7	5.30 P.M.....	0.40	10.30	13.00	66
" 23, "	8	6.10 A.M.....	0.50	11.10	12.00	74
" 23, "	9	6.00 P.M..... Disabled by	14.10 Powder-	11.00 smoke.	14.10	86
" 24, "	10	7.00 P.M.....	0.35	12.00	12.00	66
" 25, "	11	6.20 A.M.....	0.50	11.20	13.00	73
" 25, "	12	5.05 P.M.....	0.55	10.05	12.00	56
" 26, "	13	6.35 A.M.....	0.40	11.35	12.00	66
						820*

* Equal to only 410 cubic yards of material mucked out.

The Big Run in Shaft 16, North Heading.

Station 930 + 16, which was 3886 feet from the shaft, was selected as a favorable point at which to make the attempt to break the record. The rock was hard, fine-grained mica-schist, free from open seams, and practically the same as that encountered 700 feet north in Shaft 15, when the big run was made there. In 10 weeks previously, the heading had advanced 469 feet.

The subsequent removal of the corresponding bench required 17 weeks. The plant was the same as that at Shaft 15, but better arranged and probably in better condition. It had been carefully overhauled, and the machinery and tools had been put in perfect order. A supply of 60 per cent. powder, made by a special formula for the occasion, was obtained. The best-trained men were engaged, and all means known to art and force were combined for the supreme effort. The work was pushed to the utmost, not only by the foreman, but by the three sub-contractors in person, for 7 days or 168 hours; during this time an hour and 20 minutes were lost on account of the "air-tank taking fire," and two strikes of the laborers. Though the force was divided into 10-hour shifts, yet as soon as a man became exhausted he was replaced by another.

It is claimed by the sub-contractors, Messrs. Denton, Breuchaud and Pennell, that if they had not been impeded by the muck they could have fired another shot.

The bottom of the heading was about 3.8 feet above the spring-line. Area of heading above spring-line, 126 feet; area of bench below spring-line, 326 feet.

The following is the record:

1887, July 16th to 23d. Sta. 930 + 16 to 928 + 89 = 127 feet; area 125.6; cubic yards, 591. Distance from compressor, 3965 feet. Loss of pressure, 20 pounds. 2 Rattler drills, each using 530 cubic feet of air per minute.

	Feet.
Holes drilled per shot, center-cut, 8 holes 8½ feet deep,	68
side-cut, 10 holes 8 feet deep,	80
	148

	Feet.
18 shots, 324 holes,	2664
Feet drilled per cubic yard,	4.33
Total pounds powder used,	2050
Pounds powder used per cubic yard,	3.46
Pounds powder used per hole,	6 33
Advance per blast,	7.05

TABLE IX.

Log, supplied by the Contractors at Shaft No. 16.

Length, 127 linear feet. Area, 125.6 sq. feet. Rock broken, 591 cubic yards.

DATE.	No. of Shot.	Commenced work.	Mucking.		Drilling.		Charging.		Blasting.		Clearing.		Time bet. Shots.	
1887.			Hrs.	Min.	Hrs.	Min.	Hrs.	Min.	Hrs.	Min.	Hrs.	Min.	Hrs.	Min.
July 16.....	1	10.00 A.M.	1	30	5	00	0	30	1	30	0	30	9	00
	2	7.00 P.M.	2	30	5	30	0	30	2	30	1	00	12	00
" 17.....	3	7.00 A.M.	2	00	5	00	0	30	1	00	0	15	8	45
	4	3.45 P.M....	3	15	4	45	0	15	1	00	0	15	9	30
" 18.....	5	1.15 A.M....	2	00	5	30	0	15	1	00	0	15	9	00
	6	10.15 A.M. ..	2	15	5	00	0	30	1	00	0	15	9	00
	7	7.15 P.M....	2	15	5	00	0	30	1	00	0	30	11	45
" 19.....	8	7.00 A.M....	2	30	5	30	1	00	0	45	0	15	10	00
	9	5.00 P.M. ...	2	30	6	00	0	30	1	00	0	15	10	15
" 20.....	10	3.15 A.M..	2	15	4	45	0	30	1	00	0	15	8	45
	11	12.00 M.....	1	30	4	30	0	15	0	45	0	15	7	15
	12	7.15 P.M....	2	00	4	45	0	15	1	15	0	15	8	30
" 21.....	13	3.45 A.M....	1	15	4	30	0	15	0	30	0	15	6	45
	14	10.30 A.M....	1	30	5	30	0	30	1	00	0	30	9	00
	15	7.30 P.M....	2	30	5	30	0	30	0	45	2	15	11	30
" 22.....	16	7.00 A.M....	2	30	5	00	0	30	1	00	0	30	9	30
	17	4.30 P.M....	2	30	6	30	0	30	0	45	0	15	10	30
" 23.....	18	3.00 A.M....	1	30	4	30	0	15	0	30	0	15	7	00
Total.....	18	.....	38	15	92	45	8	00	18	15	10	45	168	00
Average per shot			2	08	5	09	0	27	1	00	0	36	9	20

Mucking,	23 per cent. of time.	
Drilling,	55 "	"
Charging holes,	5 "	"
Firing,	11 "	"
Clearing of smoke,	6 "	"
		100

Number of cars of muck, 953—equal to 489 cubic yards, which would correspond to a heading-area of only 104 square feet, so that about 15 to 20 per cent. of the total rock broken must have been left in the tunnel.

TABLE X.

Examples of Rapid Work.

DATE.	SHAFT.	TIME.	PROGRESS.			DRILLING.					POWDER.			REMARKS.
		Days.	Total Linear Feet.	Linear Feet Per Day.	Linear Feet Per Blast.	No. cub. yds.	No. of Drills.	No. Holes to Blast.	Av. Depth of Holes.	Total Feet Drill'd.	Ft. of Holes Per Cubic Yard.	Total p'nds. Used	Lbs. to Hole.	Pounds to Cubic Yard.	
June, 1886	23 N	22	254.5	11.52	.....	.....	4	.....	.....	.....	.....	.....	.....	..... *	Feet, Heading, 14 x 14
"	23 S	26	265.5	10.21	.....	.....	4	.....	.....	.....	.....	.....	.....	..... *	" "
1887.															
Jan. 3 to Feb. 1	15 S	24½	288.0	11.75	.....	.....	2	.....	.....	.....	.....	.....	.....	..... †	" 8 x 16
Jan. 8 to Mch. 12	15 N	54	567.0	10.05	.....	.....	3	.....	.....	.....	.....	.....	.....	.....	" "
" " "	15 S	60	730.0	12.16	.....	.....	3	.....	.....	.....	.....	.....	.....	.....	" "
Mch. 19 to 26	15 S	7	102.1	15.07	7.86	650	3	20	8.80	2288	4.16	2200	8.46	4.00 ‡	" 9 x 17
May 7 to July	16 N	60	469.0	7.81	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	" 9 x 16
July 16 to 23	16 N	7	127.0	18.14	7.05	691	2	18	8.25	2564	4.33	2050	6.83	8.46	" 9 x 16
1888.															Bench.
Mch. 31 to May 19	15 S	84	577.0	6.85	.....	3360	2	4	9.00	4788	1.40	7500	5.60	2.23	Worked from both ends.
Dec. 3 to Mch. 24	16 S	102	594.0	5.82	.....	.....	1	.....	.....	.....	.....	.....	.....	.....	Worked at one end.

* Eclipse drill.
† Slugger drill.
‡ Rattler drill.

General Cost of Excavating.

(Deduced from rate of progress given in Table I., page 717.)

Average Cost of Driving a Single Heading.

Progress, 47 feet. Area, 105 square feet. Rock broken, 183 cu. yds.

	Per shift.	For 14 shifts.		Per cubic yard.
Inside labor:				
Drilling and blasting,	$18.00	$252.00		
Mucking,	16.00	224.00		
Transportation,	9.75	136.50		
			$612.50	$3.34
Outside labor:				
At shaft,	3.00	42.00		
Engineers, etc.,		130.75		
			172.75	0.94
Coal, 3 tons per day,			122.50	0.67
Powder, 550 pounds at 16½,			90.75	0.50
Interest, on plant,				0.37
Total,				$5.82
By driving two headings at the same time, the outside labor and superintendence will be one-half, reducing the above amount by about,				0.50
Total, driving two headings at once,				$5.32

Average Cost of Excavating Bench.

Working Results: Sta. 921 + 33 to Sta. 927 + 00 = 577 linear feet. Area, 167.41 square feet. April 2d to May 19th, 1888 = 48 days.

Rock broken, 3578 cubic yards.
Material mucked out, 3360 "

				Per cubic yard.
Drilling and Blasting:				
1 boss,	$70.00 per month,	77 shifts,	$179.41	
1 drill-runner,	2.50 per diem,	157.5 "	393.75	
1 drill-helper,*	1.75 "	139.9 "	244.82	
1 nipper,*	1.00 "	85.5 "	85.50	
1 powderman,	2.00 "	46.0 "	92.00	
			$995.48	$0.28
Mucking and Transportation:				
1 boss,	70.00 per month,	84 shifts,	195.72	
15 muckers,*	1.50 per diem,	1553.3 "	2329.95	
			2525.67	0.75
1–2 drivers and mules,	2.25 per diem,	144.6 shifts,	325.35	
1 trackman,*	1.50 "	54.5 "	81.75	
2 dumpmen,*	1.50 "	171.6 "	257.40	
			664.50	0.20
Outside labor:				
1 bellman,*	1.75 per diem,	146.2 shifts,	255.85	
1 sumpman,*	1.75 "	52.4 "	91.70	
1 topman,	1.50 "	149.6 "	224.40	
			571.95	0.17
Carried forward,			$4757.60	$1.40

* Muckers were often used to replace other men.

					Per cu. yard.
Brought forward,				$4757.60	$1.40
Outside labor, continued :					
Hoisters, extra,	$2.00 per diem,	92 shifts,		184.00	0 05
2 engineers,	2.50 "	96 "	240.00		
1 engineer,	2 00 "	48 "	96.00		
2 firemen,	1.75 "	92 "	161.00		
1 pumpman,	1.75 "	48 "	84.00		
				581.00	0.17
1 machinist,	2.25 per diem,	46 shifts,	103.50		
1 blacksmith,	2.50 "	47 "	117.50		
1 helper,	1.50 "	48 "	72.00		
				293.00	0 09
Sundries :					
3 tons of coal per day, at $5.00 per ton, 48 days,			720.00		
Oil, candles, and steel, at $2.00 per diem, 48 days,			96.00		
				816.00	0.23
Powder :					
200 pounds forcite, at 40 cents per pound,			80 00		
7300 pounds John L. aqueduct powder, at 16½ cents,			1204.50		
27 bunches exploders, at 40 cents per bunch,			10.80		
16 spools wire, at 50 cents per spool,			8.00		
				1303.30	0.38
Total,				$7934.90	$2.32

Total number of shots fired, 133 ; missed, 14. Depth of holes, 9 ft. Number, 532.

Recapitulation : Cost per Cubic Yard.

Inside labor :			
Drilling and blasting,	$0.28		
Mucking,	0.75		
Transportation,	0.20		
		$1 23	
Outside labor :			
At shaft,	$.17		
At shaft, extra,	.05		
Engineers, etc.,	.17		
Incidentals,	.09		
		0 48	$1 71
Fuel and sundries,			0.23
Powder,			0.38
Interest, etc., on plant,			0.37
Total cost per cubic yard,			$2.69

Muck left in tunnel, 6 per cent.

Cost of Driving Heading and Bench Together.

Linear feet = 30. Area = 283 square feet. Cu. yds. = 314.

			Per cubic yard.
Inside labor, heading, 120 yds., $3.34	$400.80		
" bench, 194 yds., $1.23,	238.64	$639.44	$2.03
Outside labor,	$244.25	.78	
Coal and sundries,	192.50	.62	1.40
Powder, 124 cy. (3 pounds at 16½ cts.), @ .50,	$62.00		
" 190 cy. (2.3 pounds at 16½ cts.), @ .38,	73 72	135.72	44
Total,			$3.87
Working two headings at same time,			$3.17

The foregoing figures are submitted as an approximation to the truth. Acknowledgments are due to Mr. Joseph F. Banks, Mr. H. C. Allen, Mr. Gaylord Thompson, Mr. Alfred Craven, and Mr. F. S. Cook, for invaluable assistance in compiling the notes, and in making the drawings which accompany this paper.

AËRIAL WIRE ROPEWAYS.

BY J. POHLIG, COLOGNE, GERMANY.

(International Session, Pittsburgh, October, 1890.)

IT is with more than ordinary pleasure that I have complied with the request of the President of the Verein Deutscher Eisenhüttenleute, to read before this meeting and in this country a paper on a system of transport by wire ropeways which, during the past twenty years, has advanced with rapid strides in Europe, and more particularly in Germany. America, with its immense network of railways and water communications, should offer peculiarly favorable opportunities for the introduction of these ropeways as feeders, more particularly in the mining districts, where, as in Europe, a hilly or mountainous region frequently opposes serious obstacles to the employment of surface railroads.

The importance of the aërial ropeway as a factor in the development of mineral resources deserves the consideration of every mining engineer. As a case in point, the rich iron-ore of the Sierra de Bedar in southern Spain, would probably remain untouched to this day but for an aërial ropeway, 15.6 km. (9¾ miles) long, which connects the mines with the shore of the Mediterranean near the town of Garrucha, affording cheap carriage to the nearest shipping-point. A railway would have cost about $480,000 (£100,000), but such a heavy outlay would have more than absorbed all prospective profits on the sale of ore. As the aërial ropeway could be built for a little more than one-fourth of that amount, or $124,800 (£26,000), its erection furnished the successful solution of a profitable mining enterprise. This line has been in satisfactory operation for over two years; its details will be referred to later.

But beside their importance as feeders, ropeways are valuable as a means of conveyance between works and distant warehouses, and between separate buildings in the same works. Their rapidly spreading introduction for such purposes indicates how keenly the necessity of effecting economies is felt under the stress of an ever-accentuating competition.

Ropeways have not been as generally adopted in America as in Europe; the reason for this lies, I think, in the fact that thus far America's experience has been practically limited to the endless running-rope systems of Hodgson and Hallidie, which almost preclude continuous and regular working, and limit the weight of each single load to 100–150 kilos (2–3 cwts.).

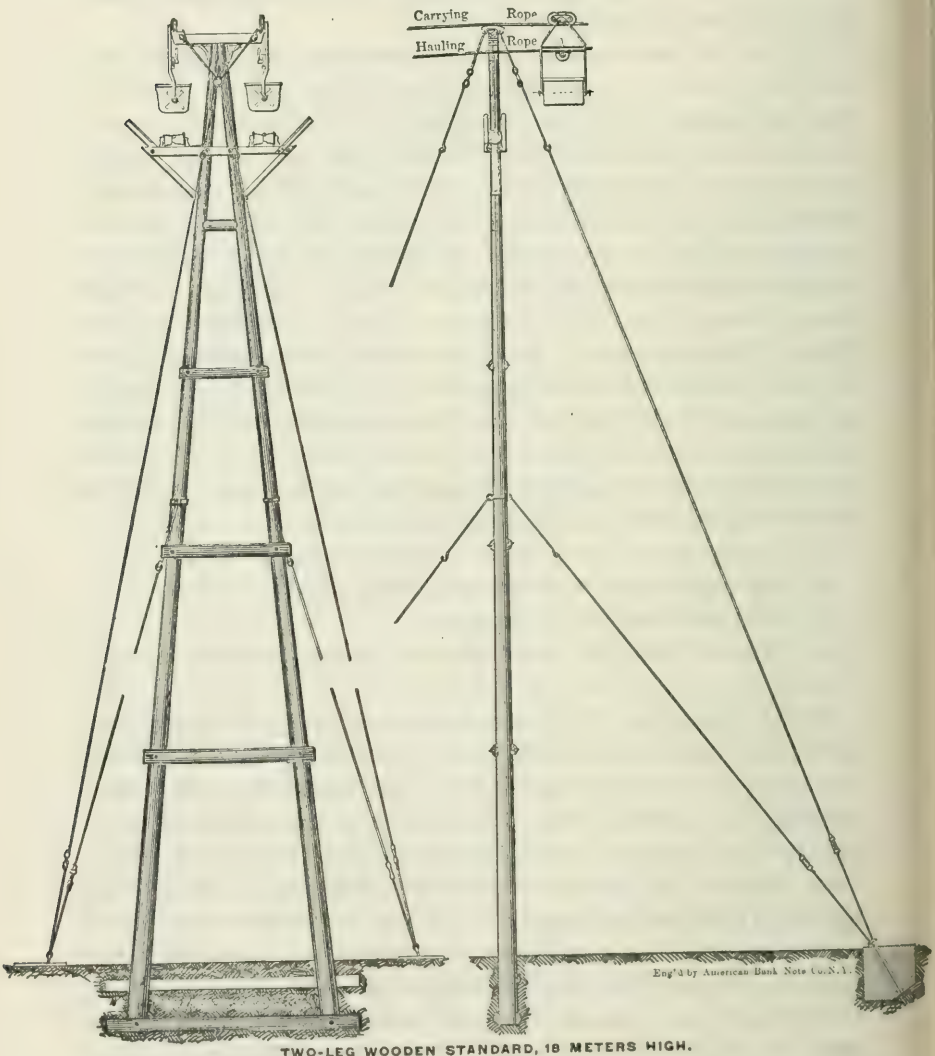
By both of these systems, one rope is employed, fulfilling the dual function of carrying- and hauling-rope for the buckets. Hallidie fixes his buckets to the rope; Hodgson, on the other hand, employs a saddle fitted with India rubber, which grips the rope by friction and is certainly an improvement. The simplicity and cheapness of these single-rope lines are at first sight very plausible, but yet their employment has been restricted. I believe the largest number of Hodgson lines in operation anywhere is to be found in northern Spain, more particularly in the iron-ore district of Somorostro, near Bilbao. Two or three of these lines are at work there, one above the other, carried on the same supports. The drawbacks experienced in operating them, however, have been so serious that to-day the German double-rope system has almost entirely superseded the Anglo-American or single-rope system, despite the endeavors of the resident engineers to remedy its defects, which are:

1. The slipping of the buckets in rainy and frosty weather.
2. The impossibility of working up steep inclines.
3. Great wear and tear of the ropes.
4. Frequent falling of the buckets in passing over the carrying-pulleys.

The characteristic of the German system (of which Bergrath Freiherr Franz von Dücker is the father), consists in the employment of two ropes—a heavy, fixed carrying-rope and a light, travelling hauling-rope. The buckets are fitted with special devices for gripping the hauling-rope. As is often the case with a new invention, a bad design of the details thwarted the satisfactory execution of the inventor's ideas, and as a result of the failure the system was regarded with very little favor in Germany some twenty years ago. Not until several years later, when Theodore Otto and Adolph Bleichert, of Schkeuditz, near Leipzig, began to improve its mechanical details were the doubts as to the merits of the system dissipated. Since then the demand for this class of ropeways has increased rapidly. In 1876, Otto and Bleichert dissolved partnership, each continuing the business independently and vying with the other to perfect its features. My own firm, formerly of Siegen, but now of Cologne

and Brussels, has taken an active part in this competition, first as Mr. Otto's representative, but during the past ten years as a constructor of "Otto" ropeways for its own account. I have made

FIG. 1.

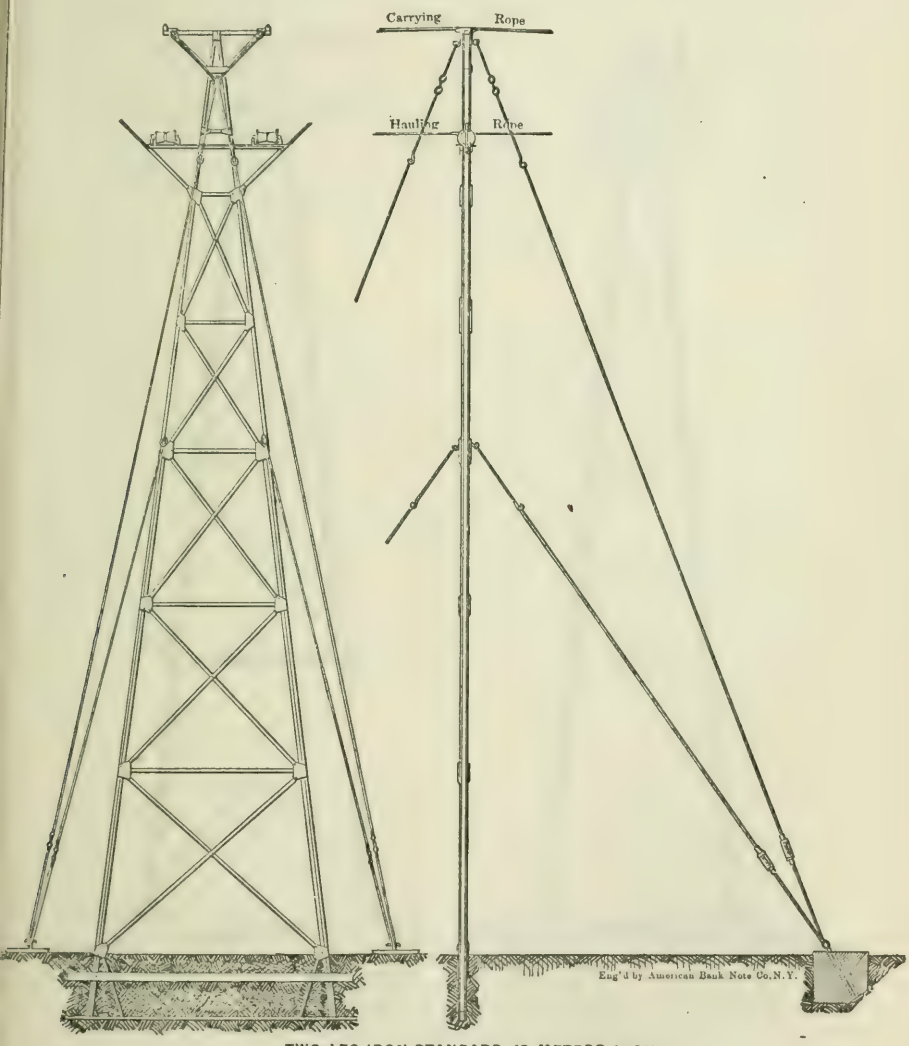


TWO-LEG WOODEN STANDARD, 18 METERS HIGH.

a special study and business of the construction and manufacture of these ropeways, and have erected some of the most important lines now in operation.

Competition and the ever-increasing demands upon this system of carriage have resulted in great improvements: ten years ago loads of more than 200–250 kilos (4–5 cwts.) were rarely transported, but

FIG. 2.

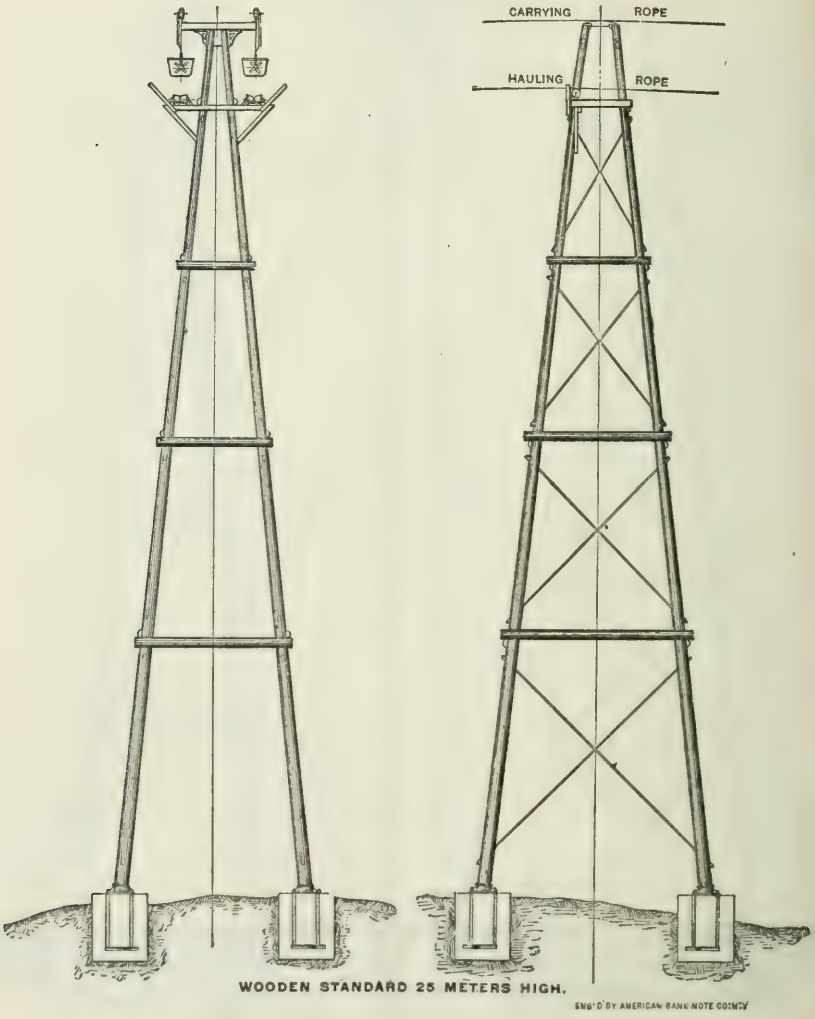


TWO-LEG IRON STANDARD, 18 METERS HIGH

at present it is not uncommon practice to carry loads of 400, 500 and even 1000 kilos (8, 10 and even 20 cwts.), so that ropeways are now capable of conveying from 600–800 tons per day of 10 hours.

Just as marked as the increased carrying capacity of the lines, has been the increase in their length; there are ropeways 10–13 km. (6–8 miles) long, and one even of 16 km. (9¾ miles). These facts suggest that the active field for these lines can still be greatly ex-

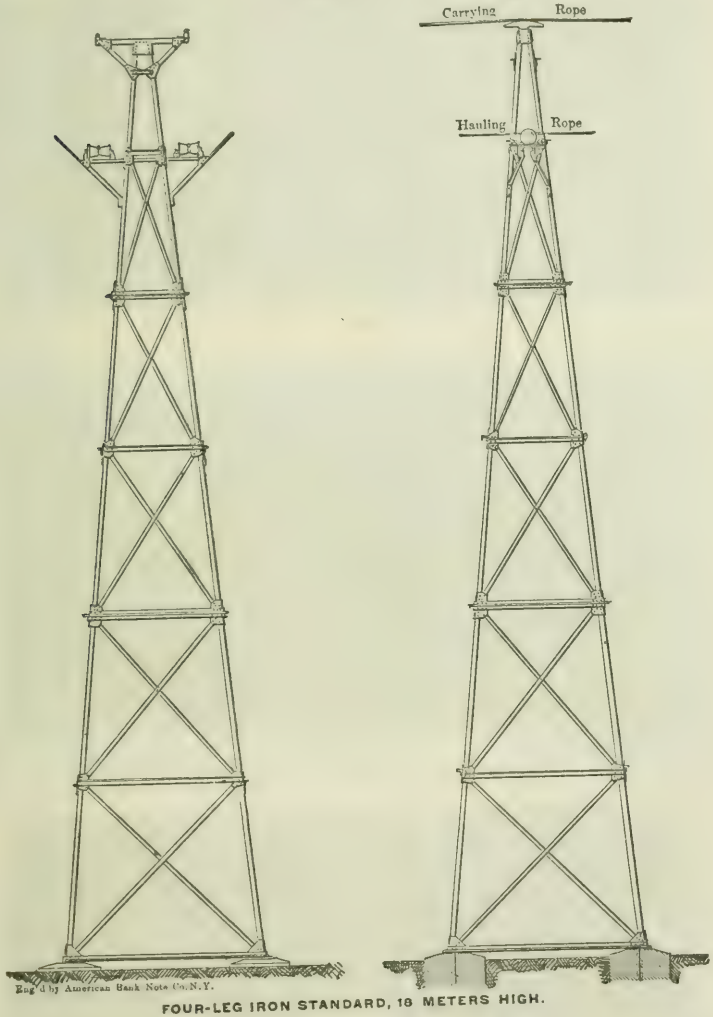
FIG. 3



tended. Practical men no longer look upon the ropeways as in any degree less safe or reliable than other means of transportation; hence, in suitable cases, where they can be shown to operate with greater economy than other systems, they are given the preference, and

therefore it is not surprising if, since 1873, fully one thousand of these ropeway-lines have been erected in Germany and other parts of Europe.

FIG. 4.



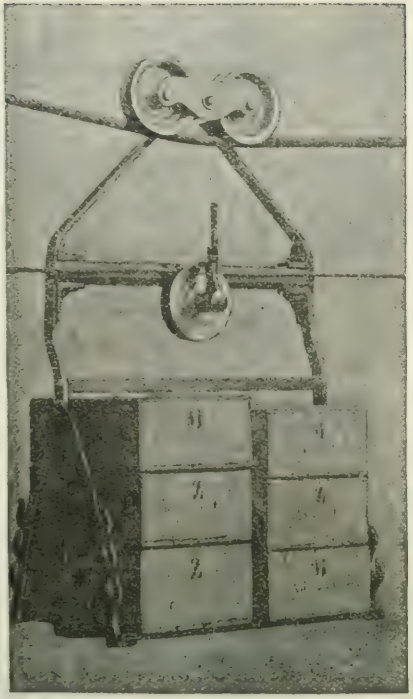
FOUR-LEG IRON STANDARD, 18 METERS HIGH.

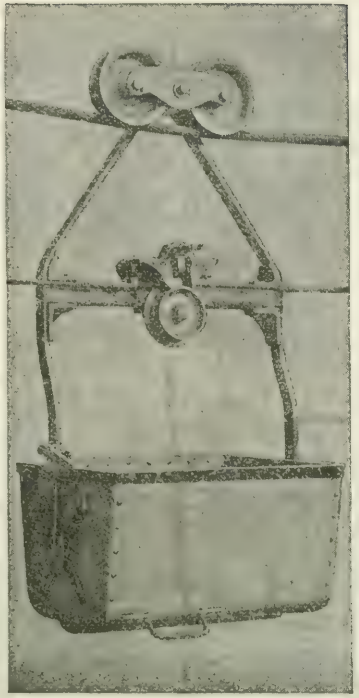
In the following brief description, I shall dwell only on the most important features of the ropes, standards and carriers.*

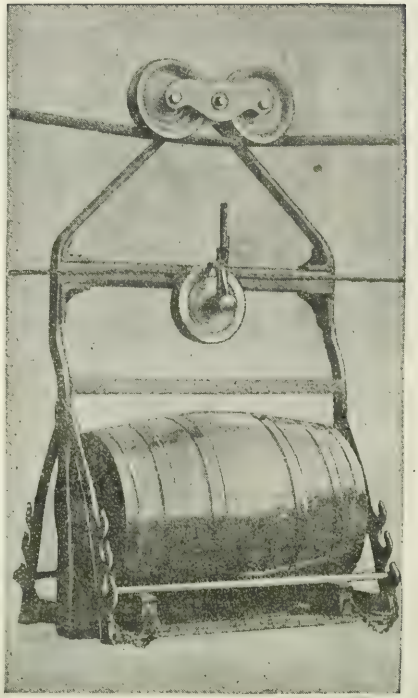
The *carrying-* or *fixed-ropes* are specially designed of stout steel

* A set of large and beautifully-executed drawings was exhibited by the author on the walls of the meeting-hall.

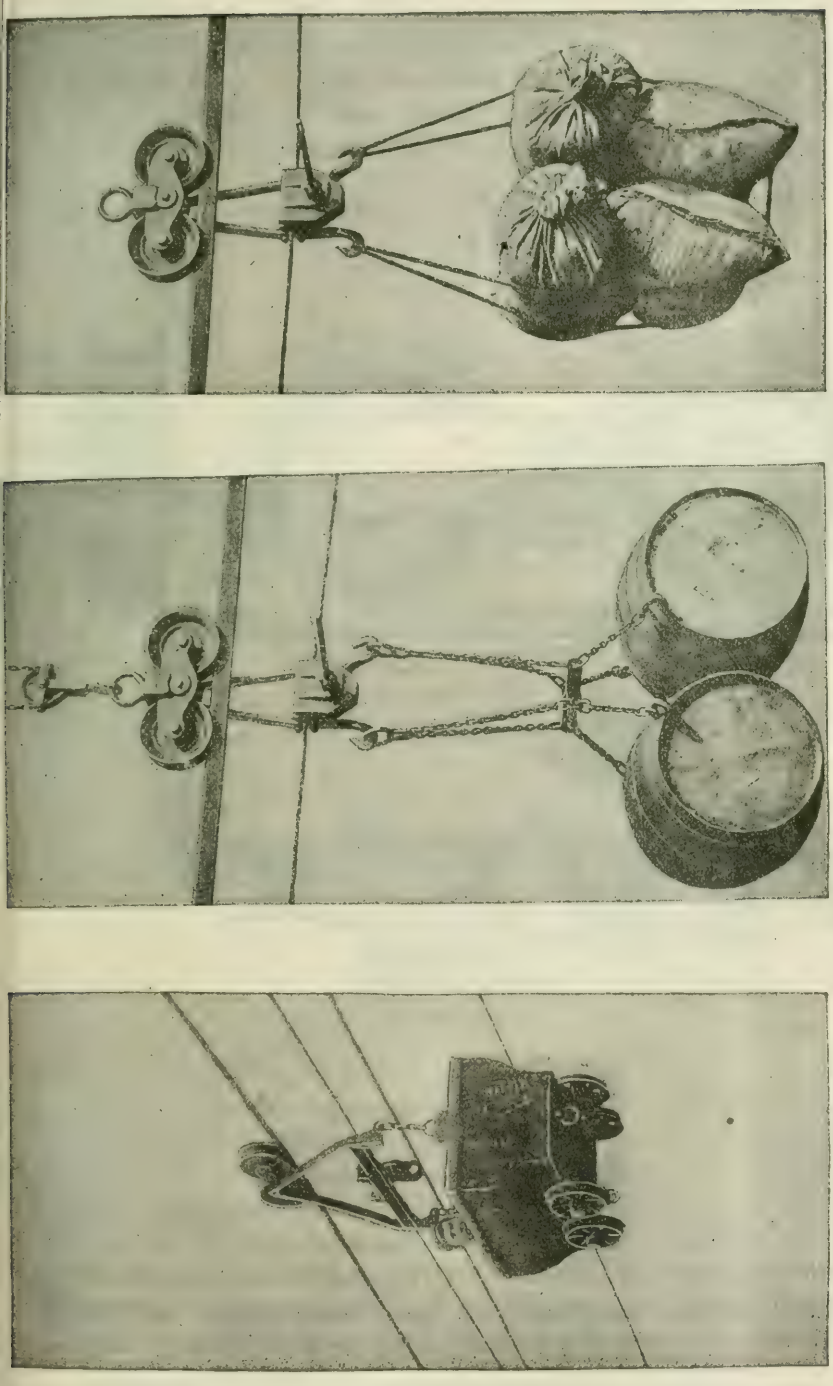
FIG. 5.







Carriers for the "Otto" Aërial Ropeways.



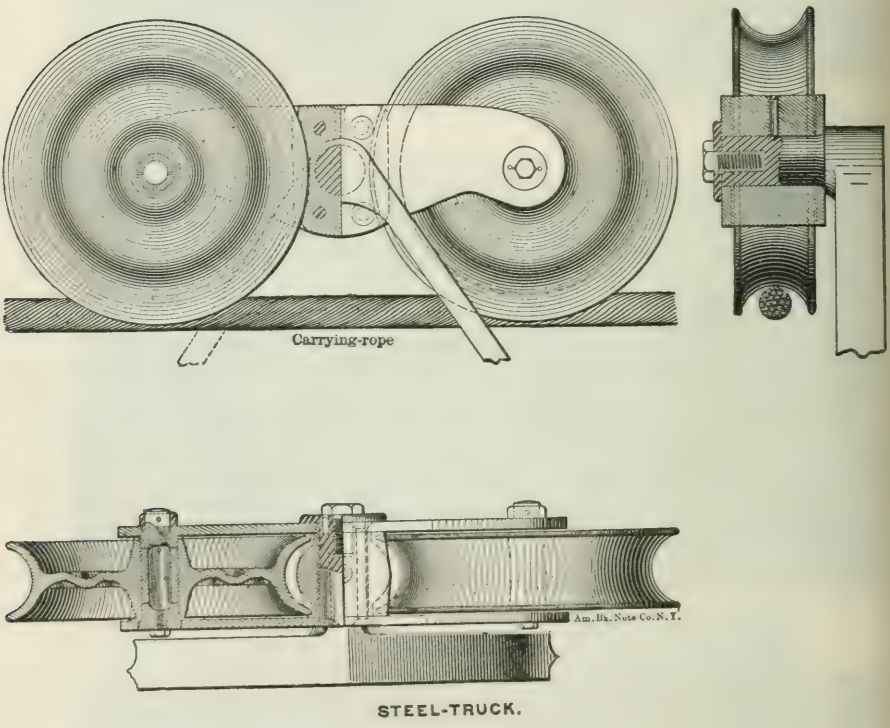
Carriers for the "Otto" Aërial Ropeways.

wire 4 7 mm. ($\frac{5}{32}$ to $\frac{9}{32}$ in.) in diameter, which has a breaking-strain of 60–120 kgs. per square mm. (38–76 tons per square inch).

The hauling-ropes, on the other hand, in order to be as flexible as possible, are made with a hempen core and of fine steel wire, which has a breaking strain of 120–180 kgs. per square mm. (76 to 114 tons per square inch).

The standards which support the ropes are usually placed 30–60

FIG. 7.



STEEL-TRUCK.

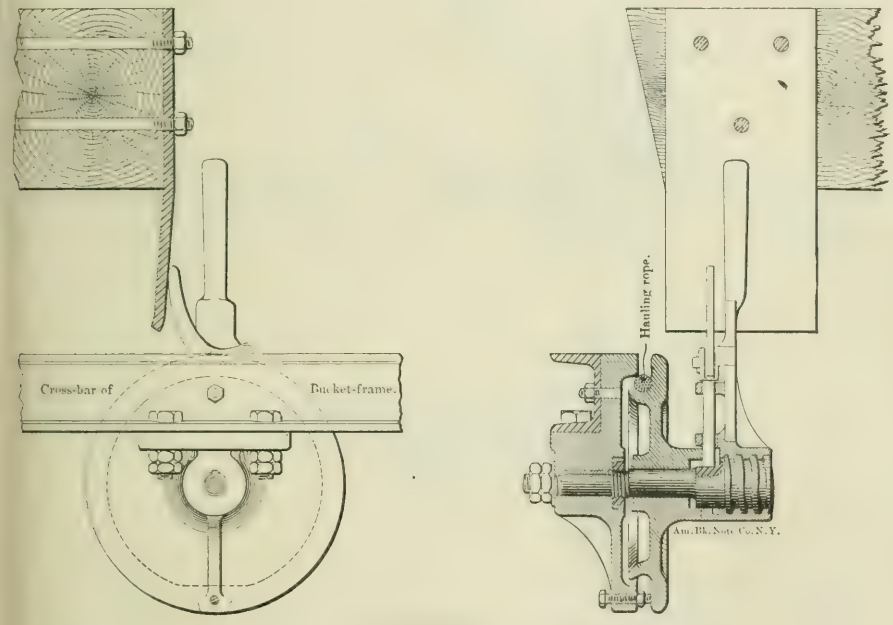
meters (100–200 feet) apart. Larger spans are adopted only where the nature of the country requires it; under special circumstances spans up to 500 meters (1640 feet) can be employed. The two-leg standard, Figs. 1 and 2, which I was the first to introduce, in 1881, is now the commonest form. When made of wood the legs may be ordinary round poles, with top cross-pieces of well-seasoned oak. If built of iron, the uprights are usually constructed of channel- or I-beams, stiffened with angle-iron, and the cross-pieces are made

of channel-iron. Where heavy loads and great spans are necessary, four-leg standards are employed, Figs. 3 and 4.

The carriers are of various designs to suit their particular service. For coal, ore, stone and similar material, sheet-iron buckets (middle design of Fig. 5) are used, with special provisions to tip them easily. For the conveyance of bulky goods, special forms of carriers, Figs. 5 and 6, are designed. For timber, boards, bar-iron, tubes, etc., two carriers coupled together usually answer.

An important feature of the carrier is the *truck* or runner, Fig.

FIG. 8.



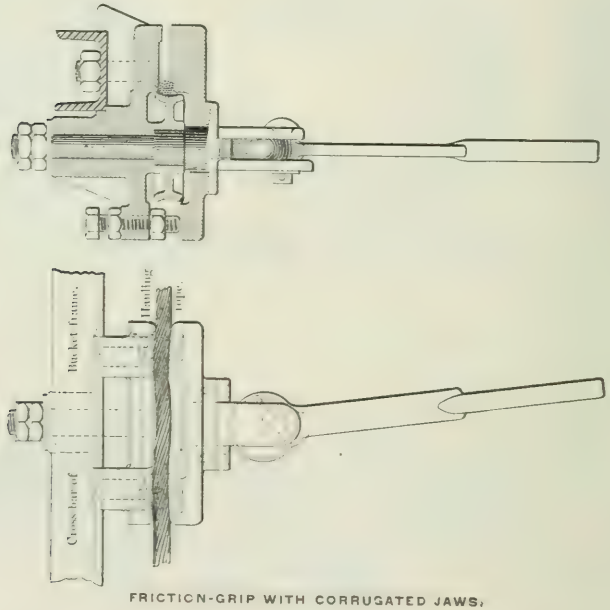
DISC-GRIP.

7. This design has been in use for over six years, and has entirely overcome the faults of the earlier trucks in which the spindles of the grooved wheels were supported on one side only; by supporting them on both ends the wheels are not liable to skew and the carriers are bound to hang always vertically. The frame of the truck consists of two steel plates connected in the middle by a cast-iron distance-piece through which the spindle of the hanger passes, and at each end by a phosphor-bronze distance-piece which forms the spindle for one of the grooved wheels. Each of these spindles is hollowed

out to contain a lubricant, which is introduced by removing the thumb-screw shown in the end. The lubricant passes through small holes to the surface of the spindle, and effects by this method such a perfect system of automatic lubrication that the spindles and rollers are preserved from almost every sign of wear even after many years' use.

Another equally important detail to ensure regular and uninterrupted working is the form of coupling, or *grip*, for attaching the carrier to the hauling-rope. This must be not only simple in design, but absolutely reliable in its action, and should reduce the wear on

Fig. 9.



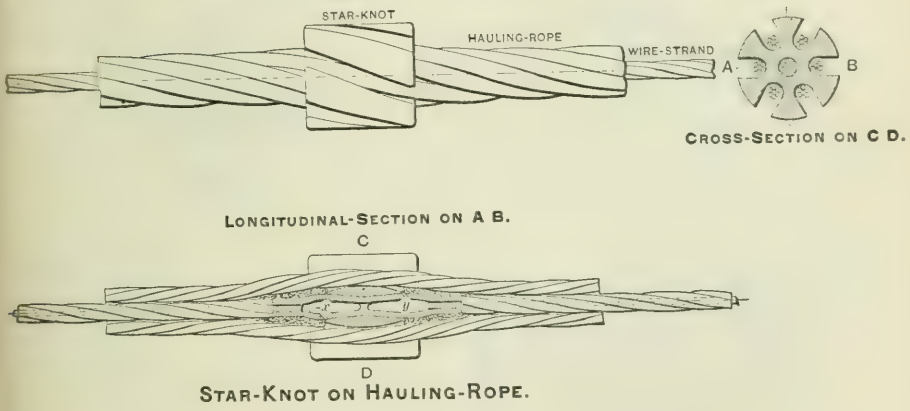
FRICTION-GRIP WITH CORRUGATED JAWS.

the hauling-rope to a minimum. The so-called friction-grip and the lock-grip are the usual types employed. With the former, the carriers can be attached to the hauling-rope at any point; with the latter, small collars or carrier-knots have to be fixed at intervals along the hauling-rope, corresponding to the distance apart at which it is intended to attach the carriers.

The form of friction-grip employed by us for the past twelve years, and now most generally used (Fig. 8), consists of two smooth-faced discs, one rigidly attached to the cross-bar of the hanger, and the other mounted loosely on a spindle and free to revolve, acting,

when necessary, as a carrying-roller for the hauling-rope. The spindle has a square thread on its outer end and carries a lever, the boss of which forms the nut. When the lever is down the discs do not grip the rope but are kept apart by a spring. As it is raised the discs approach each other, and when up the hauling-rope is tightly gripped. The lever is prevented from falling back by a small spring-trigger. As the carrier-bucket approaches a station, the lever and trigger both encounter a stop which throws them over, separating the discs and automatically releasing the hauling-rope. The bucket is then free to be switched off the carrying-rope to a siding. The discs can easily be adjusted, allowing for the size and wear of the rope. This form of grip is employed for gradients up to 1 in 6, and for loads of 450 kilos (9 cwts.) net weight.

Fig. 10.



STAR-KNOT ON HAULING-ROPE.

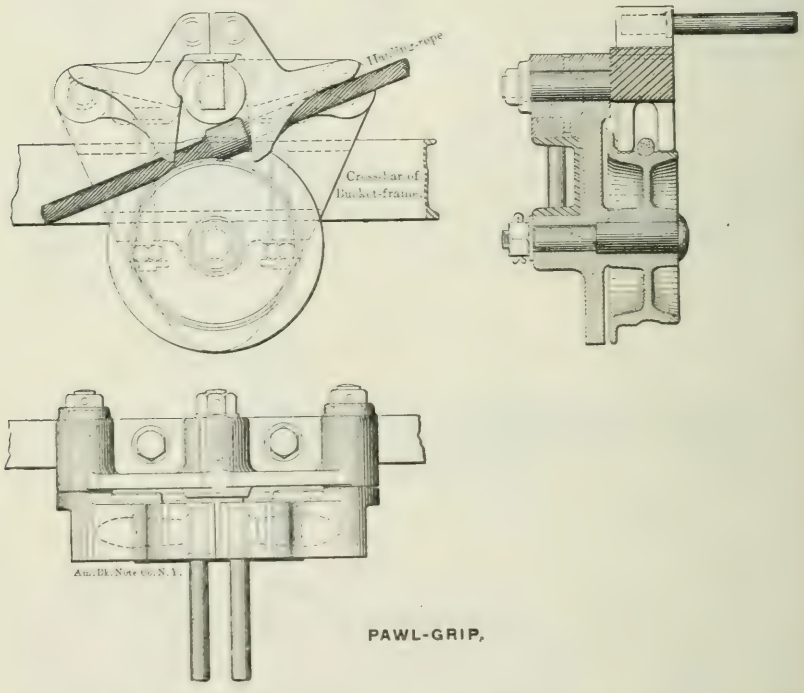
For steeper gradients, up to 1 in 3, where a friction-device is still practicable, a more powerful grip with corrugated jaws, Fig. 9, is used instead of the smooth-surface discs. One of the jaws is rigid, while the other is moved in and out by means of a lever and cam, so as to grip the hauling-rope. Both forms of friction-grip have given satisfaction. Since the buckets can be attached to the hauling-rope at any point, the tendency to a uniform wear throughout the length of the rope is promoted, and this is a distinct gain. Another advantage in using friction-grips is, that the carrying capacity of the line can be easily altered simply by adding more carriers at shorter intervals, without varying the travelling velocity of hauling-rope.

For lines having gradients under 1 in 6 the disc-grip is, in my opinion, the best existing form, as it subjects the hauling-rope to the

least wear. As an indication of how little the rope really suffers, I might mention the Fernie ropeway, erected in Giessen in 1879, where the original hauling-rope is still in regular use, transporting 300–350 tons of ore per day.

For gradients greater than 1 in 3, a lock-grip of some kind is absolutely necessary. The number of buckets and the corresponding capacity of the line for a given speed are then limited by the spacing of the knots on the hauling-rope. The design of the knot is of great importance—almost as vital as the design of the grip

FIG. 11.



PAWL-GRIP.

itself. An account of the various designs that have been tried and cast aside would occupy several hours. I will therefore confine my description to the star-knot, which I now use in preference to every other form. It consists of a cylinder (Fig. 10) whose diameter is somewhat greater than that of the hauling-rope, and whose surface is provided with spiral grooves into which the strands of the hauling-rope are inserted. When the knot is in place its ribs project sufficiently beyond the hauling-rope to furnish a hold for the pawls of the grip. In order to secure the knot in its position with absolute

certainty a steel wire strand about two yards long is passed through it and fixed by the two wedges x and y. When the knot is to be attached to the hauling-rope, the latter is first untwisted, and two yards of the hemp core are removed. The knot and the steel wire strand, which takes the place of the hemp core, are then set into the rope, and when this has been twisted tight its hold upon the knot and strand is sufficient to resist any shock to which they may be subjected. The points of superior excellence of the star-knot are:

1. Quick attachment to the endless hauling-rope without cutting it.
2. Attachment to the hauling-rope without the use of Babbitt-metal or any soldering-alloy.
3. Preservation of the full flexibility of the hauling-rope at the point of attachment.
4. Prompt detection and easy repair on any breakage of single wires.

Improvements in the form of the couplings or grips have kept pace with improvements in the knots. The earliest design of a lock-coupling was a complicated clockwork, full of springs and levers, which were ever getting out of repair. The pawl-grip, Fig. 11, which I have used for several years, is the perfection of simplicity. It never fails and requires no repair. It consists of two symmetrically disposed pawls each free to move in a vertical plane, and having forked ends which drop down on either side of the knot, the drop being checked by a stop. The rope is supported on a roller just below the pawls. To throw the pawls in and out of action, the toe of each pawl carries a pin which engages with a guide-rail at the station. The process of coupling a bucket fitted with one of these grips to the hauling-rope is as follows: The workman moves the bucket by hand along the switch-rail to the carrying-rope; on approaching this the pawl-pins come in contact with the guide-rail, raising the pawls so as to allow the hauling-rope to be guided and rest on the roller which is immediately below them and forms part of the grip. As soon as the bucket rests on the carrying-rope, the pins are released from the guide-rail and the pawls fall down in position over the hauling-rope. As a knot approaches it lifts the first pawl, which, however, falls back into its original position, but the second one resists and the bucket is automatically taken in tow. To avoid a violent impact of the knot upon the coupling, the shunter gets a bell-signal when a knot is approaching, and he pushes off the bucket at about the same velocity as the rope is travelling. The uncoupling is similarly affected. As the bucket nears the station, the pawl-pins

FIG. 12.

Garrucha Line of the "Otto" Aërial Ropeway, Southern Spain. View of the Power- and Intermediate-Station at Bedar.

FIG 13.

Garrucha Line of the "Otto" Aerial Ropeway, Southern Spain. Span of 280 meters (920 feet).

come in contact with a guide-rail, lifting the pawls out of gear and releasing the hauling-rope; the bucket, by virtue of its momentum, then moves off the carrying-rope and up a tongued rail, which switches it to a siding. This is the only lock-coupling, to my knowledge, that is thrown in and out of gear automatically.

With thousands of these pawl-grips now at work, I do not recall a single instance where a carrier has become uncoupled on the journey. I employ this grip exclusively on all mountain lines, and have used it in safely transporting loads of more than 1000 kilos (1 ton) up gradients of 1 in 1.

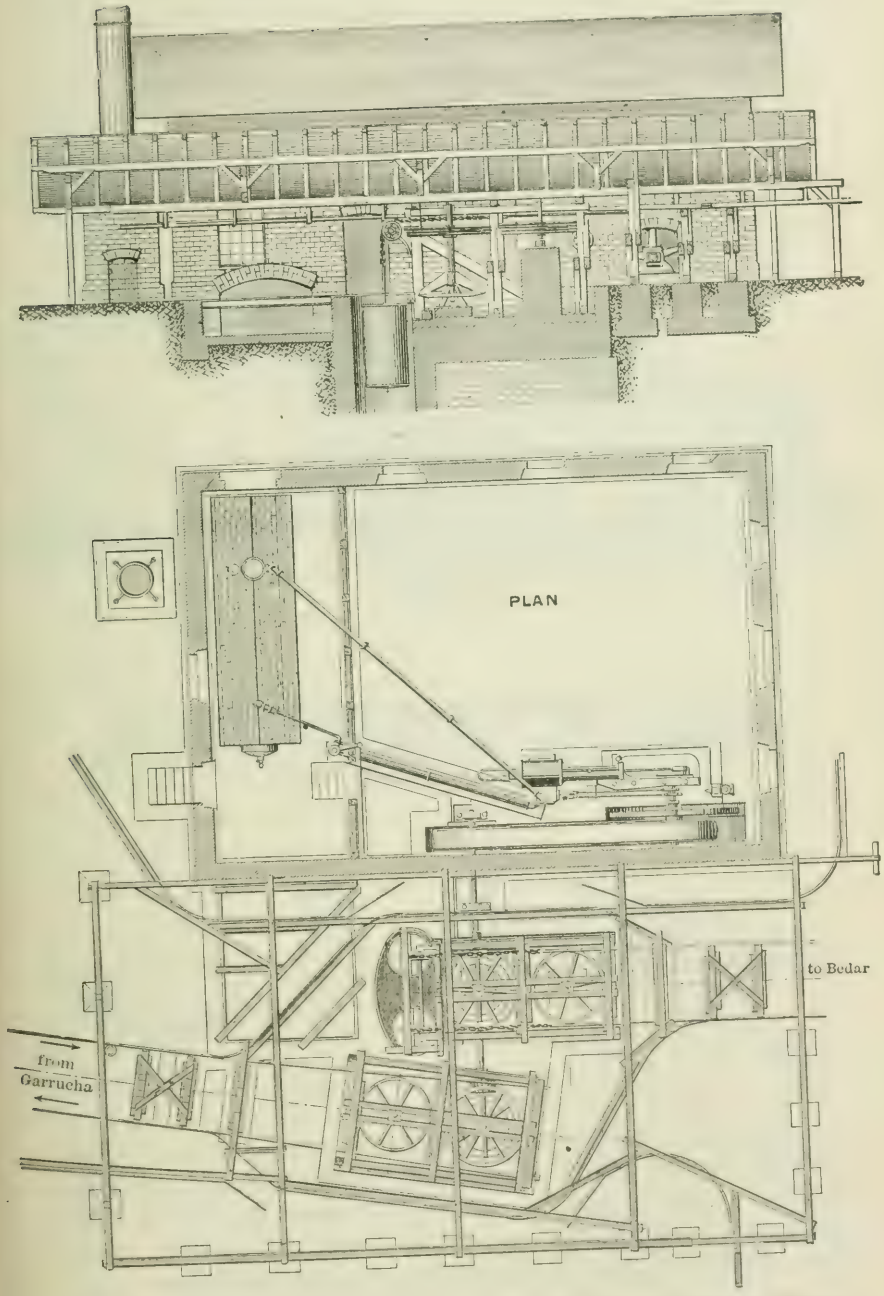
Description of Some Ropeway-lines.

Garrucha Line.—The most important Otto ropeway yet constructed is the one I have already alluded to, for the transport of iron-ore in southern Spain, from the Sierra de Bedar to Garrucha. This line, Figs. 12 and 13, is 15.6 km. (9.69 miles) long and divided into four independent sections with lengths of 2.4, 3.4, 5.3 and 4.5 km. (1.40, 2.11, 3.29 and 2.8 miles) respectively. A 30 horse-power engine drives the first two sections, and a 70 horse-power engine the last two.

After leaving the loading-station at Serena, 276 meters (905 feet) above the sea, the line crosses a number of deep valleys, one nearly a kilometer (upwards of half a mile) wide and 100 meters (328 feet) deep, and traverses mountain-ridges, the highest of which is 358 meters (1174 feet) above sea-level, to the village of Pinar de Bedar, where, at an elevation of 290 meters (951 feet) the first power-station is located. From here the line deflects to the right and again passes over several valleys and ridges, with a gradual descent to an angle-station 113 meters (370 feet) above sea-level. It then bears to the left, extending over a more or less hilly country to the second power-station near Puerto del Coronel (Figs. 14 and 15), at an elevation of 45 meters (147 feet). From here it again turns to the right, descending at a comparatively easy gradient to the unloading-station on the coast, near the town of Garrucha.

Viewed from the mountain-ridge between Bedar and Serena, whence the line can be traced from end to end, the prospect is imposing—660 buckets travelling back and forth, growing smaller in the distance, until they seem like faint specks; and the ropes, brightly reflected in the sunlight, looping in the sea with a silver cord. The capabilities of the system are nowhere more strikingly exhibited, and the precision with which the carriers move and deliver their loads heightens the effect.

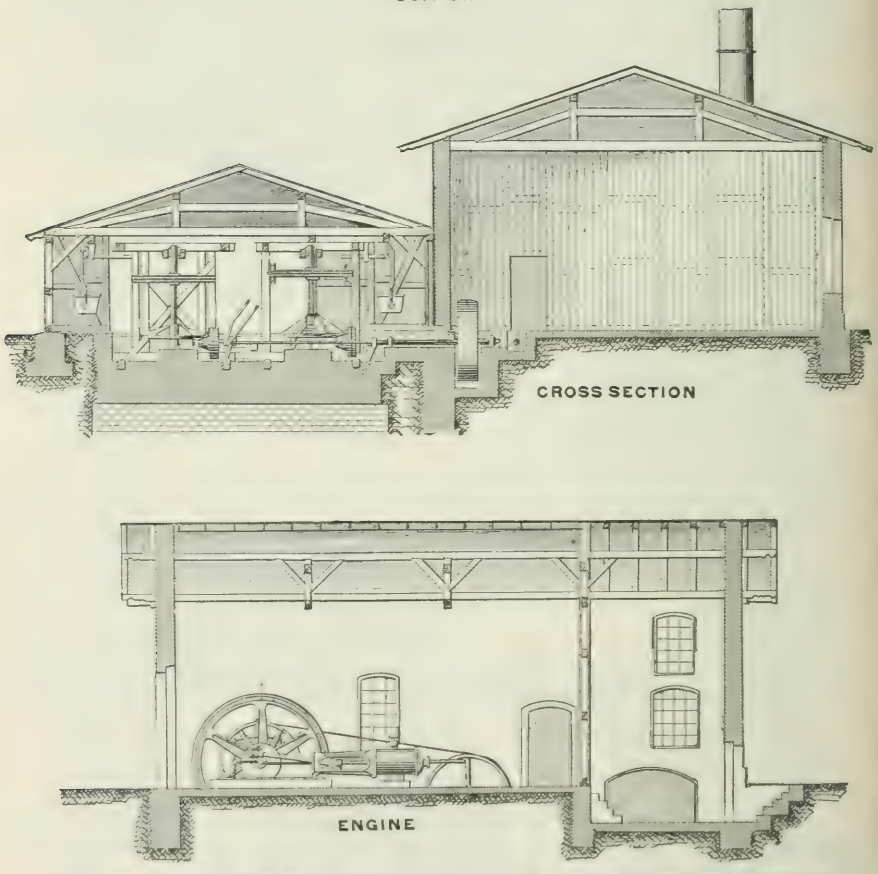
FIG. 14.



Garrucha Line of the "Otto" Aërial Ropeway, Southern Spain. Plan and Section of Power- and Intermediate-Station near Puerto del Coronel.

The greatest span of the line, near the Villa Reforma, is 280 meters (918 feet). In this span the rope sags 20 meters (65 feet), and carries 6 loaded and 6 empty buckets at a time. Its height above the valley is 50 to 60 meters (164–196 feet). The other long spans of the line range from 100 to 225 meters (328–750 feet), but the average dis-

FIG. 15.



Garrucha Line of the "Otto" Aërial Ropeway. Section of Power and Intermediate-Station near Puerto del Coronel.

tance between the supports is only about 40 meters (130 feet). The steepest gradient, taking into account the sag of the rope, is 1 in $2\frac{1}{2}$, and the tallest standard is 36 meters (118 feet) high, Fig. 12. The guaranteed capacity of the line is 400 tons per day of 10 hours. With a travelling rate of 1.5 meters per second (300 feet a minute or about $3\frac{1}{2}$ miles an hour), and with two buckets of 350 kilos (7

cwts.) capacity arriving every minute, or 1200 buckets per day of 10 hours, the actual quantity carried by the line is 420 tons, making its capacity 6552 ton-kilometers (4095 ton-miles), which, so far as I know, has not yet been equalled by any other line. Owing to the increased demand for Bedar ore, the line has been worked since the commencement of 1890 in two shifts of 8 hours, and no less than 900 tons per day have been transported to the coast.

The carrying-ropes for the loaded and unloaded side respectively are 33 and 26 mm. ($1\frac{5}{16}$ and 1 inch) in diameter, and the size of the hauling-rope is 18 mm. (about $\frac{3}{4}$ inch). The latter is fitted with the star knots above referred to. The pawl-grip, previously described (Fig. 11), is employed throughout.

At the loading-station, bins of 800 tons aggregate capacity are erected, from which the ore is spouted into the buckets. Great care has been bestowed on the design and arrangement of the power-stations and all their gear and engines. The engine- and boiler-houses are solidly built and large enough to be used as repair-shops. (See Figs. 14 and 15, showing the second power-station near Puerto del Coronel.)

The unloading-station on the coast is 46 meters (150 feet) long by 15 meters (50 feet) wide, and 10 meters (32 feet) above the ground-level. Its storage-capacity is 18,000 to 20,000 tons, so that 4 to 6 vessels can be loaded at a time.

At the various stations sidings are arranged for docking empty carriers of the several sections of the line. Electric signals are used and the stations are connected by telephone. Despite many difficulties, the line was surveyed, erected and ready for work within 10 months, its total cost amounting to $124,800 (£26,000).

At the desire of the mining company, I undertook to work the line for a number of years at the rate of 1.50 francs (28.8 cents) per ton carried, this price covering all costs for labor, maintenance and repairs.

Völklingen Line.—The second line which will be of special interest is at the Völklingen iron-works. It serves to convey refuse across the river Saar to a waste-dump, and to carry back coal from a neighboring colliery, for use in the rolling-mills. The plant consists of three intercommunicating lines, as shown in the General Plan, on the right-hand side of the accompanying Plate:

The blast-furnace line; the rolling-mill line; the Hostenbach line.

Their aggregate length is about 3.3 km. (2 miles), and they are driven from one central station.

The blast-furnace line, which is used for the removal of slag, passes through an underground tunnel, in order to leave the space around the furnaces free. Suspended rails have been used in its construction in place of the customary wire-carrying cables. Before delivery into the buckets the slag from each furnace is granulated and collected in a large iron receiver, which has a perforated false-bottom (see Arrangement of Slag-Receivers, Plate). Receivers have been built of 100 and 200 cubic meters (3530–7060 cubic feet) capacity, or sufficient to hold all the slag produced at night, so that the rope-line is operated during the day-turn only. Each receiver is divided by a partition, allowing one side to drain, while dried sand is being loaded into buckets from the other. The buckets are filled from spouts placed around the bottom of the receivers, and fitted with slide-gates. The receivers are covered in on the ground-level with a strong iron grating, so that the workmen can pass over them without let or hindrance.

This method of granulating and removing the slag without the employment of elevators, etc., I used for the first time in 1882–83 in connection with the ropeway at the Johanneshütte, near Siegen. It has since met with great favor in Westphalia where a number of works have adopted it, even among those who at first were strenuously opposed to granulating because of the greater space occupied by slag-sand than slag-blocks. It cannot be denied that the quicker filling-up of the dump which results from granulating the slag is in many localities a serious drawback; yet the advantages of this system over the costly haul of solid or liquid slag will often more than offset the objection, aside from the consideration that ropeways afford a means for piling the waste-dump to any desired height; or, if suitable dumping-ground does not exist near the furnaces, for transporting it to any desired distance, possibly to fill in old quarries or level up waste land. Many attempts have been made to adapt this method of transport to solid slag—the slag being run into specially designed buckets and allowed to solidify, but in every instance such efforts have ultimately failed. I tried it on a large scale at the Niederscheldener iron-works, near Siegen, but was finally obliged to give it up because I could not succeed in making sufficiently durable buckets, either of cast- or wrought-iron or steel. To line the buckets with fire-brick would have made them too heavy. I can, therefore, confidently recommend the system of granulated-slag transport as the simplest and cheapest, provided that local condi-

tions permit the use of a low-level loading-station and favor an easy drainage of the water.

The rolling-mill line is carried above the mill to the unloading-station for coal, which consists of an iron scaffold 9.4 meters (30 feet) high, situated in front of the puddling-works. Below it are seven large iron pockets of 500 tons aggregate capacity, into which the coal is dumped from the ropeway-buckets. When coal is required in the works it is drawn from the pockets through suitable slide-gates and delivered by light portable railway-trucks.

The main line to Hostenbach for the further conveyance of the slag and return-transport of the coal (see the General Plan, Plate), crosses the railway-line and the river Saar to the top of a wooded hill, 30 meters (98 feet) high, on which a dumping-station 19.6 meters (63 feet) high has been erected (Plate, left side). This serves at the same time as an angle-station for the extension of the line to the Hostenbach colliery. The loading-station at Hostenbach consists of two large bins, each of 100 tons capacity, fitted with six spouts for loading the ropeway-buckets (Plate, left side).

All three lines are driven by one 40 horse-power coupled engine, the power being transmitted by a belt and friction-clutch. Each line is so arranged with switches that the buckets can be run from one to the other without difficulty. Moreover, if at times it is not necessary to carry back coal from Hostenbach, the slag-trucks, upon arriving at the dumping-station, are emptied and then passed through the station and recoupled to the river-branch of the line, as indicated by a dotted curve in the plan of the station (Plate, left side).

The main line is designed to carry 500 tons of slag to the dump, and 300 tons of coal to the mill. As a rule two buckets of 400 kgs. (8 cwts.) capacity arrive in each minute, travelling at a velocity of 1.5 meters per second (300 feet a minute). They are, therefore, 45 meters (150 feet) apart, and on an average 1200 loaded buckets are carried per day of ten hours.

The diameters of the carrying-ropes for the main line are 33 and 30 mm. ($1\frac{5}{16}$ and $1\frac{3}{16}$ inch). Those for the Hostenbach and rolling-mill lines respectively are 30 and 25 mm. ($1\frac{3}{16}$ and 1 inch). The hauling-rope is of one size throughout, namely, 15 mm. (nearly $\frac{5}{8}$ inch) diameter. The gradients are easy, and disc-grips, therefore, used.

All the standards are of iron except on the slag-dump, where wood trestles are used and are allowed to be buried under the accumulating slag. Fig. 2 shows the design for all the iron standards

except the two on the banks of the Saar. As the river-span is upwards of 100 meters (328 feet), these two, which are 19.84 meters (65 feet) high, have been built of the four-leg type.

The general method of operating the plant is as follows: The empty buckets at the power-station are coupled to the hauling-rope at the point *A* (see Plan of Power-Station, Plate), and travel into the tunnel, where they are automatically disengaged in front of any one of the receivers. When the buckets have been filled they are again coupled up to the hauling-rope, and on reaching the point *B*, are automatically uncoupled and shunted round on the hanging-rail, *s*, to the point *C*, on the main line, along which they travel to the angle-station, where at the point *D* (see Station for Slag-Dump, Plate, left side), they are shunted off on the hanging-rails, *s' s'*, and the slag is dumped.

The empty buckets are then pushed round to the point *E*, coupled to the hauling-rope and despatched to the colliery. Arriving at *F* (see Loading-Station at Hostenbach Mine, Plate, left side), the buckets are filled with coal from the bins, pushed round to the point *G*, and started back, passing *H* and *I* in the angle-station and reaching the central station at *K*. From here the buckets are moved round to *L*, and pass over the rolling-mill line to *M* (see Unloading-Station on the Rolling-Mill Line, Plate), and thence along the hanging-rail, *s' s'*, to the coal-bins, into which the coal is tipped. The empty buckets are then pushed to the point *N*, and dispatched back to the central station. On arriving there, at *O*, they are shunted round to the starting-point, *A*, and thus the circuit is completed. If no coal is wanted from the mine, all that is required is to throw in the switches, *w w*, at the angle- and central stations, in order to short-circuit the line.

For the removal of ashes and rubbish, short lines are used, which can be connected with the central station. When all the sections are in operation, twenty-two men are required:

At the central station: 1 engine driver, 3 shunters for coupling and uncoupling the buckets, 1 man for moving the trucks.

In the tunnel: 3 shunters for loading and coupling up the buckets.

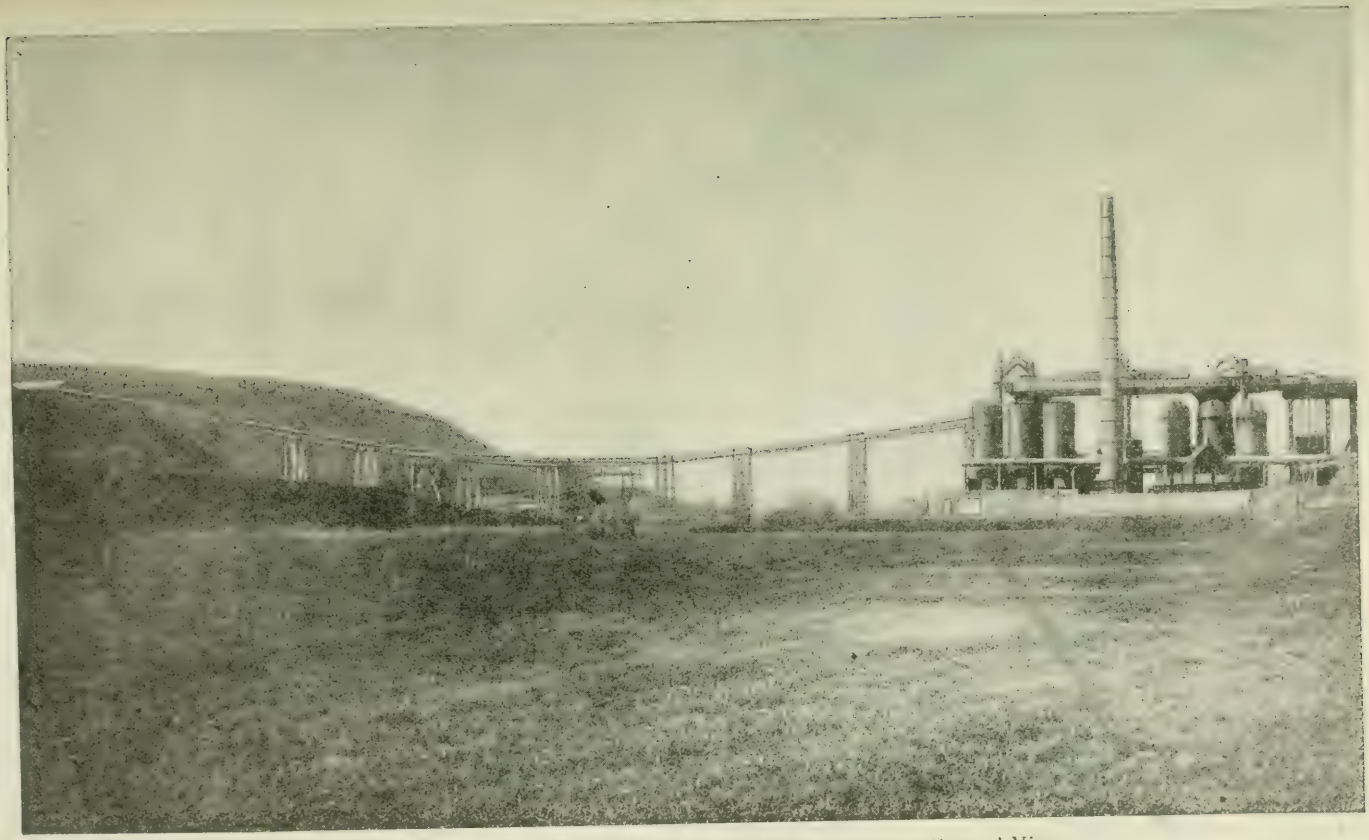
At the angle-station: 2 shunters and 3 men to move and tip the trucks.

At the colliery: 2 men to shunt and 1 to load the trucks.

At the rolling-mill station: 2 shunters and 1 man to unload.

For the plant in general: 1 inspector and 2 fitters.

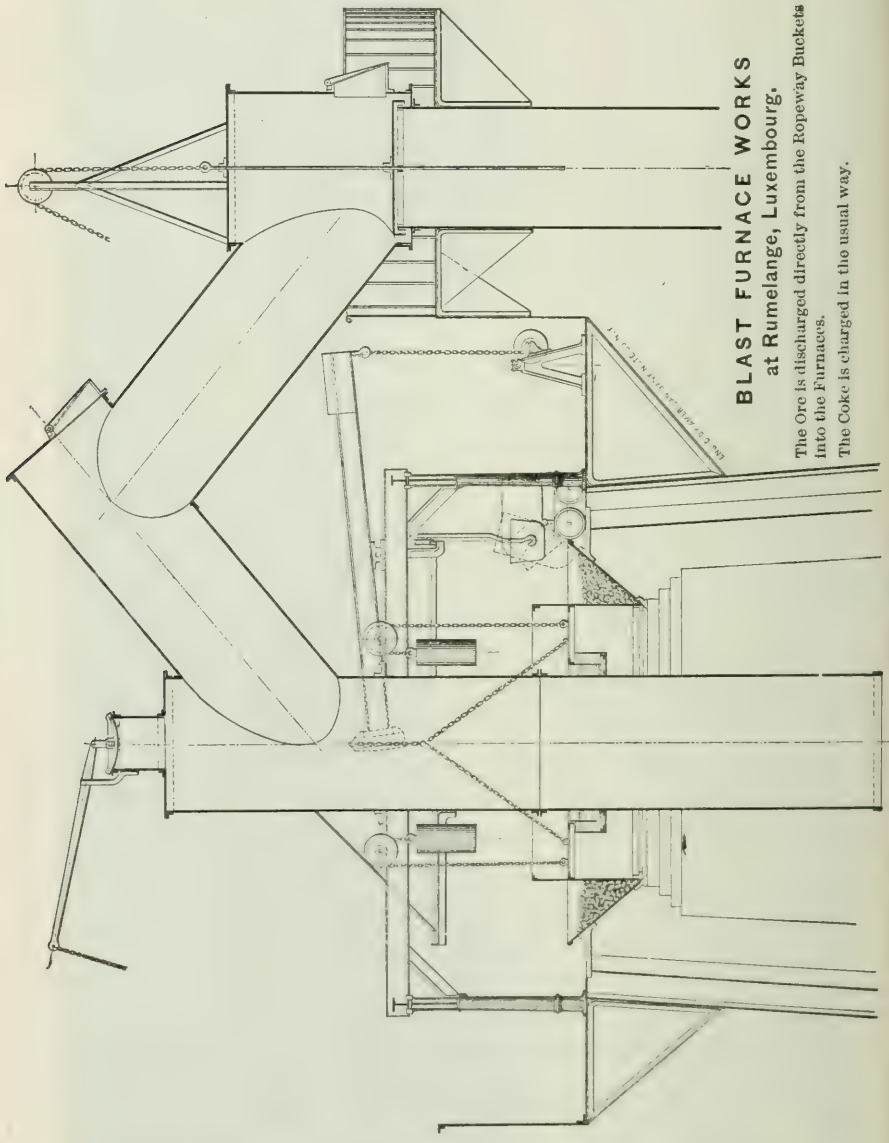
Fig. 16.



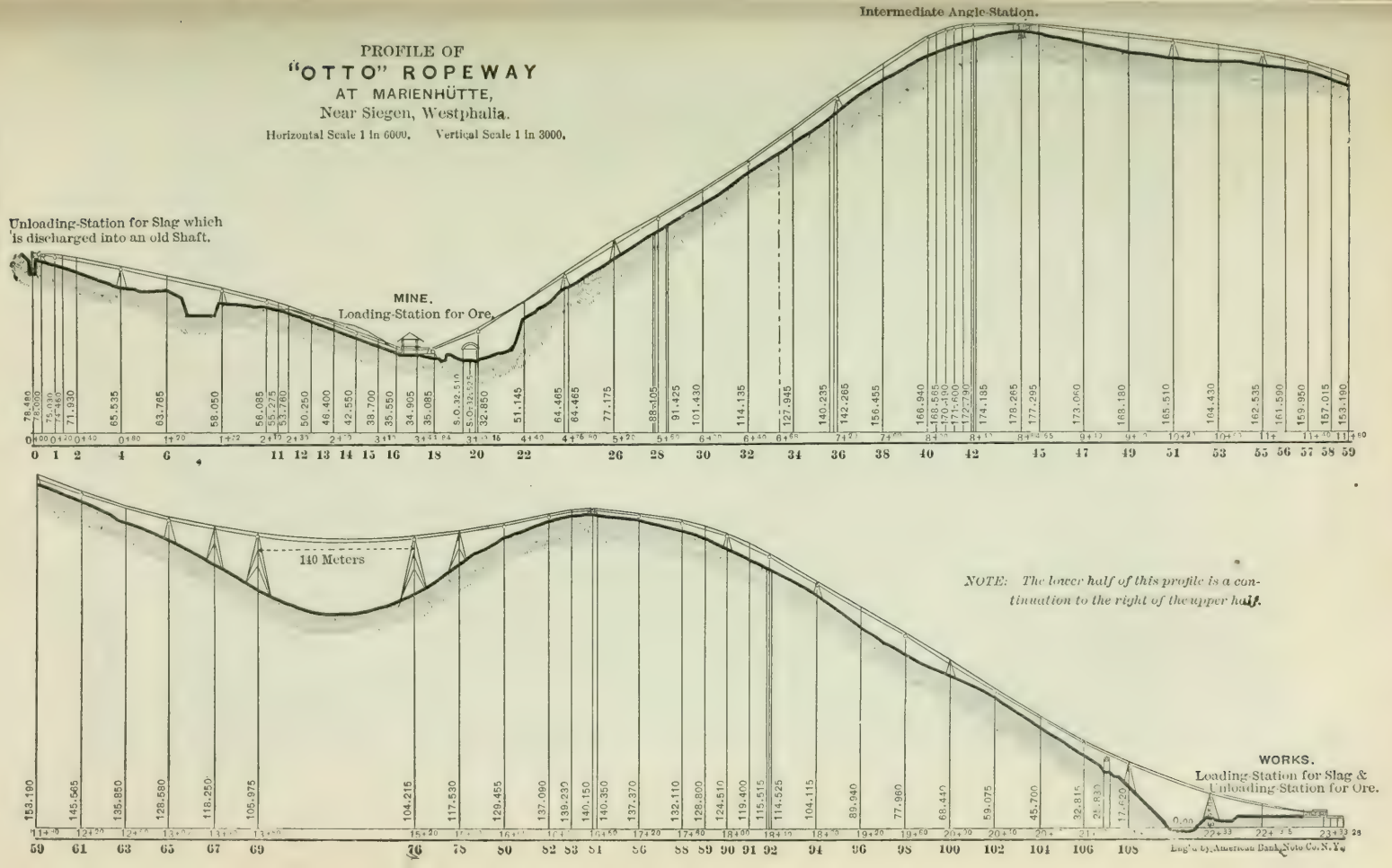
Rumelange Line of the "Otto" Aërial Ropeway, Luxembourg. General View.

The line has been worked continuously since 1889, and has given great satisfaction.

Fig. 17.


BLAST FURNACE WORKS
at Rumelange, Luxembourg.
The Ore is discharged directly from the Ropeway Buckets into the Furnaces.
The Coke is charged in the usual way.

Rumelange Line.—This was the first double aërial line erected, and the first by which blast-furnaces were directly supplied. The general features appear in Fig. 17, and the charging-arrangement


PROFILE OF
"OTTO" ROPEWAY
AT MARIENHÜTTE,
Near Siegen, Westphalia.
Horizontal Scale 1 in 6000. Vertical Scale 1 in 3000.
Intermediate Angle-Station.
Unloading-Station for Slag which is discharged into an old Shaft.
MINE.
Loading-Station for Ore.
140 Meters
NOTE: The lower half of this profile is a continuation to the right of the upper half.
WORKS.
Loading-Station for Slag & Unloading-Station for Ore.
Engd by American Bank Note Co. N.Y.

in Fig. 17. Iron-ore is carried direct from the mine to the top of the furnaces.*

Marienhütte Line.—At these works, which are situated near Siegen, in western Germany, a line is employed for carrying iron-ore from the mines to the furnace-yard and returning slag to the mine for back-filling. Fig. 18 shows the profile, and Figs. 19 and 20 the loading- and unloading-stations for the ore.

Aiseau-Presle Colliery Line.—This line, near Farciennes, in Belgium, derives special interest from the fact that the mine-cars themselves are carried over the line instead of ordinary buckets. The weight of each loaded car is 900 kgs. (18 cwts.), and the amount of coal transported daily, 800 tons.

Lines in the Transvaal.—Ropeways are now being introduced into the Dark Continent for the transport of gold quartz. Four lines have already been determined on for the Sheba, Edwin Bray, Oriental and Amsterdam Gold Mining Companies. Two of these are now completed. The nature of the region traversed is shown by the profile, Fig. 21, which offers a further illustration of the peculiarly favorable adaptation of these aërial ropeways to a very difficult, broken country.

Conclusion.

In conclusion, gentlemen, I trust that what I have now brought to your notice may decide you to make a trial of these aërial ropeways and lead to their introduction into this country. I am sure that my fellow-countrymen and colleagues, equally with myself, esteem it an honor to be able to introduce to your notice something of a practical nature in return for the many great and practical improvements which you have sent us across the water.

Discussion.

R. W. Raymond, of New York: In the absence of Mr. E. G. Spilsbury, Manager of the Trenton Iron Company, of Trenton, N. J., who would have said something on this paper, if it had been presented, as was expected, at the New York Meeting of the Institute of Mining Engineers, I desire to express my thanks to Mr. Pohlig for bringing before us the interesting details of this system,

* For a full description, see *Zeitschrift des Vereins Deutscher Ingenieure*, vol. xxxii., p. 1045.

and, at the same time, to correct the impression which the concluding paragraph of his paper seems to indicate, that the system is

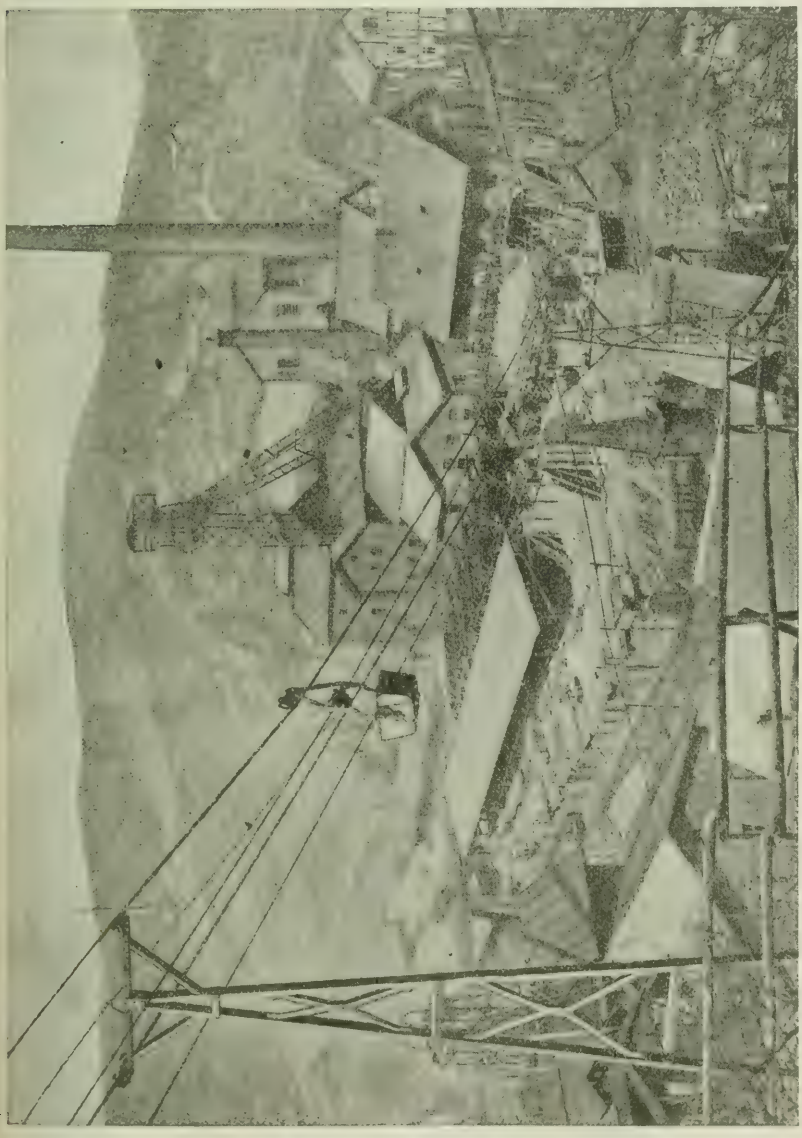


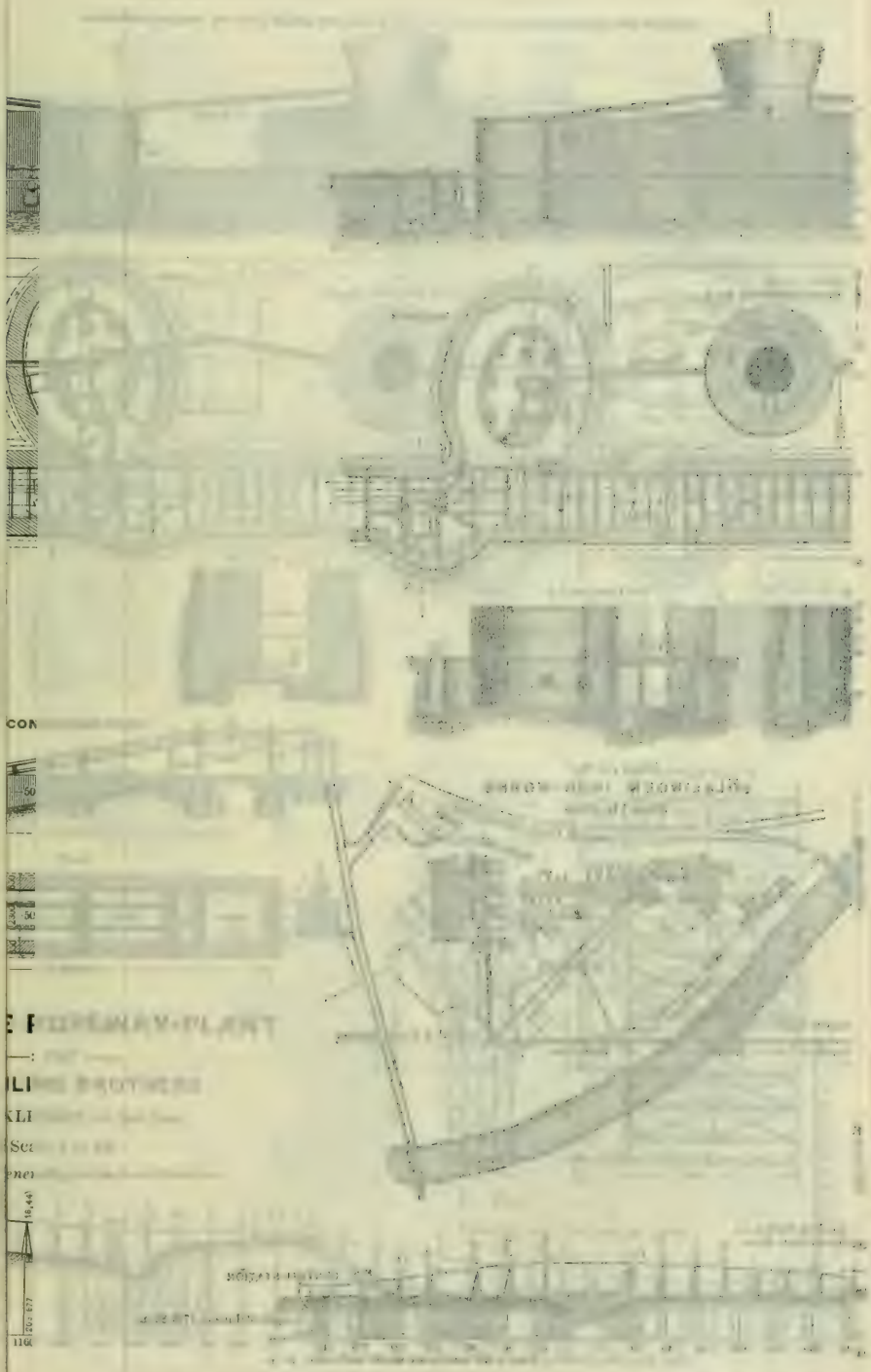
FIG. 19.

Marienhütte Line of the "Otto" Aërial Ropeway, near Siegen, Westphalia. View at the Mine, Showing the Loading-Station for Ore and Intermediate-Station for Slag.

unknown and untried in this country, and that American engineers need to have their attention called to it and to be convinced of its

FIG. 20.

Marienhütte Line of the "Otto" Aerial Ropeway, near Siegen, Westphalia. View at the Iron-Works, Showing the Power- and Unloading-Station for Ore.



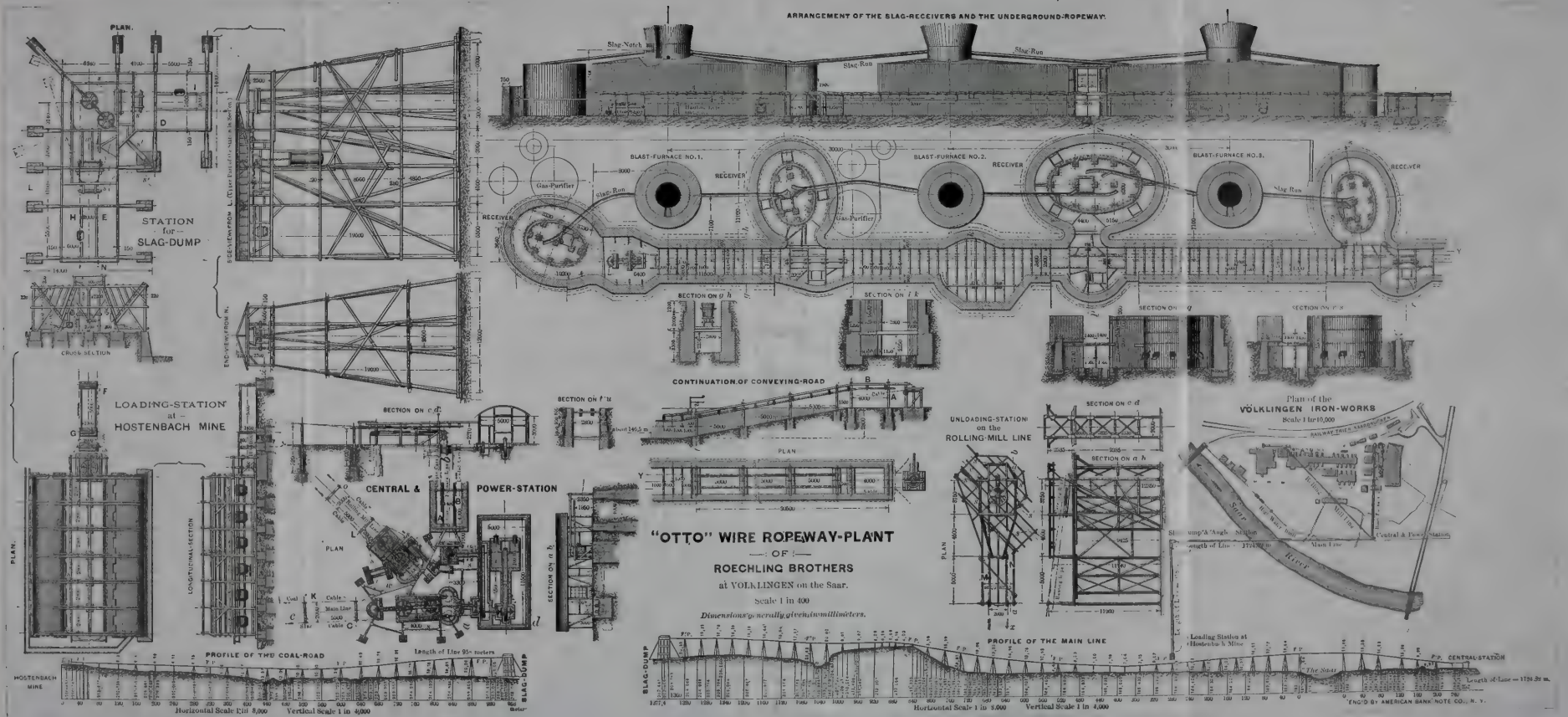
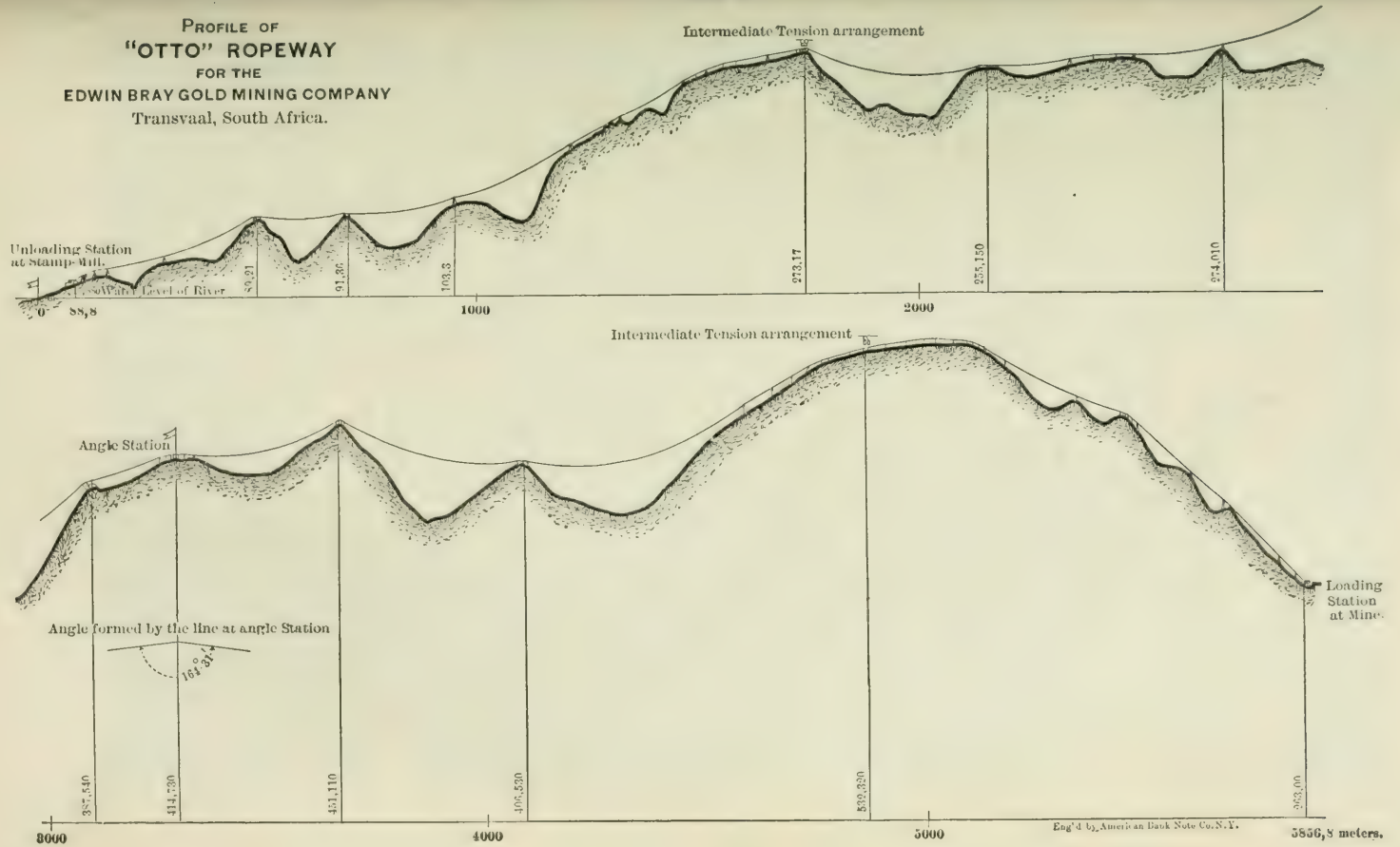

ARRANGEMENT OF THE SLAG-RECEIVERS AND THE UNDERGROUND-ROPEWAY.
PLAN.
STATION for SLAG-DUMP
CROSS SECTION
LOADING-STATION at HOSTENBACH MINE
PLAN.
LONGITUDINAL-SECTION
CENTRAL & POWER-STATION
PLAN
CONTINUATION OF CONVEYING-ROAD
PLAN
UNLOADING-STATION on the ROLLING-MILL LINE
PLAN
"OTTO" WIRE ROPEWAY-PLANT
— OF —
ROECHLING BROTHERS
at VOLKLINGEN on the Saar.
Scale 1 in 400
Dimensions generally given in millimeters.
Plan of the VÖLKLINGEN IRON-WORKS
Scale 1 in 10,000
PROFILE OF THE COAL-ROAD
Length of line 95· meters
Horizontal Scale 1 in 8,000
Vertical Scale 1 in 4,000
PROFILE OF THE MAIN LINE
Horizontal Scale 1 in 8,000
Vertical Scale 1 in 4,000
ENGD BY AMERICAN BANK NOTE CO., N. Y.

FIG. 21.


PROFILE OF
"OTTO" ROPEWAY
FOR THE
EDWIN BRAY GOLD MINING COMPANY
Transvaal, South Africa.
Intermediate Tension arrangement
Unloading Station at Stamp-Mill.
Low Water Level of River
0
88,8
1000
2000
Intermediate Tension arrangement
Angle Station
Loading Station at Mine.
Angle formed by the line at angle Station
164° 31'
3000
4000
5000
5856,8 meters.
Engd by American Bank Note Co. N.Y.

merit. The Trenton Iron Company has been engaged for some years in the erection of aërial wire-rope tramways of this German type.

ON THE DARBY PROCESS OF RECARBURIZATION.

BY A. THIELEN, RUHRORT, GERMANY.

(Pittsburgh International Session, October, 1890.)

In experiments for the production of steel the principal problem has always been the introduction into, or removal from, the iron of a definite quantity of carbon. Although the solution of this problem had engaged a more or less blindly and empirically directed effort for a long time, it was only with the development of analytical chemistry that the *rôle* which carbon plays in the constitution of steel was disclosed in its full significance, and that many of the problems confronting the steel-maker were thereby more clearly defined than before.

For a long time, and to a certain extent even to this day, the percentage of carbon in steel has been the controlling means of determining its quality and hardness. In many cases the percentage of carbon alone has been relied on for judging large consignments, such as the large deliveries of wire-billets and blooms to America during the past twenty years; and though to-day the influences of the other constituents are recognized and many other considerations are taken into account, and though, moreover, some special steels, such as the Hadfield manganese steel, apparently dispense to a certain extent with the assistance of the carbon, yet for our Bessemer, Thomas, Siemens and crucible steel the carbon still remains one of the most important constituents, and I venture to say it is indispensable.

In order to produce in the Bessemer converter a steel of a certain required percentage of carbon, two methods, as is well known, present themselves. By the one the decarburization process is stopped when the desired carbon limit in the bath has been reached; by the other the bath is completely decarburized, and then a calculated amount of carbon added in the form of spiegel, ferro-manganese, etc. In Germany the latter method only was adopted, and it was retained when the Thomas-Gilchrist process for the production of low-phosphorus steel from high-phosphorus iron came into use, since from the nature of the new process complete decarburization is neces-

sary before dephosphorization can take place. But while in the old acid process the carburization could be easily effected by the simple addition of spiegel, etc., the new one brought difficulties which were not easily overcome, inasmuch as the phosphoric acid of the slag and the oxides whose presence in large but varying proportions is characteristic of the basic process, took part in the reaction. Though a portion of the oxide could be removed by pouring off the slag, and the amount of it left dissolved in the bath could, by improvements in the process, be reduced to an approximately constant quantity in each charge, and even though the danger of rephosphorization was diminished by as complete a removal of the slag as possible, and by suitable additions to that portion of it still remaining in the bath, nevertheless the production of low-phosphorus steel high in carbon was not accomplished with that degree of certainty and facility which would lead one to regard the introduction of another, simpler method of carburizing as other than very desirable. The circumstances governing the basic and acid open-hearth processes in these respects were of the same nature.

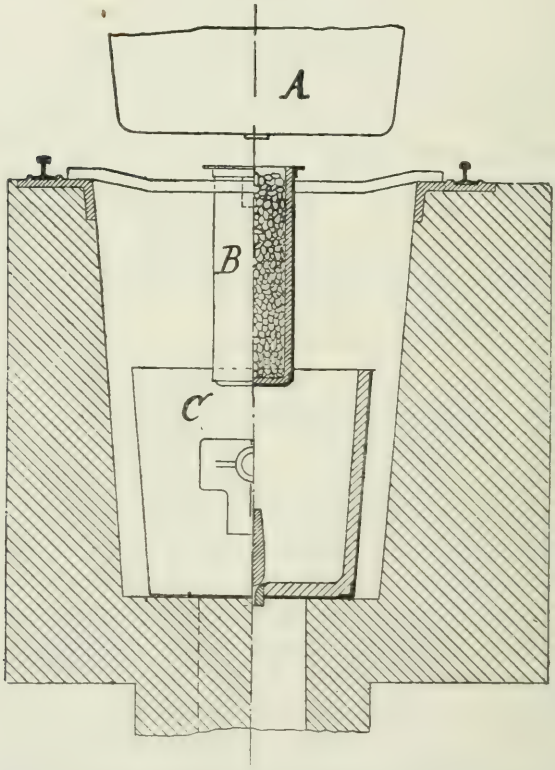
It will be readily understood, therefore, why methods which had formerly been proposed for the introduction of carbon by other than the indirect way (through the addition of spiegel, etc.), but which had hitherto remained without results, should, after the introduction of the method of making steel in basic-lined vessels, be brought forward again from various quarters. All of these processes were based on the great affinity between carbon and iron at high temperatures. It was sought to reduce and carburize the bath of steel by the introduction of carburetted gases, tar, petroleum, etc., as well as by the addition of many mixtures of these substances with solid bodies, such as burnt dolomite. But the results of all these experiments did not lead to the permanent introduction of any of them until Mr. John Henry Darby, the Managing Director of the Brymbo Steel Works, succeeded in satisfactorily solving the problem.

Having noted an increase of carbon in two bars which had been heated in a coal-fire for welding, he tried the effect of bringing fluid steel into intimate contact with solid carbon, and found a rapid absorption of the carbon by the steel. Upon this experiment he based his patented process, in which fluid steel is carburized by filtration through pieces of carbon, the latter being preferably in the form of graphite or wood-charcoal.

The first apparatus which he employed is shown in Fig. 1. It consisted of a sheet-iron cylinder, B, open at the top and closed at the

bottom by a perforated plate, the whole vessel being lined with refractory material. It was filled with the carburizing material and then placed between the two steel-ladles, A and C, or between the open-hearth furnace and the steel-ladle. The fluid metal was poured through this carbon-filter, and by the consequent absorption of carbon it was brought to the required percentage.

FIG. 1.



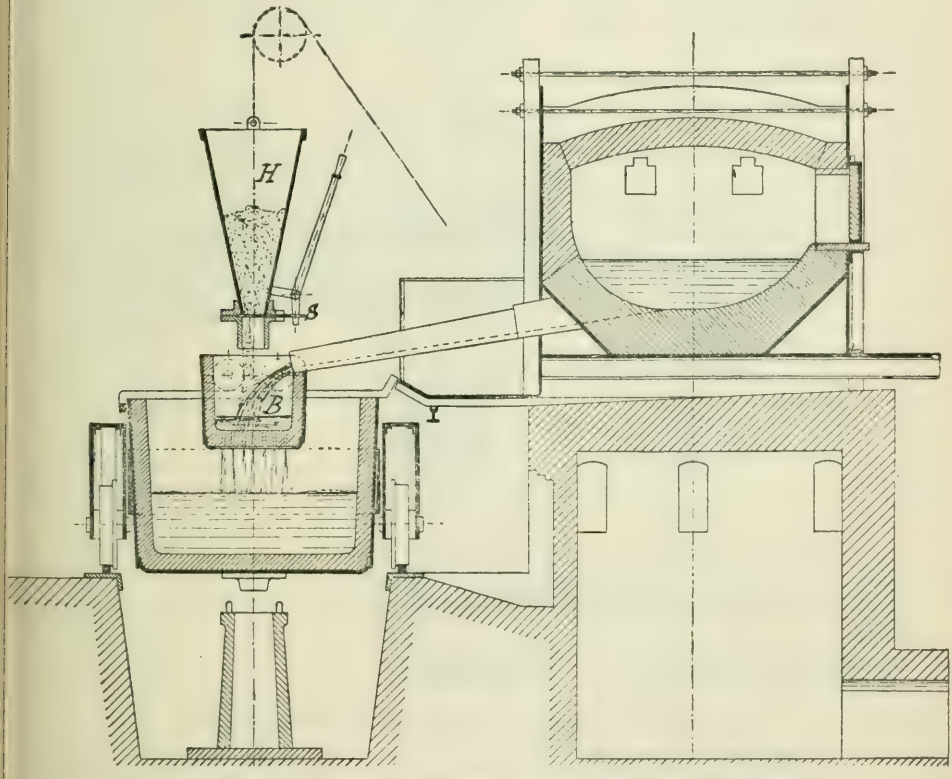
First Form of Carburizing Apparatus for Open-Hearth Practice.

Further experiments demonstrated that the absorption of the carbon was so rapid as to render unnecessary the prolonged contact afforded by the above described method of filtration, and hence Mr. Darby constructed the apparatus shown in Fig. 2.

In this the cylindrical filter is replaced by the vessel B, which is conveniently supported between the open-hearth furnace and the steel-ladle, or between two steel-ladles. Like the first filter, it is also lined with refractory material and furnished with a perforated

refractory bottom-plate. Above this vessel is placed a feed-hopper, H, containing the roughly broken carburizing-material. This receptacle is fitted with a slide-valve, S, the opening of which permits the carbon to be discharged at any desired rate into the vessel, K, where it mixes with and carburizes the fluid steel, which then flows out through the holes in the bottom-plate. As a rule the carburization is effected during the first third of the cast. The remainder of the charge of soft steel is then mixed with the already

Fig. 2.



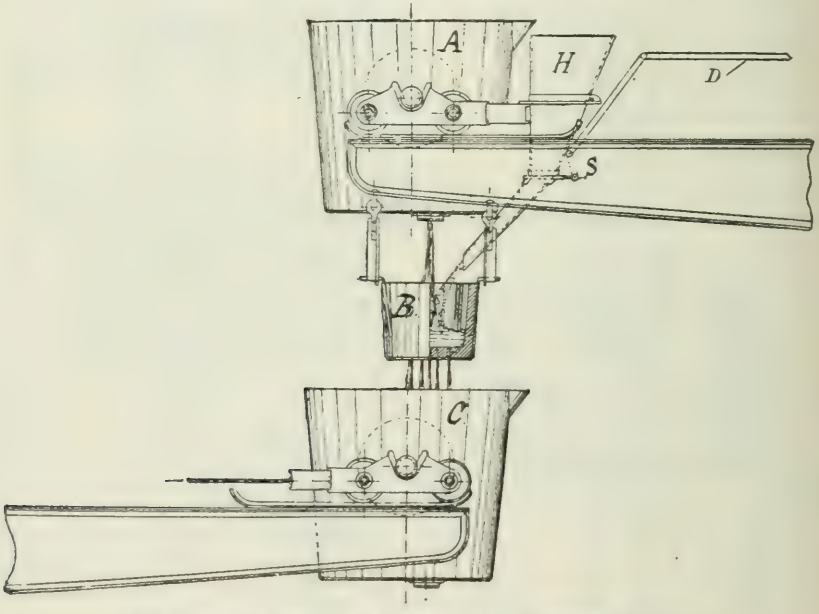
Second Form of Carburizing Apparatus for Open-Hearth Practice.

carburized metal in the ladle to form a homogeneous whole. When graphite is employed as the carburizing agent, the loss by burning is 15 to 20 per cent., but it is greater when coke is used.

Mr. Darby's experiments were confined to the carburizing of open-hearth steel. By the use of his process in conjunction with the basic, he succeeded in producing from phosphoric raw material

an excellent product, which with any desired percentage of carbon up to 0.9 per cent., contained only small traces of other substances, and was distinguished by its exceeding toughness from all previously known kinds of open-hearth metal. It has been worked into chisels, knives, wire, etc., with the best results. The certainty of obtaining the required percentage of carbon in the metal was very great, the range of variation from the desired limit rarely exceeding 0.01 per cent. to 0.02 per cent. The rolling-properties of the material were excellent, even with very small additions of ferro-manganese or ferro-silicon. For the harder steels the process was soon employed at Brymbo to the exclusion of any other.

FIG. 3.

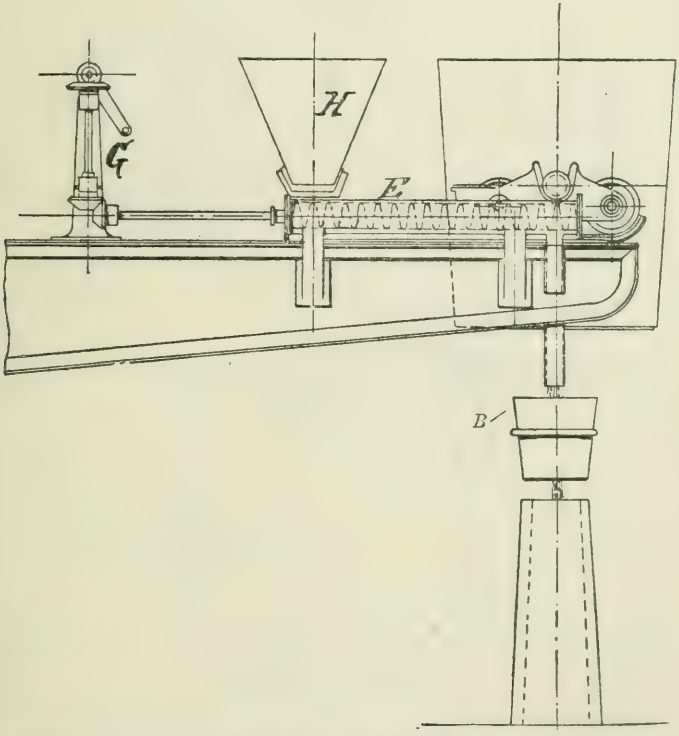


First Form of Carburizing Apparatus for Bessemer Practice.

In order to apply his process to the Bessemer, Mr. Darby entered into an agreement, in 1889, with the Phœnix Company, of Laar, near Ruhrort, on the Rhine. Experiments were made in the basic Bessemer works with an apparatus of the type described in Fig. 2. to which, however, the arrangement shown in Fig. 3 was given. The carburizing vessel, B, was placed between the two ladles, A and C, and while the steel flowed from the first into the second ladle, it was carburized by material discharging from the hopper, H, at a

rate regulated by the valve-handle, D. It soon became manifest, however, that though the basic converter-steel could be readily carburized by the process, it lost its rolling-properties to such an extent that the ingots fell to pieces in the rolls. The cause of this trouble was discovered to be that for four minutes more than sixty fine streams of steel were subjected to the oxidizing influence of the atmosphere. This difficulty was met by substituting for the 60 holes in the bottom of the vessel a single one of adequate diameter; a considerable improvement in the rolling-property of the ingots was

Fig. 4.

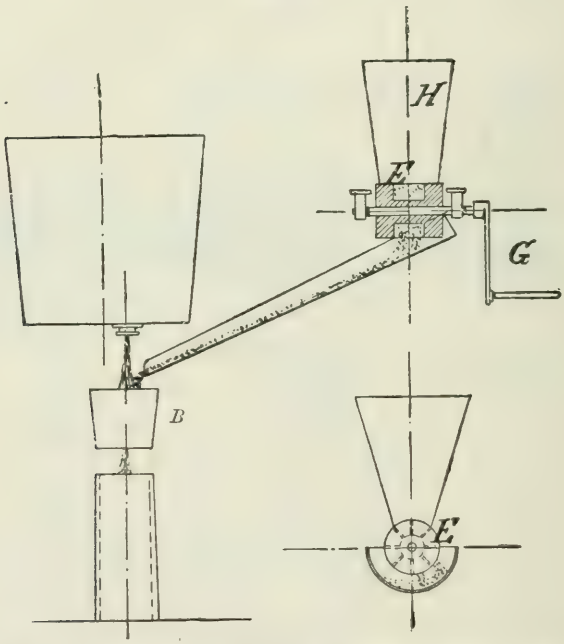


Second Form of Carburizing Apparatus for Bessemer Practice, showing Worm-Feed.

effected, but yet it was not as good as that of the ordinary Thomas ingots, and hence the apparatus was superseded by the arrangement shown in Figs. 4 and 6. Here the carburizing-vessel was replaced by a refractory-lined funnel, B, provided with a hole at the bottom, and set between the casting-ladle and the moulds. By a feed-

arrangement attached to the side of the casting-crane, the ground carburizing-material could be regularly supplied to the funnel, B. This feeding device consists of an accurately bored cylinder, E, provided with a lathe-cut worm, the hopper, H, and the turning-gear, G. The amount of carbonaceous material discharged from E is obviously proportional to the revolutions of the endless screw, and hence can be accurately controlled. By the employment of this apparatus the carburizing is successfully accomplished, and ingots of good rolling-properties are obtained. As ground coke was found to

FIG. 5.



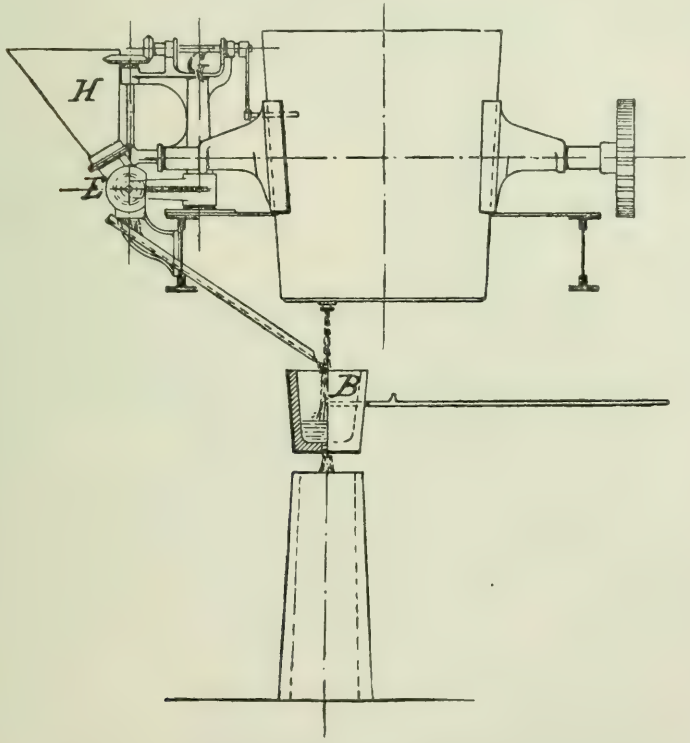
Modified Second Form of Carburizing Apparatus for Bessemer Practice, showing Fan-Wheel Feed.

be more quickly absorbed than graphite, all the experiments were conducted with the former, and this may possibly account for the very even distribution of carbon in ingots weighing as much as 1400 kilos each and measuring 16 inches square. The increase of carbon not infrequently found in the tops of ingots of this weight was overcome by the exercise of a little care in diminishing the amount of the carburizer added to the upper portion during the cast. In order to test the equal distribution of the carbon, ingots

were rolled down to 2-inch billets and tests taken from the two ends and the middle of each.

Fig. 5 shows another arrangement for carrying out the process. The carburizing agent drops from the hopper, H, into the fan-wheel, E. By turning this wheel with the crank, G, the filled compartments discharge themselves and supply the material in regulated proportion to the vessel, B. The results obtained from a trial of this apparatus are given in Table I., on page 801.

Fig. 6.



Second Form of Carburizing Apparatus for Bessemer Practice, showing Worm-Feed.

Although this method of carrying out the process was an advance on all earlier methods, yet it had the disadvantage of depending on the workman who revolved the feed-screw for the proper degree and uniformity of the carburization. To effect the rotation by mechanical means appeared impracticable, because the feed should be continually adjusted to the changing stream of molten steel, and must

therefore vary with the temperature of the metal and condition of the tap-hole. The observation that the carburizing-material, and more particularly the finely-ground coke, is almost immediately absorbed at its first contact with the molten steel, led to an arrangement, shown in Fig. 7, by which this last difficulty finally was obviated. The carburizing agent was added direct to the stream of molten metal discharging from the converter before it reached the steel-ladle, while the slag was retained in the converter until the carburization was complete, by placing a block of refractory material or a suitably formed guard, E, protected by a refractory covering, at the mouth of the vessel. The quantity of the carburizer delivered from the hopper, H, is controlled as before by the slide-valve, S.

Some of the results obtained in operating by this method are given in Table II., on page 802.

Seventy per cent. of all the rail-steel charges blown at the Phœnix works (Fig. 7) during the month of June, 1890, were produced in this way, and also a number of charges of other grades of Thomas-steel for tempering and welding, all of which gave satisfactory results. In using this method of carburizing with the Thomas process the same addition of ferro-manganese is required as is usually necessary for charges of soft steel not previously carburized, or for harder grades of steel previously carburized with spiegeleisen.

After a series of very successful results had been obtained in this simple way, the practice of the process was extended to the open-hearth and acid Bessemer departments. The results obtained have been so favorable that the employment of a special carburizing-vessel has been altogether discarded in the works.

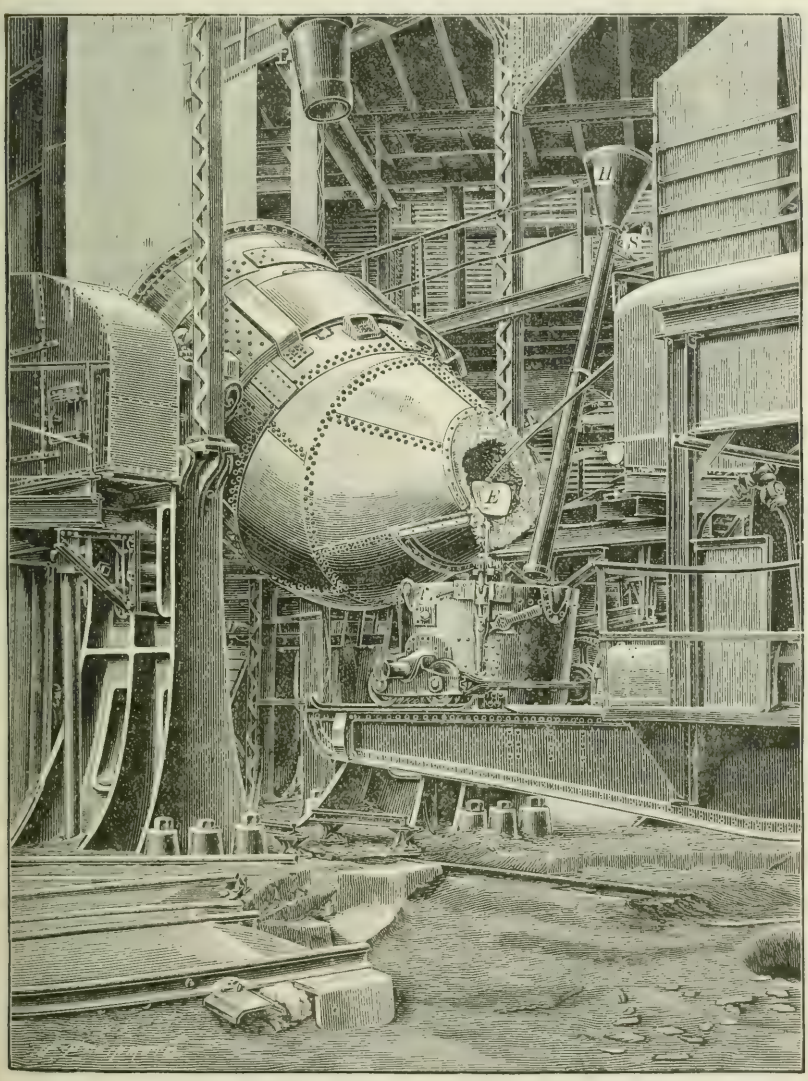
Summarizing the advantages of this carburizing process in each of the methods for producing steel, we find :

1. In the basic Bessemer or Thomas process, that carburizing takes place in almost the entire absence of the slag (which is rich in oxides and phosphoric acid), and hence that the action proceeds with certainty, and is accompanied by no noteworthy re-phosphorization, and that it is practicable to carry it up to any desired percentage of carbon without simultaneously increasing the percentage of manganese. By dispensing with the addition of spiegel a considerable economy is effected.

2. In the acid Bessemer process, that the carburization up to the highest grades of hardness is effected with far greater certainty than by the aid of spiegel, and without the increase in manganese which

is unavoidable in the ordinary process. Here, also, a considerable economy results from the saving of spiegel. A number of trials

Fig. 7.



Latest Form of Carburizing Apparatus for Basic Bessemer Practice, at the Phœnix Steel-Works, Ruhrort, Germany.

have been made at American works, the results of which are given in Table IV., on page 803.

3. In the acid and basic open-hearth processes, that the advantages secured are very nearly identical with those obtained for the acid and basic Bessemer processes, the very considerable cost of ferro-manganese and ferro-silicon being either wholly or largely avoided. The use of this carburizing process in conjunction with the basic open-hearth affords means for the production of a steel which can compete for many purposes with crucible-metal. By this method, moreover, a material can be easily produced which will probably prove valuable as a raw material for the crucible process.

To illustrate the saving resulting from the use of the new process, the following cases are cited from practice:

The production in the basic converter of 9 to 9.5 tons of rail-steel having a tensile strength of 50 kgs. per sq. mm. (71,115 pounds per square inch),* requires:

By the ordinary process:

Spiegeleisen, 10–12 per cent.,	600 kgs. (1320 lbs.)
Ferro-manganese, 60–65 per cent.,	80 " (176 ")

By the carburizing process:

Spiegelesen,	nil.
Ferro-manganese, 60–65 per cent.,	80 kgs. (176 lbs.)
Coke,	60 " (132 ")

The calculation for acid working likewise results in favor of the carburizing process.

If steel of the same hardness is to be produced in the basic open-hearth furnace, a charge for a ten-ton output will require:

By the ordinary process:

Spiegeleisen, 10–12 per cent.,	250 kgs. (550 lbs.)
Ferro-manganese, 60–65 per cent.,	80 " (176 ")

By the carburizing process:

Spiegeleisen,	nil.
Ferro-manganese, 60–65 per cent.,	Up to 12 kgs. (26 lbs.)
Coke,	50 " (100 ")

In this case, therefore, the economy resulting from the employment of the carburizing process is likewise considerable, and the same holds true for the acid open-hearth process.

* A stress of 1 kilogramme per square millimeter is equivalent to 1422.3 pounds per square inch.

It is evident from the above that in the point of cost the advantage of the carburizing process increases with the hardness of the material to be produced. To its superiority in affording a means for producing the harder grades of steel with far greater ease and certainty, should therefore be added the very considerable economy in the cost of production, in contrast with which the cost of the old process rose considerably with the grade of hardness.

During the latter part of last July an experiment was made in connection with the carburizing process, the result of which has been very satisfactory. In our manufacture of tires the ingots are hammered into flat disks, which are perforated in the middle, and the hole is enlarged before the disk is reheated for rolling. This mode of manufacture demands a perfectly even, non-porous top surface for the ingot, as any deficiencies would be detected in the finished tire. Although open-hearth steel, directly carburized, is comparatively quiet, yet as it is not absolutely dead immediately after pouring, it was not deemed advisable to use it as a material for tires. Experiments, however, were made by adding small proportions of aluminum in the bath, and it was found that an addition of at most 0.04 per cent. of aluminum resulted in an ingot which was absolutely free from blow-holes and cavities, and eminently suited for the manufacture of tires. In further experiments with aluminum several castings were made of the carburized metal, and these proved equally satisfactory, being positively without flaw.*

TABLE I.—*Showing the Results of Carburizing Basic Converter-Metal by the Second Method, Fig.* 5.

Ingot No.	Carbon required, per cent.	Carbon Obtained, per cent.		
		Number of Ingot-Section.		
		A.	B.	C.
1	0.25	0.255	0.260	0.251
2	0.25	0.244	0.248	0.254
3	0.30	0.285	0.311	0.306
4	0.35	0.330	0.324	0.340
5	0.35	0.350	0.357	0.352
6	0.45	0.461	0.455	0.470

* An interesting collection of samples of steel carburized by the Darby process was exhibited by the author, and upon the conclusion of the reading and discussion of his paper, was donated by him to the Engineers' Society of Western Pennsylvania, in whose behalf the gift was suitably acknowledged by Prof. J. W. Langley.

TABLE II.—*Showing the Results of Carburizing Basic Converter-Metal by the last Method, Fig.* 7.

No. of Heat.	CARBON, PER CENT.			PHOSPHORUS, PER CENT.	
	Before Carburization.	After Carburization.		Before Carburization.	After Carburization.
	First Ingot.	First Ingot.	Last Ingot.	First Ingot.	
1	0.073	0.285	0.290	0.040	0.040
2	0.070	0.268	0.266	0.052	0.070
3	0.082	0.293	0.290	0.045	0.054
4	0.078	0.301	0.320	0.038	0.046
5	0.080	0.274	0.276	0.049	0.049
6	0.091	0.315	0.320	0.074	0.085
7	0.083	0.298	0.290	0.063	0.072
8	0.095	0.274	0.277	0.048	0.049
9	0.072	0.284	0.279	0.055	0.067
10	0.083	0.269	0.266	0.042	0.050
11	0.089	0.300	0.312	0.044	0.056
12	0.077	0.323	0.320	0.073	0.075

TABLE III.—*Showing the Results of Carburizing Basic Open-Hearth Metal.*

No. of Heat.	CARBON REQUIRED, Per cent.	CARBON OBTAINED, PER CENT.		
		Before Carburization.	After Carburization.	
	First Ingot.	First Ingot.	First Ingot.	Last Ingot.
1	0.51	0.075	0.510	.....
2	0.60	0.072	0.592	0.592
3	0.65	0.069	0.668	0.660
4	0.70	0.076	0.708	0.700
5	0.80	0.070	0.804	0.804

TABLE IV.—*Showing the Results obtained by Carnegie, Phipps & Co. in Carburizing Acid Converter-Metal.*

No. of Heat.	Carbon Required, per cent.	Carbon Obtained, per cent.
1	0.30 to 0.32	0.30
2	0.28 " 0.30	0.27
3	0.28 " 0.30	0.32
4	0.36 " 0.42	0.37
5	" " "	0.39
6	" " "	0.42
7	" " "	0.43
8	" " "	0.42

TABLE V.—*Showing Results of Carburizing Acid Open-Hearth Metal.*

No. of Heat.	Carbon Required, per cent.	Carbon Obtained, per cent.	
		Beginning of cast.	End of Cast.
1	0.30 to 0.32	0.30	
2	0.28 " 0.30	0.27	
3	0.28 " 0.30	0.32	
4	0.36 " 0.42	0.37	
5	" " "	0.39	
6	" " "	0.42	
7	" " "	0.43	
8	" " "	0.42	
9	0.50	0.51	
10	0.60	0.592	0.592
11	0.65	0.668	0.660
12	0.70	0.703	0.700
13	0.80	0.804	0.804

DISCUSSION.

PERCY C. GILCHRIST, London, England: Allow me to congratulate Herr Thielen on the very clear way in which he has put the matter before us. I fully endorse the necessity for the process. It is well known by all who have had anything to do with the basic process that it is remarkably easy to make very soft steel, and

very difficult, requiring a great deal of skill, to make steel with a regular percentage of carbon above 0.45 or 0.50 per cent. at the outside. Consequently, when Mr. Darby at the outset told me of this process I immediately went to see it, and have followed it step by step with the greatest interest. You will remember, those of you who are conversant with the basic open-hearth and the basic converter processes, that in order to perfect them, two discoveries were required in the case of the basic open-hearth, and three discoveries in the case of the basic Bessemer. This Darby process for giving a regular amount of carbon from 0.1 up to 0.9 or 1 per cent., is one of those discoveries. Mr. Darby carried on the process, as the author tells you, in the basic open-hearth; and Mr. Thielen was so satisfied with the results that he thought it possible to apply it with success to the basic Bessemer; and great credit is due to him and his staff for the successful way in which they have overcome some of the difficulties which were encountered in applying the new process to the basic Bessemer. In July of this year Mr. Arthur Cooper, General Manager of the North-Eastern Steel Works (which you will remember is a basic Bessemer plant, producing some 3000 tons of basic ingots a week) went with me to Ruhrort, where we were permitted to inspect freely the operations in the basic open-hearth and also in the basic Bessemer. I will not weary you with the details of what we saw. They are much more clearly put in the paper that we have just heard than I could set them before you now. Everything that Mr. Thielen has said in his paper agrees with what we found for ourselves on the occasion of our visit. So satisfied was Mr. Cooper with the results that within a very few weeks—I am speaking from memory, but I am inclined to think within a week—of his return, he carburized some hundreds of tons by this method at the North-Eastern Steel Company's works; and, knowing that this paper was promised, I requested him to give me some results that might be taken as typical of all his results. You will have noticed that the German experience showed a slight, but only a very slight, rephosphorization. Mr. Cooper's experience up to the time that I left England showed no rephosphorization at all. To that extent it differs, and is an improvement on the German experience. On September 16, 1890, Mr. Cooper sent me the following record of his results:

Recarburized Ingots.

Blow 370.				Blow 378.			
Ingot No.	Carb.	Phos.	Mn.	Ingot No.	Carb.	Phos.	Mn.
1	.44 per ct.	.06 per ct.	.....	1	.32 per ct.	.06 per ct.	.....
2	.47 "	.06 "	.....	2	.32 "	.06 "	.....
3	.47 "	.06 "	.69 per ct.	3	.27 "	.07 "	.58 per ct.
4	.48 "	.06 "	.....	4	.35 "	.06 "	.....
5	.48 "	.06 "	.....	5	.32 "	.07 "	.....
6	.48 "	.06 "	.66 per ct.	6	.33 "	.07 "	.46 per ct.
7	.47 "	.06 "	.....	7	.32 "	.07 "	.....
8	.44 "	.06 "	.....	8	.35 "	.07 "	.....
9	.48 "	.06 "	.58 per ct.				

Mechanical Tests.

Date.	Blow No.	Ingot No.	Length of Test.	Bearing.	Ball.	Drop, 20 feet. Deflection in inches.			
						No. 1.	No. 2.	No. 3.*	No. 4*
July 24	370	1	30′ 0″	3′ 6″	16 cwts.	$1\frac{5}{8}$	$3\frac{1}{2}$	Stood.	Stood
"	370	5	"	"	"	$1\frac{3}{16}$	$3\frac{1}{16}$		
"	378	10	5′ 0″	"	"	$4\frac{3}{8}$	$8\frac{3}{16}$		
"	378	11	"	"	"	$4\frac{5}{16}$	8		
"	378	12	"	"	"	$4\frac{1}{4}$	$8\frac{3}{16}$		
"	378	13	"	"	"	$4\frac{5}{16}$	$8\frac{5}{16}$		
"	378	14	"	"	"	$4\frac{3}{16}$	$8\frac{7}{16}$		
"	378	15	"	"	"	$4\frac{5}{16}$	$8\frac{3}{8}$		
"	378	16	"	"	"	$4\frac{1}{4}$	$8\frac{1}{16}$		
"	378	17	"	"	"	$4\frac{3}{8}$	$8\frac{1}{16}$		

* Rail reversed for last two blows.

In blow 370 the amount of carbon aimed at was 0.45 to 0.50 per cent. The actual results showed 0.44 of carbon as a minimum and 0.48 as a maximum in the nine ingots. The phosphorus in each ingot appears to be .06 per cent. The manganese varies from .58 to .69. The mechanical test for this charge was 30 feet long, 3 feet 6 inches bearing, 20 feet drop, and weight 1600 pounds. There were four blows. The first blow gave $1\frac{5}{8}$ inches deflection; the second, $1\frac{1}{2}$ inches. The rail was then reversed and had two more blows, each of 20 feet, and it did not break.

In blow 378 the carbon aimed at was 0.30 to 0.35. The carbon

obtained was 0.27 for the lowest result and 0.35 for the highest. The phosphorus varied from .06 in the first, second and fourth ingots, to .07 in the other five ingots. The manganese was .46 to .58. The result of the falling-weight test, as shown in the table, is exceedingly regular.

Mr. Cooper tells me he has found that special care is required in the selection of the carburizing-agent. His experience thus far has pointed to the avoidance of material containing much ash or oxide of iron. Observing all proper precaution in this respect, we shall be able to obtain what, I think, is the right product—an alloy of iron, carbon and manganese, which will roll like copper.

G. J. Snelus, Workington, England: Many of our members will remember that so long ago as 1878, when I read my first paper upon the basic process, I pointed out that a possibility was then presented for making a steel which would exceed in quality the famous tool- and cutlery-steel made from Swedish material, inasmuch as we had become able to produce a metal which contained less phosphorus than that celebrated product from Sweden. The first piece of steel which was made in 1872 by the basic process contained about .001 per cent. of phosphorus from a metal containing $1\frac{1}{4}$ per cent. Such results were repeatedly obtained by my friends and co-workers, Mr. Thomas and Mr. Gilchrist; and it occurred to me very early that if it was possible to get a basic material so pure as this, it must undoubtedly take the place of even Swedish metal.

Mr. Darby very kindly allowed me to see this process at the Brymbo works last year; and in our journey over here to America he very kindly admitted that he was working upon the suggestions which I then threw out. But of course I admit that the point of importance is his discovery that solid carbon will recarburize this steel. It was quite evident that you could recarburize a pure metal with other compounds—take cemented steel, for instance; but Mr. Darby's use of the solid carbon is a very great step in advance; and there can be no doubt that it will tend to revolutionize the manufacture of steel.

I venture to say, however, that I think its most important application will be in the manufacture of fine steel; because I found years ago, in investigating the difference between steel made from our Cumberland iron containing, say .003 per cent. of phosphorus, and that made from Swedish iron containing about one-half the quantity, that it was the phosphorus in that immeasurably small quantity which prevented our Cumberland iron from being used for cutlery-

steel, because that small quantity of phosphorus produced brittleness and prevented our getting a keen cutting edge. When I had satisfied myself of this fact, it naturally occurred to me that if we could produce a steel with as small a quantity of phosphorus as we know we now can, that this would ultimately be the basis for fine steel. It did not occur to me that we should attempt to use this process of recarburizing for the coarser grades of steel for rails and for plates; but from the discovery of Mr. Darby it appears that this is practicable, and it certainly reflects great credit upon him and upon our worthy President, Mr. Thielen, for the clever way in which they have carried the process to success.

P. C. GILCHRIST, London, England (communication to the Secretary): To substantiate my statement at the meeting regarding the entire absence of rephosphorization, I have obtained from Mr. Cooper, of the North-Eastern Steel Co., the following figures giving the phosphorus in the metal both before and after carburizing:

No. of blow	82,764	82,766	82,768	82,774	82,780	82,781	82,782	82,791	82,794	82,798
Phosphorus in soft steel, before carburizing, pr. ct.	.06	.07	.06	.07	.07	.07	.07	.07	.07	.07
Phosphorus in hard steel, after carburizing, pr. ct.	.06	.07	.06	.07	.07	.07	.07	.07	.07	.07

It is evident, therefore, that the steel can be carburized without rephosphorization, but to attain this result great care is required to keep back the whole of the slag in the converter.

BRITISH CONTRIBUTIONS TO THE METALLURGY OF IRON AND STEEL.

BY SIR JAMES KITSON, BART., LEEDS, ENGLAND.

(Opening Address at the Pittsburgh International Session, October, 1890.)

BY the courtesy of the President and Council of the American Institute of Mining Engineers, I have been invited to take the chair and open the proceedings of this congress. It is a graceful compliment to those members of the Iron and Steel Institute who are your honored guests at this moment—an acknowledgment of high consid-

eration for scientific services rendered to a great industry by the Iron and Steel Institute and a mark of distinction conferred upon its President. Though conscious of my own deficiencies, I gratefully accept the position, feeling that the Institute is worthy of the respect you show it; for we count among our members men who merit the honors which you offer, men whose names are known throughout the world as those of leaders in the scientific study of the laws which govern the acquisition and production of the metal which we know as iron and steel.

Our Institute is not exclusively British; in truth, it is cosmopolitan, having enrolled on its list of members distinguished metallurgists of the United States, Germany, France, Sweden and Russia. The birthplace and headquarters of the Institute are in England. May I then, without incurring a verdict of unseemly presumptuousness, ask you, for a moment, to consider how much the world is indebted to my native land for the vast benefits which mankind has received from its engineers and its manufacturers of iron and steel?

The inventions of Watt and Trevithick, the development of the steam-engine, with all its consequential powers; the construction of the locomotive and the creation of the railroad by Stephenson, are directly the work of England and of Englishmen. These inventions have created that system of railways and service of ocean steamers which have brought the nations of the earth together and made new lands accessible and serviceable to man, and which are destined, in due time, to diffuse knowledge and civilization into every quarter of the globe.

The puddling-furnace of Cort and his method of rolling the puddled blooms in grooved rolls, produced wrought-iron in quantities and at a cost previously unattainable. Through these inventions, South Wales became the seat of the rail trade, and for many years it was enabled to dominate the rail trade of the world. The puddling-furnace continued, from the latter part of the last to the middle of the present century, to be the instrument by which practically all the malleable iron in the world was made.

The hot-blast process of Neilson, patented in 1828, secured a great economy of fuel and a large increase of production. It entirely altered the condition of the pig-iron manufacture, and was truly ranked by Mushet as "a means of developing the national wealth, of equal value with Arkwright's invention of cotton-spinning."

The economy effected in the use of coal in the various processes of the manufacture of iron and steel has been great and progressive.

The margin for economy which then existed has been narrowed, but still, as we know on the authority of Sir Lowthian Bell, the duty performed is by no means equivalent to the whole of the useful effect which coke is capable of affording. "Taking a ton as being able to produce twenty hundredweight of pig-iron from Cleveland ironstone; this, after an allowance for ash, etc., when burnt to carbon dioxide, represents 147,200 calories, whereas something like 90,000 calories has been the limit of the power obtained." This sacrifice of 40 per cent. lends itself to further reduction, as does the heat held in the stream of molten slag flowing from the furnace.

Nasmyth's steam-hammer gave the means of forging iron, which led to rapid developments and improvements in the applications of iron. The reversing rolling-mill engine of Ramsbottom has been invaluable in the rolling of plates and bars of great dimensions. The forging-press, too, has been designed and improved by English inventors, to whom the world owes much for other mechanical appliances devised to meet the various wants of the Bessemer process and the treatment of massive iron and steel.

To you, the most important invention of modern times has probably been that of Sir Henry Bessemer. It has given you a material in quality and quantity for your railroads which it was a physical impossibility to obtain by the ancient methods of iron-making. It is no exaggeration to say that without the Bessemer process for steel rail making, the present railway-system of the new lands you have peopled, and the binding together of the vast territory between the Atlantic and Pacific under one civilized power would not yet have been effected. Sir Henry Bessemer has prepared, at my request, for communication to this meeting, an account of his discovery of the process of manufacturing what is known as Bessemer steel; and some history of his investigations and the way in which, step by step, he overcame the difficulties which he encountered in the development of his remarkable invention.

It reads as follows:

(1.

Denmark Hill.
September 10th 1890

Dear Sir James Kitson,

It is with great pleasure that I accede to your request, to furnish you with a brief outline of the circumstances which led to the invention of my Steel process; the more so on account of the generous interest you have always taken in the invention, notwithstanding that it aims at the overthrow of the older method of producing malleable iron; in the production of which you have obtained so distinguished a position. -

At the time of the Crimean War I had invented a mode of firing elongated projectiles from a smooth bore gun, the rotation necessary to

2.)
insure their proper position during flight being obtained by utilizing a portion of the powder-gas to produce rotation by reaction; after the manner of producing rotation in Durbins and not by rifling the gun, consequently rendering all smooth-bore guns at once suitable for firing elongated shot or shell.
I of course offered this plan to our own Government, but it was discarded without a trial.

A little after this period I happened to be on a visit for a week or two with Laird James Hay at the house of his daughter in Paris.
During this visit our host gave a farewell dinner to General Canrobert, and several other distinguished officers in the French army, who were going out to the Crimea;

Among the guests present on this occasion was Prince Napoleon; and while taking a cigar in the Library after dinner, the conversation

turned naturally enough on artillery, and I (3)
then mentioned my system of firing elongated
projectiles from smooth-bore guns,
The Prince was so impressed with the impor-
tance of this idea, that he said he was sure
that his cousin the Emperor, would be much
pleased if I would explain my invention to
him; and that he would get an appointment
made with the Emperor for that purpose.
This was done, and I had a long and most inter-
esting discussion with his Majesty, whom
I found most thoroughly conversant with the
whole subject of artillery,
His Majesty, in the kindest possible way, gave
me a carte blanche to make any experiments
I desired at Vincennes.
A great many 30 pounder elongated projectiles
were made, and were fired from a 4½ inch 12
pounder smooth-bore cast-iron gun, at the

4.)

Polygon at Vincennes; they were fired through a succession of thin wooden targets, placed 100 metres apart, through all of which they cut circular holes, thus showing that they went end on.

A thin coat of Black Japan had been previously put on them, and when the shots were recovered from the bank into which they were lodged, the coat of Japan was seen to be partially scraped off in spiral lines, caused by their passage through the wooden target; the angle of these scratches being carefully taken showed that from 1½ to 2¼ rotations had taken place in the length of the gun.

After many hours practice on a cold December day, we retired to the Officers quarters in the old Fortress of Vincennes, and while standing round a blazing wood-fire supping some hot spiced-wine, Commandant Minié (the inventor of the rifle) observed,

(5.

"that although the rotation of the shot was
"effected, unless we had something better to
"make our guns of, such heavy elongated
"projectiles could not be used with safety."

This casual observation was the spark that has kindled one of the greatest industrial revolutions that the present century has to record. For during my solitary ride in a Cab that night from Vincennes to Paris I made up my mind to try what I could do to improve the quality of iron used in the manufacture of Guns. —

My knowledge of iron metallurgy was at this time very limited, but this was in one sense a great advantage to me, for I had very little to unlearn; and so could let my imagination have full scope.

After many months of trial, and much building up, and pulling down of reverberatory furnaces I cast a small model Gun; the iron was very

8

wight

6.] and in turning it, little chard curley shavings were cut off, it was wonderfully tough for cast-iron, but wonderfully brittle if classed as malleable-iron,

The little model gun looked very beautiful when highly polished, and I took it over to Paris and begged the Emperor to accept it, as the first fruits of my studies of the Metallurgy of iron.

He expressed himself much pleased with it, and complimented me on having achieved a step in the right direction, and with his own hands placed it in a bearreau; saying, "Some day it may become an interesting relic."

About this period I began fully appreciate the fact, that if I could improve cast-iron and render it malleable, and still retain its fluid state, that apart from its use for artillery it would be of the greatest commercial value for all engineering purposes. — I therefore persued

(7.

my experiments with greater ardour than ever. For I was convinced that I was on the eve of producing a quality of metal that would supersede Wrought iron.

At this time I devoted myself exclusively to these investigations which were very costly, and I became most anxious to obtain the opinion of some able engineer as to the value of my invention, lest I might be deceiving myself and living in a fools paradise.

I consequently consulted Mr. Geo. Rennie the eminent civil-engineer, I shewed him a small upright fixed cylindrical converter, and in it we made a charge of 7 cwt. of Blaenavon pig-iron into an ingot of malleable-iron, —

Mr. Rennie was in raptures with the result, and said, "you must not keep this light under a Bushell for a single day longer" and by-the-bye there is a first rate opportunity for you,

8.)

The British Association met at Cheltenham next Tuesday, and a paper there by all means, "I am President of the Mechanical section, it is true "all the papers are arranged, but if you will write "a paper, I will take the responsibility of putting "it first on the list."

He kept his promise, and I read my paper "on the Manufacture of Malleable-iron without fuel" and which appeared verbatim in next days "Times"

The entire iron-trade of the Kingdom was startled by the facts detailed in this paper, backed as they were by two small bars of malleable-iron, one of which had been piled and re-rolled,* a few days later the iron-masters came trooping up to London to see the new process. There are many interesting incidents connected with these visits which I cannot trespass on your time to relate.

* one of these bars, in the possession of the Iron and Steel Institute among my old samples

(9.

but some idea of the excitement may be gathered from the fact that notwithstanding the imperfect state of the process at that time, I was actually paid £ 27.000. for Licences granted within one month of the reading of my paper at Cheltenham.—

At many iron-works the managers set to work to test the invention in the cheapest possible manner with such means as they had at their disposal, all of which attempts were failures.

In my experiments I had used Blaenavon pig-iron, which was successful; and at that time I had no idea that other brands of iron would fail in the manner they did.

No sooner were these failures known than an extraordinary revulsion of feeling was manifested, the most perfect distrust of the invention became universal, the public press, which had spoken of it in such glowing terms, now condemned it as impracticable, and spoke of it as —

10.) "A brilliant meteor, that had flitted across "the metallurgical horizon dazzling a few "enthusiasts and then vanishing for ever in "total darkness."

Prior to this invention pure malleable-iron in a fluid state was wholly unknown; indeed whole days of exposure to the most powerful furnaces then in use entirely failed to bring this material into a state of fusion, notwithstanding which I proposed to convert ordinary melted cast-iron into this malleable fluid state, in quantities of five Tons at a single operation, in the short space of fifteen minutes, by the mere chemical action of cold atmospheric air, and in entire absence of any fuel whatever, except that which existed in the form of carbon, and silicon, in the crude iron itself. It is not surprising therefore that the iron-masters, as well as the whole press

joined in one general chorus of condem- (11
nation of what they then believed to be a
perfect chimera, which none but a wild
enthusiast could have ever believed to be possible
I knew far otherwise, but it was no time to
argue the question, words would have been of no
avail; so I set earnestly to work to try and
master the difficulty that had so unexpectedly
arisen; this was no easy task.

All the old investigations had to be gone over
again, experiments on a much larger scale,
with larger and more powerful machinery, were
found necessary, and the difficulties to be
surmounted had reference more to chemical
than mechanical questions; a laboratory
was therefore fitted up, and I engaged the
services of a professor of chemistry, at a high salary
to make an analysis of all the iron and other
materials employed in our experiments, as well

12.)

as a systematic analysis and classification of all the results obtained.

The very large scale on which these operations were carried out involved a considerable outlay in various ways, but there was no slackening of exertion, no cessation of the severe mental and bodily labour; in this way another long and weary year had passed, and but little real progress had been made towards the removal of the difficulty; many new paths had been struck out, but they had led to no practical results.

Several weeks were sometimes necessary to make and fit up the apparatus required to test each new theory as it was formulated, and it too often happened that the first hours trial of the new scheme dashed all the high expectation that had been formed, and we had again to retrace our steps; thus week after week went on amid a constant succession of newly formed hopes and

(13.

crushing defeats, varied with occasional evidences of improvement.

I however worked on steadily, six more months of anxious toil had glided away, and things were much in the same state, except that many thousands of pounds had been uselessly expended, and I was much ~~worn~~ worn by hard work and mental anxiety.

The large fortune that was almost within my grasp seemed now far off, my name as an engineer and inventor had suffered much by the defeat of my plans; those who had most feared the change with which my invention had threatened their long vested interests felt perfectly reassured, and could now safely sneer at ~~the~~ my unavailing efforts, and what was far worse, my best friends tried first by gentle hints, and then by stronger arguments to make me desist from a pursuit

14.] that all the world had proclaimed to be utterly impossible, it was indeed a hard struggle; and I had well nigh learned to distrust myself, and was fain at times to surrender my own convictions to the mere opinions of others. — Those most near and dear to me grieved over my obstinate persistance, but what else could I do? I had the most irrefragable evidence of the absolute truth and soundness of the principle on which my invention was based, and with this knowledge I could not persuade myself to fling away the promise of wealth and fame, and loose entirely the results of years of labour and mental anxiety, and at the same time confess myself beaten and defeated. Happily for me the end was nigh, and in a few more months I had fully succeeded in producing steel worth from £50 to 60 pr Ton from Cleveland pig-iron which had cost me only £7 pr Ton

(15.

the conversion of the crude iron into steel; being effected by simply forcing minute streams of cold atmospheric air upward through it for a space of fifteen minutes!

Thus was the so-called fallacious dream of the enthusiast realized to its fullest extent, and it was now my turn to triumph over those who had so confidently predicted my failure. I could now see in my mind's eye at a glance the great iron industry of the world crumbling away under the irresistible force of the facts so recently elicited. —

In that one result the sentence had gone forth, and not all the talent accumulated during the last 150 years, of all the thousands whose ingenuity and skill had helped to build up the mighty fabric of the British iron trade — no nor the millions that had

16.) been invested in carrying out the existing system of manufacture, with all its accompanying powerful resistance, could reverse that one great fact, or stop the current that was destined to sweep away the old system of manufacturing wrought-iron and establishing homogeneous steel as the material to be in future employed in the construction of our ships, and our guns, our viaducts and our bridges, our railroads and our locomotive engines, and the thousand and one things for which iron had hitherto been used; and yet with all this newly developed power, I was paralyzed for the moment in face of the stolid incredulity of all practical iron and steel manufacturers, which stood like the solid wall of a fortress, barring my way to the victory I had already won.—

I announced the fact of my complete success

17.) to the world, and held in my hands the most undeniable proofs of the truth of my assertion, but no one would now believe it, possible, they remembered but too well the great expectations that were excited two years previously by the first announcement of my invention at Cheltenham, and were not again to be disturbed by the cry of "Wolf"!

Thus it happened that after the hard battle I had fought for so many years, I found myself as far as ever from the fruits of my labour, for not a single iron-master or steel Manufacturer in Great Britain could be induced to adopt the process.

Anxious to possess still further practical proofs of the value of my invention, I made a few hundred-weight of steel bars at my experimental works in St. Pancras, of all the sizes and special qualities required in an engineers work-shop.

18.)

These I took to the works of my friends the Messrs Galloway engineers of Manchester, and unknown to any of their workpeople, these bars were given out and employed for all the purposes for which steel has hitherto been used in their extensive business.

So identical in all essential qualities was this steel with that usually employed by their workmen, that during two months trial of it they had not the slightest idea or suspicion that they were using steel made by a new process was ever entertained by them, in fact they were accustomed to use steel of the best quality costing £60. pr Ton and they had no doubt whatever but that they were steel doing so. —

I may here remark that this tool-steel was made from Swedish Charcoal pig-iron costing delivered in Sheffield 7£ pr Ton. and it was with this high class raw-material that our firm

continued for about two years to manufacture (19.
tool-steel for engineers, for which we obtained
£64. pr Ton and with which such firms as
Sir William Fairbairn, Sir Joseph Whitworth
Messrs Sharp Stewart &c &c were regularly supplied
up to the time when larger and more profitable
work had made it not worth our while to
continue the manufacture of tool-steel.

Indeed so satisfied were Messrs Williams and
& John Galloway with the crucial test of our
tool-steel at their works that they entered into
Partnership as Steel Manufacturers with
myself, my partner Mr Robert Longsdon, and
my Brother-in-law Mr Allen. -

We built a steel works in the town of Sheffield
determined to beard the Lion in his den, and
to undersell the trade, until we forced them
in self defence to take a license under my
Patents, and carry on my process. —

20./ We soon got to work and dropped £10 pr Ton on railway-tyres &c. &c. - this soon brought the trade to a proper frame of mind. Sir John Brown & Co. applied for a licence, this was soon followed by Messrs Charles Cammell & Co. and licences were also granted to several iron-masters, of course we thus created a strong rivalry with our own firm, and forced our own prices down this we were fully prepared for, as it left still a very large margin of profit.

Some idea may be formed of its importance as a manufacture, and how much the people of Sheffield have lost by their prejudice and incredulity when I state the simple fact that on the expiration of the fourteen years term of Partnership of our Sheffield firm, the works which had been greatly increased from time to time entirely out of revenue.

(21.

were sold by private contract for exactly twenty four times the amount of the whole subscribed capital of the firm, notwithstanding that we had divided in profits during the partnership a sum equal to fifty-seven times the gross capital, so that by the mere commercial working of the process, apart from the Patent, each of the five Partners retired after fourteen years from the Sheffield works with eighty-one times the amount of his subscribed Capital or an average of nearly cent per cent every two months. a result probably unprecedented in the annals of commerce.

Yours faithfully
Henry Bessemer

Herewith, gentlemen, I have the pleasure of presenting this autograph letter of Sir Henry Bessemer to the American Institute of Mining Engineers, and would ask the Institute to place this, Sir Henry Bessemer's own account of his early struggles, among the archives of the Society.

The Siemens-Martin process of the manufacture of open-hearth steel has met with its greatest development in England, and enabled us to produce a superior material for the requirements of the ship-builder, of which he has availed himself to a vast extent. In Great Britain, the production of Bessemer steel ingots in 1889 was 2,140,-791 tons; of open-hearth steel ingots, 1,429,169 tons. In Great Britain and Ireland we are justly proud of our mercantile marine. The supremacy of our ship-building and of our shipping interests is based upon the excellent quality and abundant supply, at a moderate price, of steel furnished by our steel-makers.

The demands of naval architects and marine engineers have been responded to with intelligence and enterprise. Plates, frames, shafts, forgings, steel-castings of forms and dimensions thought impossible but a few years ago, have been manufactured at low prices. These productions have enabled our naval architects to design, and our builders to construct, the magnificent ships which are the pride of our navy and our mercantile marine. The strength of our position and the remarkable economy of this important industry can be best demonstrated to you by a fact within my own personal experience. I have, within the past two months, as chairman of a shipping company, entered into contracts for the purchase of several new first-class cargo steamers, to be built of steel, of the most approved design, with triple-expansion engines, at the cost of seven pounds Sterling per registered ton. These ships, of a carrying capacity of 4000 tons, will be built and afloat within seven months of the laying of the keel.

This example will suggest to you that our methods must be economical, and that great attention is given to the economy of manufacture. For this we are much indebted to the researches of our chemists, not only in England but also in Germany and France, an interesting illustration of which is seen in the use of basic slag as a manure. This by-product was first applied as manure in England, in the summer of 1885, by Prof. Wrightson, of Downton, Salisbury, who was employed by the North Eastern Steel Company to make a series of experiments. The results were so satisfactory that basic slag immediately became a merchantable article instead of a waste-product. In the twelve months ending June, 1890, 103,680 tons were delivered from works in Great Britain. Owing to the very much more rapid development of the basic process in Germany, a very much larger quantity of slag has been produced there, and during the past three or four years not only has the whole make

been utilized, but stocks have been used up, as well as a considerable proportion of the total quantity exported from England. The make of basic slag in Germany is estimated at about 325,000 tons for the year 1889.

The combinations of steel with other metals continue to be the subject of research and experiment. Many of our members have devoted earnest attention to this fruitful field of industry. Communications by Mr. Riley, Mr. Hadfield, M. Osmond and M. Brüstlein and others will shortly come before us in the form of professional papers or notes. It may be stated in general terms that, owing to the high cost of aluminum, it cannot at the present time come into extensive use. Mr. Hadfield and M. Osmond agree that combinations of iron with silicon and iron with aluminum present many points of resemblance. But when the addition of 0.15 per cent of aluminum, the least that has been found efficacious, means an addition of twelve to twenty shillings (three to five dollars) per ton on the cost of the steel ingot, its employment in competition with silicon, as ferro-silicon or silicon-spiegel, cannot be more than experimental. I hear, however, that an American company promises the production of aluminum at a much lower figure than that of a year ago, and that a German chemist, now living in Chicago, promises to produce it at a cost of about seven pence (fifteen cents) per pound.

The remarkable success which has rewarded the researches of experimenters on the combinations of steel with chromium, tungsten, aluminum and nickel, should be a stimulus to continued exploration in this field. The most important results appear to have been obtained by the use of nickel in combination with steel. When Mr. James Riley's communication on the subject of the alloys of nickel-steel was read before the Iron and Steel Institute in May, 1889, Mr. J. F. Hall, of Messrs. William Jessop & Sons, Sheffield, at once laid claim to priority of invention, informing the Institute of the innumerable experiments which he had been making with alloys of nickel and steel during several years. Since then uninterrupted experimental research has been diligently prosecuted. Under the superintendence of Mr. Hall, armor-plates have been manufactured from this material and tested by British Admiralty officials, giving results which rank about 75 per cent. above any similar plates ever tested in England. During the same period, Messrs. Schneider & Co., of Le Creusôt, France, have furnished nickel-steel armor-plates, which have been tested by the French, Danish

and Chilian Admiralty authorities, with results almost identical with those obtained by Mr. Hall. Within the last month a nickel-steel armor-plate of Messrs. Schneider & Co.'s manufacture has been tested at Annapolis, and has confirmed those results. At the present time, Mr. Hall has under course of manufacture breech-loading ordnance-cannons and projectiles of nickel-steel for the English War Office, and he has already applied the same material in the production of rifles and sporting-gun barrels, boiler-tubes for torpedo-boats, telephone- and telegraph-wire. When it is considered that nickel-steel can be made in various tempers, giving tensile strain of 107 tons per square inch, with an elongation of 3 per cent. in 8 inches, 97 tons tensile with an elongation of 7 per cent., and 50 tons tensile strength with an elongation of 45 per cent., it is impossible to foresee to what uses this remarkable material may not be put; and now that the various patents and interests owned by Messrs. J. F. Hall, of Sheffield, W. H. Marbeau, of Paris, William H. Schneider, of Le Creusôt, and J. Riley, of Glasgow, have been associated, their joint labors promise to give rapid development to this interesting alloy. Works have already been erected by this association in France for the special manufacture of ferro-nickel, and will shortly be followed by others in England and in the United States.

The treatment of large masses of steel by tempering, hardening and annealing in water, oil, or molten lead, has been carefully studied, but the secrets of the behavior of masses of steel in heating are yet far from being all disclosed. M. Osmond, in his excellent paper,* delivered before the International Congress in Paris last year, points out that tempering in oil was known to the ancients—being mentioned by Pliny; and Shakespeare also was aware that the Moors possessed the knowledge that different temperatures of water effected the results of the hardening process. Othello says: "I have another weapon in this chamber; it is a sword of Spain, the ice-brook's temper." One day, in conversation with Mr. Gladstone, I related this to him, and he at once capped the quotation by saying: "Yes, and I remember the words of Homer, in the "Odyssey," 'As when the smith dips into cold water a great axe, and it hisses loudly as he is tempering it, for so it is that iron is made hard.'" ("Odyssey," ix., 391–394).

I have ventured to make passing mention of some of the men and

* "On Critical Points of Iron and Steel," *Journal of the Iron and Steel Institute*, No. 1, 1890, pp. 38–80.

of some of the inventions brought forth in our island home. I have named some of the investigations and experimental researches that are being conducted by our metallurgists. You will see that we have need of the immense mineral riches of your prolific country, to furnish us with the raw materials or the metals required by our engineers, our army and navy and our ship- and bridge-builders. It is fortunate that we can make known these wants to men like the mining engineers of the United States, whose knowledge of those materials and their localities is as wide as their continent. We need the materials, but we need also food and clothing for our dense industrial population. For your grain and fruits, for your cattle and cotton, for your many remarkable mechanical devices, we have no precious metals to offer in return; but we have the products of our furnaces, the labor of our looms and the service of our mercantile marine.

It is well that we should meet thus, eye to eye and voice to voice, to discuss the interests and the scientific aspects of the great industry which absorbs us. It is thus we learn how much has been accomplished by persistent and intelligent labor, how much remains to be achieved, and how by free exchange of ideas and of productions, friendly understanding is promoted and personal acquaintance is built up. Through such orderings we are convinced that Providence has designed to wind the silken chain of commerce round the world.

ON THE PROBABLE FUTURE OF THE MANUFACTURE OF IRON.

BY SIR LOWTHIAN BELL, MIDDLESBROUGH, ENGLAND.

(Pittsburgh International Session, October, 1890.)

With the exception of air and water, it is open to question whether there is any form of matter which the human race could less easily spare than iron. Short of going the length of asserting that, without this metal for an anchor or steel for the compass, the adventurous navigator could never have crossed the wide Atlantic, we may credit the locomotive and the steamer, and hence iron, for that sequence of events which has peopled North America with the

Anglo-Saxon race. The result has been to raise a vast territory to a position without parallel in the progress of the world.

To the attractions afforded by this interesting episode from modern history may be added the unprecedented development, in the United States, of that branch of industry which most concerns the Iron and Steel Institute.

Let me here remind you, although we are guests upon the present occasion, and Great Britain is the birthplace and home of the Iron and Steel Institute, that its name and constitution were intentionally adopted so as to exclude geographical boundaries from being any limit to the scope of its operations. Accordingly, these cosmopolitan intentions have been followed by our numbering in our ranks considerably above 200 members belonging to other countries, and of these, nearly one-half are inhabitants of the United States. This union between members of one great family is a subject of sincere congratulation to the founders of the body, and, I feel sure, is not without interest to those who have joined it from the continent of Europe.

The present assembly is, so far as my reading goes, without a precedent in the annals of industry. Manufacturers of a metal from the Old World, where, in spite of what has been said to the contrary by some antiquarians, iron probably has been made for thousands of years, have accepted a courteous invitation from those of the New, with a view to discuss questions of interest common to both. To carry out this intention, three representatives from each hemisphere have been deputed to prepare subjects for this international congress, and I shall always consider my having been chosen by the Council of the Institute to act on their behalf as a very high and flattering distinction.

The heading of this paper implies that I propose addressing myself to the future rather than to the past; but as any estimate of the probability of what is going to happen must be founded to a great extent on what has already taken place, some reference to the ancient history of the iron-trade would seem not altogether superfluous.

As is well known, man in the earliest days of his existence is supposed to have supplied his wants, first with implements of stone, and then of bronze. This conclusion had been arrived at by the discovery of articles suitable for the chase, for war and other purposes, made of one or the other of these two substances, whereas iron was never met with, and hence prehistoric man was regarded as having known little or nothing of the existence of this metal.

More recent explorations, however, have proved that instruments of iron, when protected from atmospheric influence, have frequently been discovered in different parts of the world. If, then, as is quite possible, implements made of stone or of bronze were in use long before those of iron, it is perhaps equally certain that the latter metal was in more common use at a very remote period than was at one time believed.

Apart from these discoveries, let us consider the question of a knowledge by the ancients of the existence of iron as one of mere probability. The ores of this metal are by far the most abundant of all metalliferous minerals, and they are met with in almost every geological age. In the lowest type of savage life it could scarcely have escaped notice that a morsel of this abundant ore, surrounded by glowing charcoal, produced a metallic mass capable of being forged into objects greatly prized in daily life. Captain, now Colonel, Grant, the companion of Speke, found the isolated barbarians near the great African lakes familiar with this mode of obtaining iron; and he made a sketch for me, showing a couple of men engaged in this occupation. The means employed was simply a Catalan fire, of a less rude character than others which have been described by several authors. Some of these depend on the wind for their blast, whereas in my drawing each man is seen manipulating a couple of single-acting bellows.

As regards steel, we know that in comparatively remote antiquity, the low Catalan fire had its dimensions so increased that white cast-iron could not have failed to have been formed, and with this at hand, the power of producing the less carburized form of the metal could scarcely have long remained a secret.

So much for speculation; but we have more recently obtained proofs that the ancients appear to have had far more knowledge of the metallurgy of iron than that for which they have received credit. How else can we account for the forging of a column of the welded metal, nearly 25 feet high, and weighing some 6 tons? It is of great but unknown antiquity, and was found standing upright at Delhi, where it still remains. Heavy masses of iron have also been discovered for supporting the masonry in doorways and other openings in ancient buildings; and although the blast-furnace appears not to have been known when Agricola wrote his *De Re Metallica*, dated 1556, evidence is not wanting that carburized or cast-iron has been found dating from a much earlier period. These admissions, of course, do not necessarily imply any knowledge of the

property of iron to unite with carbon and form a body capable of cutting iron itself and of chiselling the hardest rock. The absence of this equally corrodible compound, from the monuments of antiquity, is, however, less difficult to account for than that of any other metallic bodies, all of which are less liable to perish from the action of the atmosphere than steel.

Coming down to a more enlightened period, the precise date of the invention of the blast-furnace, the *haut-fourneau* of the French and *Hoh-ofen* of the Germans, is still not known, but it is supposed to have been introduced in the latter part of the sixteenth century. The first change effected by the discovery was the gradual and almost complete banishment of the Catalan ore-process, but while pig-iron became the source of malleable iron, the Catalan, or a similar form of furnace, was retained for the purpose of conversion, and it remained in use for 200 years or more. Its place was gradually taken by the puddling-furnace of Cort; and with the exception of the small quantities of expensive bar-iron made in Russia and Sweden, almost all the metal produced in malleable form was obtained by means of this furnace from the time of its invention in 1784 down to the year 1860.

When I was invited to undertake my present duty, without having any definite plan in my mind, it occurred to me that the selection of some question in which coming events had been disregarded by iron manufacturers might not be an unprofitable use to make of the opportunity placed at my disposal. The particular event which first suggested this choice took place as recently as twenty-five years ago. The process of puddling, of course, is a very familiar one to every one in this room. It consists, as we all know, in melting pig-iron in a furnace of the reverberatory type, after which the workman, by very severe and exhausting labor, stirs the fluid metal and thereby constantly exposes fresh surfaces to the oxidizing influences present. This treatment burns off the greater portion of the associated metalloids, viz., carbon, silicon, sulphur, and phosphorus, which the metal has absorbed during its passage through the blast-furnace. So severe is this labor that two men have their strength a good deal taxed in producing, for a day's work, 22 cwts. of puddled bars. To obtain this, an equal weight of coal has to be thrown into, and consumed on the grate of their furnace.

Something like thirty years ago, Henry Bessemer proposed a plan in which the strong and never-tiring arm of the steam-engine should do the work of the wearied puddlers, and the heat required was sup-

plied, not by coal, but by the combustion of the metalloids just referred to. Instead of 22 cwts. being the weight, and 12 hours the time, 10 or 12 tons of cast-iron are rendered malleable in very little more than the same number of minutes, the duty of the men during the conversion consisting chiefly in watching the progress of the operation.

In viewing the impressive but simple process of "blowing" a charge of metal, it is difficult to realize the disappointments and the large expenditure of money and indefatigable energy before the position referred to was reached. As a matter of history, however, in the year 1870, such substantial progress had been made, that 240,000 tons of Bessemer ingots were delivered for consumption. But what had been the action of the puddling-furnace owners during the onward march of this formidable and threatening competitor? It was this—in the year 1860 the number of puddling-furnaces at work in Great Britain was given at 3462. In 1870 they had increased to 6700, which grew to 7575 at the end of the next five years. In 1887, on the other hand, the number had fallen to 2875, in which year it was estimated that in the North of England alone there were 1000 puddling-furnaces unemployed, representing, with the finishing mills 1½ million sterling (7⅓ million dollars). In Wales 600, and in the remainder of the United Kingdom another 1000, had shared a similar fate. Thus there was property which must originally have cost nearly 4 millions sterling (19½ million dollars) lying idle. In addition to this calamitous change, the 2875 furnaces at work were not actively engaged above three-fourths of the time. In the meanwhile the manufacture of Bessemer steel went on increasing, so that in 1877 there was produced in the United Kingdom fully one million tons, and above an equal weight in other countries. Since 1887 the number of puddling-furnaces in operation has risen to 3346 in 1889, but the annual production of Bessemer and open-hearth steel has increased from 1,000,000 to 3,569,960 tons. It cannot, however, be questioned that the iron manufacturers were placed in a position of extreme uncertainty and difficulty. The superiority of steel, particularly as a material for rails, had been amply proved, but it also had been thoroughly ascertained that the hematites of Cumberland and Lancashire were the only ores in Great Britain suitable for Bessemer purposes. The growing demand for steel, therefore, soon exceeded the powers of supply from these two counties, and the consequence was a rise in the price of their pig-iron far exceeding that of those districts whose produce was useless for

steel-making. The market values of Cleveland and West Coast of England metal are here given to show the nature of the differences which arose after the change of conditions referred to:

Year.	Pig-Iron. Thous'nd Tons.	Cleveland Price Per Ton.			West Coast Price Per Ton.			Difference in Price Per Ton.			Imported Ore. 1000 Tons.
		s.	d.	Dollars.	s.	d.	Dollars.	s.	d.	Dollars.	
1870	5,963	50	3	12.20	72	3	17.55	22	0	5.35	208
1871	6,627	49	8	12.05	89	6	21.74	39	10	9.69	211
1872	6,741	97	1	23.60	170	0	41.32	72	11	17.72	576
1873	6,566	109	2	26.50	156	9	38.06	47	7	11.56	738
1874	5,991	70	11	17.25	108	6	26.38	37	7	9.13	1,009
1875	6,365	54	6	13.25	78	3	19.02	23	9	5.77	738
1876	6,555	37	4	9.10	59	4	14.43	22	0	5.33	972
1880	7,749	45	0	10.95	81	6	19.80	36	6	8.85	3,060
1882	8,586	43	5	10.55	56	7	13.74	13	2	3.19	3,692
1884	7,811	37	0	9.00	44	11	10.81	7	11	1.81	3,135

The embarrassment experienced at the time was further intensified by an almost unprecedented activity in the general iron-trade of the country, which, finding no relief from Cumberland or Lancashire, had to be met by the metal obtained from the more plentiful ores of Cleveland and Lincolnshire, etc. As usually happens upon such occasions, the excited state of the market in Great Britain affected prices in the world at large, so that a state of things similar to that described prevailed in every part of the civilized globe where iron was largely made.

Gradually the existence of a deposit of hematite ore in the North of Spain seems, so far as its abundance and ease of extraction were concerned, to have dawned then, for the first time, on the minds of the British ironmasters. Improvements in the navigation of the river Nervion, and in steam-navigation, have enabled the produce of the mines at Bilbao to compete in point of price and quality of metal, not only with that on the west coast of England, but with the cheaply wrought and most favorably situated bed of ironstone, spread over a great tract of country in the hills of Cleveland. Gradually the large rail-trade of Middlesbrough was extinguished, and to a large extent, its manufacture of iron boiler- and ship-plates has suffered in like manner. As a result of this revolution, something like 48 out of 103 furnaces in the north-eastern district of England, all erected for smelting native ironstone, are now engaged in providing pig-iron from imported ore for the steelmakers. Nor

has the disturbance been confined to this locality, or to other cheap iron-making centres, but the hematite mines of Cumberland and Lancashire have also been seriously affected by having to compete with these foreign ores. This may be judged by the fact that, while twenty years ago the importations were only 130,000 tons, last year they amounted to 4,031,265 tons.

While considering whether a useful lesson might not be learned from this retrospect of our trade, particularly by members of our Institute of less advanced years, reports reached me from opposite quarters of the globe of the so-called "direct process," beginning again to find favor in the minds of gentlemen well qualified to express an opinion on a scheme which, if successful, would be a rival to the united power of the blast-furnace and the Bessemer converter.

The great advantages possessed by the present mode of treating the ores of iron are too well understood to render it necessary that the blast-furnace should occupy our attention at any length upon the present occasion. These, therefore, will only be spoken of in order to contrast them with those of a system by which it is intended to obtain wrought-iron and steel by direct treatment. In doing this, it will be made apparent that the blast-furnace is able to perform a duty which has never been approached by any other form of apparatus, viz., the complete reduction of the metal contained in the ore.

Many eminent metallurgists have regarded with favor the principle contained in a variety of the processes known as "direct," and it must be admitted that there is, in the idea, a simplicity as compared with the present mode of manipulation which is highly attractive. In the latter, the iron for the blast-furnace has been made to unite with certain metalloids, although we know at the time that in order to render the product malleable, these substances will have to be removed. Among those who have labored in this field, not the least distinguished was my late friend, a former President of the Iron and Steel Institute, Sir William Siemens. In the hope of solving a problem in which his predecessors had not been successful, he spared neither time nor money. Ritter von Tunner, a high authority in all things connected with iron, has, in our frequent correspondence on such matters extending over many years, expressed himself at one time hopefully on the direct process. In a recent communication, however, he seems to have abandoned all expectation of its being able to compete with the combined forces of the blast-furnace and the Bessemer process. On the other hand, so far as I know, Siemens died believing in its future success.

I have elsewhere endeavored to prove that reduction is not

exclusively effected in the so-called reducing zone of the blast-furnace; on the contrary, that oxygen, amounting to about 25 per cent. of its original quantity, arrives at or near the tuyeres, combined with iron in the form of some suboxide. Further, that final deoxidation of this suboxide is probably effected by the quantity of carbon, generated by a dissociation of carbonic oxide, as expressed by the formula, $2CO = C + CO_2$. This reaction is inferred from the results of many experiments in the laboratory and from repeated observation at the furnace itself, confirmed by the fact that the gases at the tuyeres contain a quantity of oxygen, about 2 cwts. per ton of metal produced, and something like $1\frac{1}{2}$ cwt. of carbon; the former not being accounted for by the atmospheric air used, nor the latter by the weight of fuel burnt in the hearth. Whether or not it is the actual carbon precipitated from the carbonic oxide, or coke itself, which completes reduction is immaterial from a heat-producing point of view; what seems proved is, that it is reserved for the crucible to complete this work, at or about the period when fusion takes place.

As a theoretical proposition, we have been reminded that the exact quantity of carbon required to deoxidize 20 cwts. of iron in the form of ferric oxide is 4.38 cwts. It is further assumed that this mixture of ore and carbon must be raised to a temperature of 800° C., for which 1.27 cwt. of carbon ought to suffice, provided the whole amount of carbon that is used is burnt, half to monoxide and half to dioxide. It is suggested, not that the work can be done for these 5.65 cwts., but that they are the limits towards which we may work.

We will imagine, as indeed has been proposed, that the operation is carried out in the furnace used by Siemens, which was one of the reverberatory type, constructed on the rotating principle. As bearing immediately on this question, it may be mentioned that after a vast number of experiments I arrived at the conclusion that the following temperatures may be regarded as those which are required for setting up the actions given in the subjoined table:

	C.	F.
Carbon monoxide commences to reduce calcined Cleveland ore* at	210°	410°
Carbon commences to reduce calcined Cleveland ore at	399°	750°
Carbon dioxide commences to oxidize spongy iron at	426°	800°
" " is expelled from limestone at . .	538°	1000°
" " commences to dissolve carbon at .	815°	1500°

* Cleveland ore is specified, because there are slight variations in the temperatures at which different ores are affected by carbon monoxide.

It must be understood, however, that at the temperatures named the action goes on but slowly, but that it speedily increases as the heat is raised.

Returning for a moment to the blast-furnace, we have in the crucible the zone of greatest heat, fusion of the slag and iron, and removal of the last portions of oxygen from the iron.

During the next 40 feet in height the chemical changes are insignificant, that portion of the interior being almost entirely utilized by the descending materials in absorbing the heat contained in the gases, and returning it to the hearth.

At a distance of 60 feet above the hearth carbon dioxide is given off from the limestone, where the temperature, as a rule, suffices for this gas to dissolve carbon, and thus prevent its combustion at the tuyeres.

Lastly, reduction in the upper zone is effected, where the temperature in a modern furnace ought never to be such as, with good coke, would permit the resulting gas to act on carbon ($CO_2 + C = 2CO$).

So far as my observations, experimental and otherwise, have enabled me to judge, the maximum quantity of oxygen representing the second equivalent in carbon dioxide in the gases, is that corresponding with the equivalent contained in the ferric oxide of the ore. From this it might be inferred that reduction is completed in what has been generally known as the reducing zone, *i.e.*, the uppermost and cooler portions of the furnace. We know, however, that the deposition of carbon, accompanied by the generation of some carbon dioxide, as already mentioned, is also most abundant at moderate temperatures. These facts might reasonably lead us to expect a quantity of carbon dioxide in excess of that due to the reduction of ferric oxide by carbon monoxide.

It will, however, be shown immediately that a condition of equilibrium in the gases is determined, not alone by the relative quantities of the two carbon oxides, but also by the amount of oxygen still remaining in combination with iron. Now, the quantity of carbon required to be burnt at the tuyeres in order to fuse the slag and iron, added to that rendered necessary to reduce the metalloids found in the pig, is such that at the various temperature zones through which the ore descends, a position of equilibrium is reached, when the gases contain at the outside about 6 cwts. of carbon per ton of iron, in the form of dioxide, and when the metal still retains about one-fourth of its original quantity of oxygen.

This observation refers to ores of the type of Cleveland, but in

those of a different character the composition of the gases may be materially modified. Thus, in certain Swedish furnaces, the fuel consumed does not greatly exceed two-thirds of the quantity necessary for smelting Cleveland ironstone. The volume of carbon monoxide for every ton of iron is correspondingly diminished, the result being that, instead of having two volumes of this substance in the escaping gases for each volume of carbon dioxide, the two gases are found in almost equal volumes. In one respect, however, all furnaces are alike, viz., in there being an increase of carbon and oxygen in the gases at the tuyeres, as compared with the proportions found a short distance above that level.

Thus it will be seen that at no time during the descent of the ore in a properly constituted blast-furnace is it, with one exception, ever exposed to a high temperature, in an atmosphere of gases capable of oxidizing iron. The exception is when carbon monoxide is split up into carbon and carbon dioxide; and then the precipitated carbon serves, as has already been observed, to remove the oxygen taken up by the iron during the act of dissociation, when the ore arrives at the tuyeres.

Now it is not too much to say that the conditions of every reverberatory furnace are the very reverse of those just laid down. If coal is the fuel burnt, which generally is the case, we have vapor of water, carbon dioxide, and very often a good deal of free atmospheric air, heated probably at least to 1500° C., passing over the materials.* Thus exposed, spongy iron, when formed by direct reduction as proposed, could not fail to be very rapidly oxidized. But the circumstances under which it is now expected to produce spongy iron, consist in burning half the carbon to dioxide, and the other half to monoxide. From the information contained in the table just given, we learn that while carbon monoxide is a reducing agent, the product of reduction—carbon dioxide—is of an opposite character, and, in consequence, is able to reoxidize the metallic iron by the production of which it was generated. The cause which determines this reflex action is difference of temperature, and, as might be expected, when both gases are present there is brought about a position of equilibrium determined, as has been said, not by temperature alone,

* Of course it would be quite possible to maintain an excess of the reducing gas—carbon monoxide—in the furnace, but this would mean a great waste of fuel.

but also by the proportion of oxygen still remaining united with the iron.

Thus, if a mixture of 2 volumes of carbon monoxide and 1 volume of carbon dioxide is passed over calcined Cleveland ore at about 417° C., one third only of the oxygen can be removed, *i.e.*, the ferric is reduced to ferrous oxide.

If, on the other hand, the mixture consists of equal volumes of the oxidizing and reducing gases, the same effect is produced, provided the temperature be raised to a bright red; and spongy iron similarly exposed is oxidized, and becomes also ferrous oxide.

At a white heat the attraction of the metal for oxygen is so intensified that further action ceases when 12 per cent. of the combined oxygen is removed from the ore. At the same time 10 volumes of carbon dioxide are able to keep in check the reducing power of 90 volumes of carbon monoxide.

I see no reason, however, why a close approximation to complete reduction, in the direct process, should not be effected by using a sufficient quantity of carbon mixed with the ore, in the manner already described. The difficulty to be apprehended is the re-oxidation of the spongy iron when, by the revolution of the furnace, a fresh surface is exposed to the flame which is serving to heat the materials under treatment. Certain it is that Siemens failed to reduce the loss of metal to within reasonable limits. Forty-five years ago I labored for some time with Mr. William Neale Clay who was very sanguine of being able to dispense, wholly or partially, with the blast-furnace. The results, in our case, were the same as those obtained by others who have followed us in the same path since that time. As regards the carbon required as a source of heat for raising the temperature of the mixture of ore and carbon, I see no prospect of reducing this with any approach to the quantity previously named, *i.e.*, 1.27 cwts. per ton of iron. Not only theoretically, but practically, in the Bessemer converter, the combustion of the metalloids and a portion of iron suffices to raise the bath of melted cast-iron to a proper temperature for keeping malleable iron in a state of fusion. Attempt the same thing in a puddling-furnace, rotating or otherwise, and the oxidation of the same metalloids and iron, producing the same quantity of heat as in the converter, requires to be supplemented by the heat afforded by 15 cwts. of coal, in each case commencing with iron in a fluid state. In both, there is a great loss of heat at the chimney, but this we will neglect, and regard the

remainder in each as due to radiation, convection, etc, inseparable from the larger size of the furnace and the small amount of work turned out as compared with Bessemer's apparatus.

If, however, we take the weight of a cubic foot of pig-iron and contrast it with the same volume of the mixture to be employed in the direct process we are describing, the iron turned out on such a mode of computation will be about one-tenth of that obtained when pig-iron is the raw material. To what extent the 15 cwts. of heating coal per ton of iron will be increased by any such diminution in the yield of product it would be difficult to predict, but it is perhaps not necessary to pursue this branch of the inquiry at greater length in this place.

Up to this point, the economy of the direct process has been argued upon thermal and chemical grounds, but the question of the capabilities of a reverberatory-furnace, as compared with the united action of a blast-furnace and a Bessemer converter, is perhaps best answered by the amount of labor required by the two systems. In an audience, consisting largely of practical men, I do not apprehend a denial that our present "indirect process" costs in wages, for the smelting and Bessemer processes, less than one-half of that paid in converting pig- into puddled-iron; and probably we shall agree in supposing that if the reverberatory-furnace should have to do the work of the blast-furnace, this double cost may have to be doubled again.

In a locality where charcoal appears to cost 60s. per ton, and lignite 12s., but where ore, very suitable for steel-making, is valued at 5s. 6d., it is sought to neutralize the disadvantages of coal, dear and unsuitable for blast-furnace work, by having recourse to a direct process. To carry out these views a highly carburized bath (*hochgekohltes Metallbad*) of pig-iron is made in a Siemens furnace. To this molten pig, melted or highly-heated ore is added, by which the excess of carbon in the fluid metal, reduces a corresponding portion of iron of the ore just added. The bath of metal is now to be restored to its pristine state of carburization by the addition of solid carbon, after which the slag formed is removed and more ore is to be added, and this alternate system of treatment is continued until the required quantity of steel is obtained.

The cost of 1000 kilos., say one ton of the steel, produced by this modification of the direct process, is thus stated by its author:

Ore containing 52 and 54 per cent. of iron, but yielding 47.4 per cent. of steel,			£	s.	d.	Dollars.
	2110 kilos. @ 5s. 6d.	=	0	11	7.2	2.82
Charcoal,	240 " @ 60s.	=	0	14	4.8	3.50
Lignite coal,	2000 " @ 12s.	=	1	4	0	5.83
Labor,		=	0	11	0	2.67
Repairs,		=	0	8	0	1.94
Carburizing the metal,		=	0	1	0	0.24
			3	10	0	17.00

After what has been said, I shall be surprised if an ore containing say 53 per cent. of iron, when reduced in such an oxidizing atmosphere as that to which it has to be exposed in a Siemens furnace, will be found to yield 50 of steel.* The other expenses do not require criticism, because, even upon the statement given above, the weight of ore and coal used in this direct and single operation is greater than that consumed in the blast-furnace and at the converter taken together; the labor is very much higher, and I apprehend that to the remaining item of "repairs" (8s.), a considerable sum will have to be added for other expenses connected with the establishment. Further, it may be said that the abnormal conditions of the locality are not such as to render the process one of general application, even if all the expectations entertained of it by the inventor were realized.

M. Chenot at the French Exhibition 35 years ago was awarded a "*grande medaille d'or*" for what M. LePlay, a scientific Frenchman of repute, declared to be "the greatest metallurgical discovery of the age.".† In 1862 it was reported that 2000 tons per annum of bar-iron were being made by it in the north of Spain, and I myself saw the process in operation there in 1870. It consisted in applying external heat to a mixture of ore and charcoal in a close retort. The process was practiced for some years subsequent to my visit, and it may be therefore inferred that time enough had been devoted to the question to have proved its commercial value or the reverse. It is now a considerable time since all trace of this discovery has been swept away after having had a very long trial. According to an account given by Dr. Percy there was a loss of 18 per cent. more ore in obtain-

* It should be mentioned that in this particular case it is proposed to have the initial highly carburized iron made in the Siemens furnace itself or obtained elsewhere, most probably, I presume, from a blast-furnace. As no pig-iron is alluded to in the estimate of cost, it is taken for granted that the entire operation is conducted in the open-hearth.

† Percy's *Metallurgy, Iron and Steel*, p. 335.

ing merchant-bars than is incurred in the ordinary method of producing this form of malleable iron. The fuel consumed is about the same in both cases; but as more than one-third of that used in the direct process was charcoal, its actual cost must necessarily have been much higher in the latter case.

M. Gruner, whose valuable labors in the metallurgy of iron have been universally acknowledged, gives a much more unfavorable account of the Chenot process.* According to him the waste of metal (Fe) upon a ton of finished iron was 45 per cent., and the consumption of fuel amounted to four tons. This calculation does not appear to include the power for driving the machinery, so that $4\frac{1}{2}$ tons would be the probable total weight of coal required per ton of finished bar-iron.

The consumption of coal for the blast-furnace and Bessemer process on a ton of merchant-steel does not exceed $2\frac{3}{4}$ tons, while the actual waste of iron (Fe), instead of being 45 per cent., as in the direct process, may be taken at 15 per cent.

In addition to this serious waste of material, there are other inconveniences connected with the manipulation of the product. The iron sponge is so prone to oxidation that, when warm, it takes fire on exposure to the air—a property which compels its storage in close vessels. M. Gruner mentions eight days as the period thus required for reduction and cooling. If this be correct, sufficient appliances would have to be provided to hold about 2700 tons of the light sponge, in the event of it being desired to rival the production of one of the Edgar Thomson blast-furnaces. Besides this there will be the preparation of from 3500 to 4000 tons of ore per week, which has to be used in the size of hazel nuts.

A few years ago, there existed among the mountains of Savoy, a small blast-furnace, making, upon the occasion of my visit, about 25 tons of iron in the week. The fuel was charcoal, and the blast was created by a small stream of water falling down a 12-inch square wooden box, about 15 or 20 feet in height. The water was received in a square closed cistern of wood, by which the air carried down with the stream was separated from the water and driven into the furnace. This was the instrument, but with only half the power of the one I saw working, which, ages ago, finally extinguished all the Catalan direct furnaces in almost every part of the civilized world.

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* *Traité de Métallurgie*, tome ii., p. 244.

In the year 1727 charcoal pig-iron cost at Kidderminster £6, 5s. ($30.36) per ton, of which the ironstone cost only 30s. ($7.29), the weekly make being under ten tons. To convert this into malleable iron in a Lancashire fire, as it is called, a further expenditure of £5 ($24.30) was incurred, bringing up the total cost to £11, 5s. ($54.66).

One hundred and sixty years after that date finds us in England bringing ore 1000 miles over sea, selling the pig-iron made from it at little more than 40s. ($9.72), and disposing of steel rails made from the same pig at less than 75s. ($18.23).

It is against the appliances which have enabled us to perform what may be regarded as a marvel of economy—against the blast-furnace making 1000 and even up to 2000 tons in a week, and one or two Bessemer converters, as the case may be, turning the whole product into steel as the metal runs from the furnace, that a process, which has practically failed to make one step in advance during the last fifty years, ventures, in our day, to wage war.

If there is a spark of vitality in this obsolete plan, what have our ironmasters been about that they have not blown it into life? We have had, and have now, men among them who have distinguished themselves by the manner in which they have taken hold of every sound idea, scientifically propounded or otherwise, and ensured its success, often beyond the most sanguine expectations of the most sanguine inventor. It may well be asked how is it that this direct process has, up to this time, ended in direct failure? Perhaps someone else will endeavor to answer the question, for, as regards myself, my fears as to its merits were published sixteen years ago. All that I can add to what was then said, is that its future chances of success must depend on counteracting the obstacles referred to in this paper, obstacles which hitherto have constituted a barrier between every direct process and commercial success.

It has also been proposed to accelerate the direct process, and for that matter a great many other things, by the use of water-gas, a form of combustible of which a good deal has been said of late. It is not to be denied that hydrogen gas, forming, as it does, half the volume of water-gas, the other half being carbon already partially burnt, may as a combustible have an assured value in certain special cases. The hydrogen element affords above four times as much heat as the same weight of carbon, but unfortunately and unavoidably it requires more than four times as much carbon to produce it. We might therefore as reasonably expect a hydraulic wheel to pump back all the water which had served to move it, as to hope to be able

economically to burn any combustible twice over; for that is the impossible problem the water-gas makers ask us to accept as possible. It is almost superfluous to say that the same line of argument is more than equally applicable to a proposal to use water-gas in the blast-furnace, which was also mooted some ten years ago. Its utter unfitness for this office is apparent when we consider that any steam formed by the combustion of hydrogen, and any carbonic acid generated by burning carbon monoxide at the tuyeres, will be returned to their original condition of water-gas the next instant after their formation. In other words, heat generated by their being burnt will be entirely absorbed by the return of the constituents to the form in which they existed in the water-gas previous to its combustion.*

In face of the admitted advantages in quality and cost of conversion of ingot- over puddled-iron, the fact that $2\frac{1}{4}$ million tons of ordinary malleable iron were made in 1889 in Great Britain may seem, and no doubt is, difficult of complete explanation.

There are several circumstances which conduce to this continued vitality, but into these there is not time to enter here. Possibly the puddling-furnace may be employed for many years to come, but everything seems to point to a gradually decreasing demand for its product. Should the day ever arrive when the labors of Bessemer, or Martin's adaptation of the Siemens furnace for decarburizing pig-iron, entirely supersede the process of puddling, it must not be forgotten that the world owes to Cort an invention, which of itself had revolutionized the art of locomotion by sea as well as by land.

Mention has been made on a previous page of the hurtful effects of phosphorus in pig-iron for steel-making purposes. Unfortunately for Great Britain, its more extensive beds of ore were deposited at a geological period when the prevalence of animal life served to concentrate in particular localities the element so much objected to by iron manufacturers. This source of contamination, as it was deemed, appears to have been in active operation during the carboniferous age, and still more so while the rocks of the lias and oölite were in course of formation. Thus it is that the ores, known under the general appellation of clay-ironstone, yield an iron containing from .5 to 2.5 per cent. of phosphorus, which has been derived from phosphates, chiefly of lime, a common constituent of animal bodies.

* It is a remarkable fact that the document recommending this application of water-gas was accompanied by a statement of an actual trial made at a Belgian iron-works, every line of which proved, in unmistakable terms, its complete failure.

More than twenty years ago it was pointed out how beneficial it would be if this phosphorus could be eliminated from the iron and laid on our land. We have lived to see this accomplished by the basic process, and having regard to the growing demand for steel, it is difficult to over-rate the importance of extending the area we have to look to for the raw material. Mr. Gilchrist informs me that the quantity of pig-iron treated in the basic-lined Bessemer converter and open-hearth furnace during the year 1889 was close on to 500,000 tons in the United Kingdom, and 1,174,000 tons in other parts of the world.

The effects of the presence of foreign substances on the properties of malleable iron and steel are now universally recognized. This observation is more particularly directed to the injurious consequences which often result from the silicon, sulphur, and phosphorus which remain after the conversion of the pig-iron into these two forms of the metal. On the other hand, the service rendered by carbon in conferring increased strength and elasticity on iron, cannot be exaggerated. This fact naturally led to the inquiry whether other elementary bodies may not assist, either in carrying off hurtful matter, or increasing the valuable properties just referred to.

Other problems connected with the resisting powers of steel, as they are affected by the manner of cooling, are also occupying the minds of those interested in the question. Among others, Mr. F. Osmond has contributed important information in a recent communication to the Iron and Steel Institute.*

Discussion.

George W. Maynard, New York City: As Sir Lowthian Bell is seeking for any sign of vitality of the "obsolete direct process," he will be interested to learn that such a process is now in operation in Pittsburgh, and that for this particular locality it is an unquestioned financial success. I refer to the Carbon Iron Company's process which has been described in our *Transactions* by Mr. A. E. Hunt.†

As at present practiced, it consists in charging an intimate mixture of wet, finely-ground iron-ore and coke upon the cinder-hearth of an ordinary reverberatory- or puddling-furnace, and heating the charge with natural gas in an atmosphere that is moderately oxidizing.

* *Journal of the Iron and Steel Institute*, No. 1, 1890, p. 38.

† *Transactions*, vol. xvi., 708, and vol. xvii., 678.

The ore is reduced by its intimate contact with the ground coke, and the iron is balled at nearly a white heat. Two points require special mention: first, the benefit of mixing the ore and coke very intimately, as by such mixture it is found that the fine particles of ore protect the carbon particles from too rapid combustion, and time is secured for the thorough reduction of the iron oxide. Second, the importance of using none but the richest ores, or cleanest concentrates, as it is only by such practice that the loss of metal can be kept down to a commercially practicable limit. Sixty-five per cent. Lake Superior ores, carrying three, or at most four, per cent. of silica furnish the present supply. One long ton of this ore is mixed with 600 pounds of ground coke, and yields at the present time 1325 to 1375 pounds of squeezed sponge which runs nearly 93 per cent. of total iron. The quantity of natural gas consumed per ton of product has not yet been ascertained, but it is roughly estimated at the amount required to puddle a ton of pig-iron in a non-regenerative furnace of the same type as the reducing furnace—that is, about 35,000 cubic feet of natural gas, or 27,000 pounds of Pittsburgh coal.

H. M. Howe, Boston, Mass.: I think that very few people of good judgment can refrain from agreeing with Sir Lowthian Bell's main conclusion that the blast-furnace is more likely to be able to do without the direct process than the direct process is to dispense with the blast-furnace; but it might be rash to conclude that because the blast-furnace will probably hold the upper hand, the direct process may not be applicable, and indeed very economical, under certain conditions.

The direct process is more difficult and complicated than the blast-furnace process. It requires greater skill and special conditions. It seems to me a little like the case of an absolute monarchy versus a limited monarchy. The latter is a far more difficult and complicated system of government than the former, but nevertheless we all think that it is much the better system. The compound engine is more complicated than the plain slide-valve engine, and yet under certain conditions it is better. The direct process seems to have great possibilities. In a blast-furnace, we have not only to do the work of the direct process, but a great deal of other work besides. The work of producing iron involves, first, the chemical work of deoxidizing the iron oxide; and second, the heating work, *i.e.*, raising the iron and the other materials associated with it by nature to a very high temperature. In the case of the direct process (and I do not speak of any one direct process in particular, but simply of the direct pro-

cess in general) it is not necessary in the nature of things that the temperature should ever exceed a red heat. In the case of the blast-furnace, however, it is hard to see how the temperature can be less than the melting-point of a very infusible silicate, chiefly of lime.

The direct process calls for certain conditions. I do not see how it can be applied to any but very rich ores or rich concentrates. Its success will also be favored by a supply of a fuel which can be applied to the direct process, but which cannot be applied to the blast-furnace, and which is much cheaper than any locally available fuel applicable to the blast-furnace. That at once gives the direct process an enormous local advantage; and many wonder whether the success of the Carbon Iron Company's process in this city is not in large measure due to the fact that natural gas, here a very cheap fuel, can be used for this process but not for the blast-furnace.

As to the loss of iron: is it quite just to say that because the loss of iron has been enormous in some of the direct processes carried out in the past, without the metallurgical knowledge or skill of to-day it must always remain so? I have had access to most of the data from which Prof. Maynard speaks, and from them I find that the loss in the Carbon Iron Company's process is about 17 per cent.; that is to say, 100 parts of iron in ore will yield 83 parts of ingots by the direct process and the open-hearth process combined.

G. J. Snelus, Workington, England: I think we all admit that the blast-furnace is not going to be ousted entirely by the direct process, and to this extent I certainly agree with Sir Lowthian Bell. Where we require pig-iron for foundry-work or for any purpose for which it is used as pig-iron, the blast-furnace must always hold its own. For many other purposes the combined processes of the blast-furnace and the open-hearth, or the Bessemer converter, will also undoubtedly hold their own; but there are many cases where a direct process of making iron, if it can be carried out scientifically, will be available, useful and economical. My first experiments on this subject were made some twenty-three years ago, at Dowlais, in the presence of Mr. Thomas Blair, of Pittsburgh. I reduced some iron-ore to a fine state of division, and then noted the time required, under the most favorable circumstances, to get rid of the oxygen and obtain metallic iron. I found it possible to effect the deoxidation of the cold ore in two minutes, which showed that the chemical hold which oxygen has upon iron is not a strong one, and that, under proper conditions, the metallic iron can be set free in a very short time. This is an important fact for consideration.

I carried out the reduction in a modified form of Gerstenhofer's shelf-roasting furnace, introducing carbonic oxide or hydrogen instead of the air used in ordinary roasting.

Dr. Percy endorsed the theoretical soundness of this process, but I abandoned its development on discovering the existence of an earlier, substantially identical patent of Mr. Anderson, which soon thereafter was allowed to lapse. I mention the matter now because of the growing importance of finely-comminuted ores. In England we have vast quantities of "Blue Billy"—the residue from burnt iron pyrites—and in America it appears that large quantities of the abundant magnetic ores will have to be brought to a fine state of division for the purpose of improving their quality by magnetic concentration. It is quite evident, therefore, that an inexhaustible supply of this fine ore is available, which can be used to best advantage by a process of direct reduction, in which it is practicable to bring the reducing agent into extremely intimate contact with the very finely divided ore. The ore should be as finely crushed as possible, and the reducing agent should likewise be in a fine state of division, or it should be a gas. Sir Lowthian Bell's statement that the theoretical quantity of carbon required for the chemical reduction is only 4.38 cwts. for 20 cwts. of iron, and that the theoretical quantity of carbon required to produce the necessary heat is only 1.27 cwt.—in other words, that a theoretical consumption of 5.65 cwts. of carbon is sufficient to produce one ton of iron—shows beyond question that there is an aim in the use of coal which is worth striving for, and which, I venture to say, some one in future years will very nearly succeed in reaching.

Sir Lowthian Bell, Middlesbrough, England: As Mr. Blair's name has been mentioned in the course of this discussion, I may state that, as long ago as 1874 I endeavored to prove to him by his own figures that nothing but failure could attend the operation of his direct process. Neglecting for a moment the question of economy, and considering only that of chemical reaction, no process could be more perfect in its character than that early one adopted by Mr. Blair. It was the Chenot process with a little refinement and so arranged that a saving of fuel, at all events in heating the ore, was attained. Almost all atmospheric oxidation of the sponge was avoided by dropping it into air-tight cases and keeping it there until it was cooled. The sponge was then mixed with coal-tar, and pressed into bricks, which were charged into the pig-bath of an open-hearth

furnace. It was there, as I pointed out to Mr. Blair at the time, that the great loss occurred.

In drawing our attention to a new development of the direct process at the works of the Carbon Iron Company, Mr. Maynard has said nothing about this loss incurred in treating sponge-blooms in the open-hearth. As for the total fuel requirement for the Carbon Iron Company's process, it appears that 960 pounds of ground coke are used per (long) ton of iron, merely to effect the reduction. That alone is more than one-half the coke consumption at the Edgar Thomson furnaces, where, as I have been recently informed, 1808 pounds of coke make one ton of pig-iron. But in addition to the 960 pounds of coke used as reducing agent, this direct process requires the equivalent of 1 to $1\frac{1}{3}$ ton of coal per (long) ton of blooms for heating the ore and balling the sponge. The total fuel consumption is therefore vastly in excess of the requirements for the combined blast-furnace and Bessemer processes.

Independently of the fuel question, however, I am surprised that no hesitation is expressed at the prospect of having to grind to a fine dust the immense quantity of ore that would be required to raise this industry to anything like a scale of importance. Conceive that you are to grind the whole ore-product of the Lake Superior region, which already amounts to more than seven million tons per annum, and you have an idea of what is proposed. I admit that there are cases in which very fine ores may possibly, to a very limited extent, be treated by a direct process; but to expect to raise that process to a great industry, comparable with the blast-furnace process is, in my opinion, one of the most Utopian ideas that ever entered the mind of man.

H. M. Howe, Boston, Mass. (Communication to the Secretary): As I have elsewhere pointed out, certain features of the direct process may, under conditions often met with, give it a great advantage over the blast-furnace process—so great indeed as to outweigh the uncontested disadvantages under which the direct process suffers.

The direct process may yield iron relatively free from carbon; it may permit dephosphorization, though at the cost of serious loss of iron; it can use fuels which are often cheaper than the anthracite, coke, and charcoal to which the blast-furnace must ever be restricted; in case of very rich ores it may use much less fuel than the blast-furnace.

Against these advantages must be weighed its disadvantages in not offering the ready means of the blast-furnace for removing the

sulphur of the ore and fuel; and probably inevitably greater outlay for labor and repairs, and for loss of iron.

I would also point out that what I have ventured to name "semi-sponge," a natural product of the direct process, should under many conditions not only be made much more cheaply than pig-iron, but should be worth more intrinsically to the open-hearth steel-maker. By semi-sponge I mean ore which has been partly but not completely reduced. In the open-hearth process we use scrap-iron and iron-ore, in different proportions, according to local traditions and the local ratio between the cost of scrap, of pig-iron, and of ore. I believe that under many conditions we can use this semi-sponge in the open-hearth process, in place of an equivalent mixture of ore and of scrap-iron, working more rapidly and with a smaller outlay for ferriferous materials.

THE DEVELOPMENT OF THE MARINE ENGINE, AND THE PROGRESS MADE IN MARINE ENGINEERING DURING THE PAST FIFTEEN YEARS.

BY A. E. SEATON, HULL, ENGLAND.

(Pittsburgh International Session, October, 1890.)

In this paper it will be my endeavor to trace the development of the marine engine and its appurtenances, and the general progress that has taken place in marine engineering generally during the past fifteen years, rather with the object of attributing much of it to the successful efforts of the makers of iron and steel, thereby rendering my thanks to them for the valuable service they have contributed to my profession, than to recount what has been achieved by the skill, ability and perseverance of my colleagues in shipbuilding and engineering.

The engineer has been at all times dependent on the ironmaster, and on more than one occasion his progress has been barred by the inability of the latter to overcome some obstacle to success in his manufacture. But on the other hand, it is equally true that the ironmaster has often been ahead of the engineer, and has been prepared, long beforehand, to supply that which would make a success for the marine engineer if he would avail himself of it.

That eminent engineer and most fruitful of inventive geniuses, Trevithick, who a century ago saw instinctively what was possible with steam of high pressure, had to resort to sorry contrivances to try his experiments with the force which he knew would be so useful. His boilers, made of copper tubes in granite tube-plates, might serve as a monument to his genius, and as an example of the straits to which the circumstances of his time reduced him, rather than furnish proof of his ingenuity. It is true that since his day the tube- or tubulous boiler has been employed successfully to generate steam of high pressure, and that therefore the boiler in question might be taken as another example of his prescience. It is nevertheless a fact that he did not succeed until the ironmaster of his day gave him iron plates, or very broad bars, as we should be disposed to call them now. The dilemma that arose about the shaft of the *Great Western* was removed by the genius of Nasmyth, an engineer; but the steam-hammer without the men to work a forge would have been a useless tool, and the cast-iron shaft would have had to take its chance; so that I infer that as much skill and invention were shown by those men who had to manipulate the iron with that hammer for the first time, as was required to conceive, design, and make the hammer itself. History has not been quite fair, for although she has recorded the one feat, and has justly awarded her tribute of praise to the inventor of the tool, so far as I am aware there is neither name nor praise recorded of the men who worked it, nor of the ironmaster who erected it. I do not mean to say that the circumstances are not known, but there are now very few men who could give the information, and although a comparatively short time only has elapsed, their names are practically forgotten.

The Substitution of Steel for Iron in Boilers.

On the introduction of Siemens steel plates the old Staffordshire list of extras was practically abandoned by steel-makers. This had more to do with the progress subsequently made in boiler construction than the introduction of the material of superior strength, inasmuch as it made it possible to construct a boiler with the minimum number of plates at a less expense than one with almost the maximum number. The ironmaster could give us plates nine feet wide twenty years or more ago, but he wanted such a high price for them, that no boiler-maker who worked in competition with his neighbors had the courage to use them, although I am of opinion that it would

have paid to do so in all districts where labor was high. And that the boiler-maker did not use them was rather in consequence of his inability or disinclination to reckon the cost. At any rate, he did not employ large plates until they were practically at the same price as small ones, and thus boilers were constructed in the patchwork form until the era of Siemens-Martin steel. Here, as usual, it was the bugbear of cost that stopped progress, and this same bugbear does to this day often bar our way to the goal that some of us see so far off. You, gentlemen, have often proved yourselves to be our saviors, and we always look to you to remove the obstacles to progress that confront us, as also to the development of new departures. Without your help we would be at a stand-still; with it we can advance step by step; and I am bound to say, in justice to you, that when we make clear our demand for some new thing, you always tackle the problem forthwith, and sooner or later give us what we need. We have not yet, however, exhausted the list of things we want, nor have you yet satisfied all our longings for the things we have asked for. When you give us steel castings without blow-holes, steel forgings absolutely free from hair-marks an eighth of an inch below the surface, steel plates that know nothing of the "blue black" state, and that will not give way under any circumstances at a less strain than that to which they have been tested, we will not only be very grateful to you, but will ask for something else that shall give you the enjoyment of exercising your inventive genius and that splendid patience with which I know you are all endowed.

The period chosen for consideration in this paper is one that embraces the genesis of the modern development of the marine engine and boiler; but it is, moreover, the time that has elapsed since the introduction of Siemens steel, to which I attribute much of the progress that has recently been made. Previous to that time, iron was used almost exclusively in the construction of marine boilers, and very little steel indeed had been used in the engine. Boiler-shells were made of Staffordshire iron plates, or of such other iron as was equal to it in both quality and price; the insides, such as the furnaces and fireboxes, were made wholly, or nearly so, of what was familiarly known as "Lowmoor quality." The price of the latter was very high compared with that of the steel we now use, and may be taken as £30 ($145.80) per ton on the average. The Staffordshire iron at that time averaged about £9 ($43.75) per ton for a boiler made of comparatively small plates, and £12 ($58.33) for a boiler made by a more enterprising engineer with larger plates. There

was practically no limit to the boiler-pressure so far as the internal parts were concerned, but the working-pressure was decidedly limited in so far as it was governed by the shell. There were then some few firms which could supply a plate $1\frac{1}{8}$ inch thick of fairly large dimensions, and in some very few instances plates as thick as $1\frac{1}{4}$ inch were used. Taking the former as the more common and also the more convenient thickness, it meant that a boiler of 15 feet diameter would pass only for a working-pressure of 75 pounds; or if 100 pounds pressure was required, the limit to the diameter was 11 feet 3 inches for single-ended boilers and something less for double-ended ones. For a pressure of 150 pounds, the diameter would have been only 7 feet 6 inches. It need not be urged that this was prohibitive of higher pressures, although it had such an effect indirectly, inasmuch as the stays, etc., being likewise of iron, would have been either of monstrous size or so close together as to render the construction impracticable. The weight of such boilers was also very much against them, and altogether it required more courage than then existed, and certainly less caution than then obtained, to attempt such strides as have been since successfully accomplished. I have stated that there was no theoretical difficulty with the insides of the boilers, but, as a matter of fact, there were some practical difficulties to encounter. Lowmoor iron, when exposed to heat, was especially liable to blister from innumerable laminations, and these blisters caused trouble enough with the pressures and temperatures then ruling. What the troubles might have been with the thicker plates, higher pressures and higher temperatures was not known, and like all unknown terrors, they were feared most. It is true that means were well known for stiffening the furnace so as to withstand higher pressures; but these very means were often sources of mischief, and were always feared as likely to lead to accidents. It is true, also, that we had the locomotive-boiler working at pressures far beyond that of the ordinary marine boiler, but marine engineers were very chary of trusting to it, and events proved that their suspicions were not without foundation; for even now that the locomotive boiler has been used on shipboard to a considerable extent, its success at sea has been decidedly qualified, and no one will use it unless circumstances compel him to, and then only when he can get a supply of fresh water.

The supply of a material possessing all the good qualities of Lowmoor iron, without its bad ones, enabled Mr. Samson Fox, and subsequently others, to weld a cylindrical tube of almost any size, and

so manipulate it by corrugating or raising ridges as to render it proof against the pressures we now require. Thus was the internal difficulty solved. The steel which is the material I allude to, had an ultimate tensile strength 30 per cent. higher than that of the iron used in the boiler-shell, and it could be manipulated in a manner unknown with the older material, besides costing no more than Staffordshire iron. For the same dimensions and same pressure, a boiler made wholly of this steel was cheaper, lighter, and of better design and manufacture than one of iron. Natural conservatism, or some other occult force, seems to have prevented the marine engineer from taking the full advantage of the splendid substitute you gentlemen were prepared to supply him with. I must not forget, however, that there was another force, one of the strongest we have to combat in our professional work, which was decidedly opposed to the new material; that was prejudice. I feel even now tempted to take up the cudgels against that old enemy of the engineer, but in this case it would be waste of energy to thrash, as the dog is dead.

The Demand for Higher Speed.

So far this was the condition of affairs when fresh demands began to be made on the marine engineer. It had been demonstrated to the shipping world that it was possible to cross the Atlantic at a higher speed than thirteen knots, and that the ships possessing this higher speed made their voyages regularly, and, if rumor could be trusted, were paying. Certainly, both the American and the British public showed a preference for them, and there was a growing taste for speed at sea as there had been on land. The Inman and White Star companies had shown that fifteen knots were not only possible, but could be sustained with commercial success. But it was urged that, to get this speed, the ships must be very large, very costly in construction, and expensive in maintenance. This was quite true, and it remained to be seen if they would pay when there were more of them, with the consequent competition. The *Britannic* was built of iron, had iron boilers, and was propelled by ordinary compound engines. She has been a very successful ship in every sense of the word, and holds her own to-day; but her speed is no more than fifteen knots still, and so she has to depend on something besides speed for a character now. She is a ship built prior to the period I am considering, and needs not to be further discussed; she is taken only as one of the causes which created a new demand.

That demand was for more speed. Now, with the same conditions, more speed meant, for any ship, more indicated horse-power, more coal, larger boilers, and, finally, to get the speed economically, finer lines; in other words, less ability to carry the heavier weights entailed by the greater power. The *Arizona* was the first to supply this new demand. Iron was the material used, the compound engine the moving machine. The result was a gain of two knots per hour, equal to 13 per cent., with an increase in coal consumption of 50 per cent. The carrying capacity for the size of the ship was considerably less, the tonnage under deck being only 5 per cent. more. It will be seen that the cost of this excess of two knots was very great—first in the larger ship, then in the larger engines, then in the increased consumption of fuel; also that, owing to the extra weight of, and space occupied by the machinery, together with the extra coal to be carried in the bunkers, the new ship was less able to earn sufficient freight to meet the expenses of the older ship, and still less able to meet her own. It is, therefore, evident that such a ship must depend almost entirely on passengers and light goods of high value, to enable her to be run profitably.

In spite of all these serious disadvantages, the demand for such ships continued, so that the *Arizona* was soon followed by the *Oregon* and *Alaska*, still larger and faster ships. These were followed by the *Servia*, *City of Rome*, and *America*. Then came the *Aurania*, *Umbria*, and *Etruria;* and, finally, we have the *City of New York*, *City of Paris*, *Teutonic*, and *Majestic*, of 10,000 tons, running regularly at twenty knots on the Atlantic. These were the replies with which marine engineers and shipbuilders responded to the demand for greater speed. In fifteen years the speed has been increased from thirteen knots to twenty, and there is no reason to doubt that these ships pay very well in spite of their enormous cost and working expenses. Such ships are not only made to pay, but they are actually possible only as a result of what has been done by the steel manufacturer. Without steel in both hull and engines, such ships could not be built to accomplish the speed, or to safely withstand the strain of sea and machinery.

The *Britannic* was built of iron, had iron boilers, and her engines were made more or less of iron in one form or another. The *City of New York* is built of steel, has steel boilers, and her engines are, to a very large extent, constructed of steel. The saving in weight by the use of steel is very considerable in both hull and machinery, and as a result of this saving either more cargo can be carried, or, for the

same cargo, the ship can have finer lines, so that a higher speed is possible with the same horse-power, or the same speed with less horse-power and less consumption of coal.

Having thus briefly sketched the origin and history of the demand for more speed in the Atlantic service, I will pass from this part of my subject with the observation that if the owners or managers of ships in that trade had been well advised, they would long ago have separated the passenger- from the goods-trade. A steamer for passenger service only could have been built for the speeds now attained at far less cost in every sense of the word. It is only those who have to work out some of the problems set them by shipowners that appear to know, and to appreciate, the tremendous cost which is paid for the ability to carry the few hundreds of tons of cargo possessed by the "Atlantic greyhounds."

In 1875 there were many ships in the British navy capable of doing fifteen knots per hour on trial-trips, but very few that could maintain that speed at sea. The necessity for having war-ships that could maintain that speed arose as soon as the mercantile ships of Great Britain could accomplish it; for in time of war not only would the latter have to be convoyed, but the possibility of an enemy possessing ships of an equally high speed with which to harass the trade of the country was apparent. There was also the necessity for unarmored cruisers which, for obvious reasons, should possess a superior speed to battle-ships, and to act efficiently as sea-police they must have a speed superior to that of any ordinary ship afloat. Hence the demand for greater speed in Her Majesty's Navy.

The competition among the various steamship companies running in connection with railway-routes, and others in competition with railway-routes, caused a demand for increased speed in still a different class of ships. In fact, in every direction there existed a demand for more speed, so that there was every incentive to the shipbuilder and engineer to exercise his keenest wit in devising new ways of accomplishing this desirable end.

It will now be my task to show how this has been accomplished by the marine engineer, so that the public can now travel at higher speed, with greater safety, with increased comfort, and at no higher cost, than obtained on the ocean fifteen years ago.

I have stated that one way of obtaining speed was by making ships with finer lines, and this had always to be done at a considerable sacrifice of carrying power. Steel has, however, to a large extent obviated the necessity for such a sacrifice. New methods of

construction and new forms of material have also tended to this end. Ironmasters now supply at reasonable rates bars of Z-section, whereby the framework of the ship becomes much lighter, and at the same time stronger, than by the old combination of angle-bars. Moreover, this splendid steel that you now give us is capable of being so flanged, and that, too, when cold, that there no longer exists the necessity of riveting an angle-bar to the plate, if it is sought to stiffen it or to connect it to other work. You supply us now, at practically no extra cost, with such materials that we can construct our ships with much fewer plates, whereby we avoid the cost, weight, and disadvantage of so many joints. You are no doubt told from time to time that this new material has a serious fault—that with all its good qualities it is not as durable as the older one. This has a sufficient foundation of truth to make the opposition to it powerful; but with very ordinary precaution and care, experience has shown that it can have the same endurance; and there is no doubt that with a better knowledge of how to treat it, this objection will entirely disappear, just as the same objection to iron ships existed, and has now been almost forgotten. The best remedy is to get rid of the mill-scale before painting.

But while admitting that the naval architect has done much to bring about a successful solution of the speed problem, I must maintain that the marine engineer has done more. His path was not so obvious nor so clear before him, and new openings have often shown themselves in unexpected places.

Whether the marine engineer starts with a fixed weight given him by the naval architect, or whether the latter accepts such from the former, and designs the ship accordingly, is immaterial. The fact remains that the engineer, in order to get the maximum power, must get each individual horse-power with the minimum of weight, in order that the ship may be a success; and his problem has been how to do this without adopting too costly processes.

His first step in this direction was comparatively obvious. An engine develops so much horse-power at each revolution. If it can be made to do sixty per minute instead of fifty, the gain in power is 20 per cent; and if the speed was at first 13 knots, it should now be 13.8 knots, or a gain of nearly a knot per hour. But how can we get this increase in the number of revolutions, and is it to be done economically? So far as the engines are concerned, it would mean an increase to the condenser only; and seeing that nearly all steamers at the date in question had too much cooling surface, it

would probably have required no increase at all. But more steam was required for this increase of power, and so the engineer of that day had to fit larger boilers to his ship. With large ocean steamers this was always possible, but for the smaller steamer it was not always possible, neither was it possible for the small steamer to carry the extra fuel. It is not necessary here to go into details, but it is a fact that the insuperable difficulty with this class in the endeavor to obtain more speed was the impossibility of obtaining sufficient space for the necessary boilers. Some other engineer would, however, attempt the increased number of revolutions by using steam of much higher pressure, trusting to its superior efficiency to offset the paucity of supply. If his engines were able to digest, so to speak, this superior article, and to make full use of its superior efficiency, he was undoubtedly on the right track. Hence the success of the *Arizona* and the ships that followed her was due to these simple departures. Later your steel admitted of still higher pressures being obtained from the boilers of succeeding ships, and the three-cylinder ordinary compound engines admitted of a higher rate of expansion without any practical drawbacks, as would have been the case with the four-cylinder compound engines of the *Britannic*. The possibility of constructing boilers for higher pressures by the introduction of Siemens-Martin steel, and the fact that such boilers cost less than the old iron ones, tempted some engineers to use steam at a pressure of 110 pounds in the ordinary two-cylinder compound engine, but with disappointing results. I am aware that this experiment of ten or twelve years ago has been tried again, and that its new foster-parents state that the system gives as good results as the triple expansion system. They deceive only themselves, and the evidence they give, so far as I have heard it, is conclusive of one thing only, and that is that the new boilers and better arrangement of cylinders that they have adopted now give better results than those they followed formerly. I was quite prepared to hear that the substitution of two cylinders for four, and the use of steam at 150 pounds pressure expanding ten times would give better results than steam of 60 pounds expanded only six times; this is really what has been proved, and no more.

Having accomplished so much as I have mentioned, it remained for the engineer to pursue the path on which he had met with so much success, and we soon find him running even these large engines at eighty revolutions. Taking, as before, the increased power as proportional to the increase in the number of revolutions, viz., 60

per cent., the speed should now be 15.25 knots instead of 13. Revolutions have now done for him as much as he thinks prudent, so he has to start in a new direction, and this time it is in the nature of an enlargement of the engines. His brother engineer of the Naval Board has had to do this long ago, because he had started with a much higher number of revolutions—in fact, the mercantile engineer is taking up what the naval engineer had left off. The latter, however, is restricted in a way not known to the former; he can only increase the size of the engine by adding to the number of the cylinders, or increasing their diameter. He has tried the first plan ever since 1867, and until the compound engines forced themselves on him, Maudsley's three-cylinder engines had given him satisfaction, qualified, it is true, by a troublesome valve-gearing. The mercantile engineer, filled with gratitude for what piston-speed had done for him, now sees his way to getting a still higher speed by lengthening the stroke, so that in 1886 he is getting close upon a thousand feet per minute, with engines whose topmost part is visible from the deck without craning one's neck through the skylight. Steam is supplied to the monster by many huge steel boilers, and the ship can carry but little more than her fuel-supply for the single voyage.

The Triple-Expansion Engine.

But before this climax has been reached, a veteran engineer has turned once more to an old idea of his own. His first experiment with it had failed through no fault of it or of his. He finds that with this new good steel he can now build a boiler of a type well known and tried, and thereby get steam of the pressure he requires with certainty and safety; and in 1881 Dr. Kirk astonishes the shipping world by the success of the *Aberdeen*, having similar engines to those of the *Propontis*, which were made by him in 1874, but failed. And why were ship-owners so interested in the success of the *Aberdeen*, and what after all was the nature of her success? Her steam-pressure of 125 pounds was but very little higher than what some other ships were then using, and at that time coal was as cheap as it had ever been; so that, although fuel-economy was and always is of importance, it was not in itself sufficient to cause a revolution so sudden and complete as that which followed. The fact is, here was the real solution of the speed question.

Since then, we have all learnt that the triple engine burns less coal, can be run at a higher speed, and has less bulky boilers to

supply it with steam than the old compounds, and that was what we were all wanting to obtain—higher speed in every class of ship, even in the so-called ocean tramps, which now glory in a ten-knot speed, whereas formerly the skipper was a proud man who could maintain a level eight knots throughout the voyage. The success, however, of the *Aberdeen* was something more even than this. Here was a ship of moderate size, good speed, and good carrying capacity, that was going to make a very long voyage by a route on which the coaling-stations were far apart, and at some of them the coal was very dear. Steamship owners could now see before them fresh fields for enterprise, and the owners of sailing ships learnt before long that this same Dr. Kirk had created poachers on the realm so long held by them. The sailing-ship has not been driven off the high seas, but steamers now run on tracks unknown to them till the triple engine existed.

The failure, in 1874, of the *Propontis* was due to the want of a suitable material for boiler-making, which by 1881 you gentlemen had supplied to us, and thereby permitted such a success to her designer that the trade of the world has been revolutionized. The saving of fuel has been stated, challenged, debated and discussed *ad nauseam*. The fact remains that it is substantial, and is approximately 20 per cent. where all other conditions are the same. Experience has proved that the three-crank engine is capable of being run at higher speeds than the two-crank engine, and that its wear and tear is absolutely less. As 20 per cent. less steam is required, the size of the boilers can, of course, be reduced by 20 per cent., or boilers of the same size will supply steam for 20 per cent. more power. This latter fact is the one by which, partly from design but oftener from unbelief, the generality of ship-owners have profited. To it the superior speed of many of the "tramps" is due, while that of small passenger steamers is almost wholly attributable to greater engine speed.

On the whole, the saving of coal by means of the triple compound engine has been rather in the direction of the allowance per horse-power than per ton carried, as the ton carried is now almost universally carried at a higher speed. The actual saving is nevertheless a very substantial one, both to the world and to the individual ship-owner, for a ship, if run at a higher speed, makes more voyages in a year, so that the trade is done with less capital, which means less tonnage. The consequent need for less tonnage no doubt depresses the freight rates, and so the unfortunate ship-owner does not reap

the harvest, but only as it is shared by the public at large. The continued depression in shipping is undoubtedly due to Dr. Kirk's genius, and hence he gets so little praise.

Forced Draught.

Mr. Thorneycroft,* now so well known as the designer and builder of steam-launches, and later of torpedo-boats, soon had to give up non-condensing engines and condense the steam so as to get fresh water for feeding the locomotive-boilers he used. The loss of means of draught in the escape-steam caused him to substitute artificial sources of air-supply, and he found that by forcing air under the grates he could get an increase of steam supply over that given formerly by the steam-blast. The pressure of air was increased from one half inch to as much as six inches of water, corresponding to that of the fiercest draught of an express locomotive; the supply of steam was excellent, and the speed obtained with a steam-launch of considerable size and torpedo-boats larger still was very high, and was at the time considered marvellous. The means whereby these improved results were achieved were in themselves simple and inexpensive. The compartment containing the boilers was made air-tight, and into it was forced a supply of air from the deck by means of one or more fans of the ordinary type, driven by an independent engine. The simplicity of these means, and the success attending the experiments, led our naval engineers in 1880 to adopt them on a large scale in some cruisers of the *Leander* class, the result being that with precisely the same engines and boilers, and with only the addition of the forced-draught arrangement, a considerable increase of speed was achieved. Since that date, similar forced-draught arrangements have been fitted to all the new ships built for the British navy, and the example thus set has been followed by the engineers of other countries.

As an illustration of what can be done in this way, it is sufficient to instance the case of the first-class belted cruisers of the *Immortalité* class. When working under natural draught, the power to be indicated was 5500 horses, and on the trial trip as much as 6000

* I have given Mr. Thorneycroft the credit of the modern application of forced or artificial draught, but the fact is that the first attempt in this direction was made by Ericson in 1828 to the steamer *Victory*, commanded by Sir John Ross, who, strange to say, enters in his log that "the boiler still continued to leak, although we had put dung and potatoes in it, as directed by the engineers."

horse-power was developed. With forced draught, due to an air-pressure equal to two inches of water, it was expected that 8500 horses would be developed by the engines; but, as a matter of fact, on the trial-trip and under these conditions over 9000 horse power was obtained, and the speed was raised from sixteen and a half knots to nineteen knots. At the present time, speeds of twenty knots per hour are being realized in Her Majesty's Navy with engines and boilers of such a size as ten years ago would have been deemed sufficient for a speed of only fifteen knots. It is, however, a fact, I regret to say, that in running continuously under forced draught the engineers of the British navy experience more or less trouble with the boilers. This is not, in my opinion, due to the system, but to the want of suitable provisions in the boiler design for the conditions under which they have to work when being pressed to the utmost. It is, in fact, a little error of judgment on the part of the authorities, into which they have been led by the perfect success of their previous efforts in the same direction; and just as the locomotive engineer of to-day is dependent on, and has the utmost faith in, the forced draught by which he gets express speeds, so the marine engineer will get over his difficulties as the locomotive engineer did, so that in the near future we shall see ships whose boilers are worked under forced draught with perfect safety and confidence doing express speeds.

In the days of natural draught and compound engines, 10 horse-power per square foot of grate was thought to be a good result; with the introduction of the triple engine and higher pressures a natural draught would give 14 to 15 horse-power per square foot of grate; and the modern forced draught now used in large ships enables us to get 20 horse-power per square foot of grate; and in smaller boats, where the air-pressure in the stoke-holes has been raised above two inches, even higher results have been reached.

With the success of the principle of forced draught came the usual number of enterprising inventors with their devices, some of them good, others not worthy of notice, but all more or less misconceived so far as the objects aimed at and the end to be attained were concerned. As a means to a particular end, forced draught has been a success; as a means to the several ends held in view by experimenters, it has not been a success. When the consumption of a larger amount of fuel per square foot of grate, with a consequent increase in the supply of steam, was the object, no one could complain; but that there should be an economy of fuel per horse-power

was not to be expected, although it is true that with some kinds of coal, and in some kinds of boilers, the more rapid combustion and the more intense heat obtained, have been attended with a slight economy in consumption. It is claimed by certain inventors that by their patents economy is effected, inasmuch as the air forced into the furnaces is to some extent heated by the waste-gases in the up-take and funnel; but seeing how low the specific heat of air is and the short time to which it is exposed to the heating process, it is very doubtful if the gain would outweigh some of the disadvantages of the system; and, on the whole, I am inclined to think that our Admiralty system of closed stoke-holes is the best, and that economy must be sought in the better firing that is possible under these conditions than to the heating of the air supply,—not that I by any means undervalue the advantage of the regenerative process.

Where forced draught has been used in the mercantile marine for speed purposes, I hear of little or no complaint, but when introduced for the sole purpose of effecting economy of fuel, the praise is of a very qualified nature, and the complaints, when made, are unmistakable.

To sum up, the three great *developments* of marine engineering that have been made in the past ten years are:

1. Increase in working-pressure from an average of 75 pounds to an average of 150 pounds per square inch.

2. The adoption of the triple and quadruple expansion system, so as to utilize steam of this high pressure efficiently.

3. The adoption of artificial means for obtaining sufficient draught for the consumption of coal, and when desired, for an increased consumption of coal, so as to give an augmented supply of steam without additional boiler capacity; with the result that in naval ships the indicated horse-power for practically the same weight of machinery is nearly doubled, and in the mercantile marine the indicated horse-power is 50 to 75 per cent. more for the same weight of machinery than was used fifteen years ago, the consumption of coal is 20 per cent. less than ten years ago, and 25 to 30 per cent. less than fifteen years ago.

The Use of Steel and Hard Alloys in Engines.

I now come to the *progress* that has been made by the marine engineer in the design and manufacture of machinery, whereby the

engine of to-day, notwithstanding that it is lighter and even cheaper than that of fifteen years ago, is worked with less wear and tear and fewer accidents. Although the tremendous demand for sea-going engineers has militated somewhat against the quality of the supply, so that the marine machinery of to-day is probably not in the hands of such experienced and skilful men as then, yet at the end of twelve months' work it is in better order now than was the case fifteen years ago. I do not wish by any means to disparage the skill and ability of our sea-going engineers, but I think that I shall be supported in saying that younger faces are to be seen in important positions in the engine-rooms to-day than was the case fifteen years ago; that whereas a natural taste and aptitude for the profession were formerly a *sine quâ non* for entrance to it, it is now flooded by young men who simply regard it as being more lucrative than some other professions and trades to which admittance is easy.

That the constant complaint of hot bearings has become practically a thing of the past is due partly to improved construction and better design consequent on a better understanding of first principles; but good white metal has done almost more, as it has been found an efficient cure for the evil in places where it has existed for years in the older engines; hence we find that while formerly it was only war-ships and those of the best mail-ship companies that indulged in the luxury of white metal, now, as a matter of economy, the meanest of steamships usually has this metal in the main bearings and crank-pin brasses; for a hot bearing always means an expensive bearing, both in coal-consumption and in wear and tear and renewals. The increased length of life of crank-shafts is due to the absence of hot bearings, as well as to the lighter strains put upon them. It is white metal which has permitted the high number of revolutions per minute that I have already pointed out as one of the conditions of high speed in ships.

I would also express the opinion that the crank-shaft of to-day is better made and of better material than was the case fifteen years ago. Improved methods of manufacture of iron forgings, and the improved steel which you, gentlemen, give us, admit a better surface, and the well-made and heavy lathes of the present day can do the work not only more cheaply, but more accurately, than was the case then. It is to be hoped that the combination of the crank-shaft makers in England, whereby the price is kept up, will act in maintaining the quality, and, if possible, raising it beyond what it was when competition was so severe.

In the construction of the modern marine engine the most marked feature of change is the quantity of steel castings introduced, sometimes in the place of forgings, sometimes in the place of iron castings, and sometimes in the place of brass castings. I ventured more than twenty years ago, when I was a student of engineering, to predict that the time would come when we should get steel cast into the various forms we required, and that, with the introduction of such a metal, cast-iron would gradually disappear from the marine engine. In a sense, my prophecy has been fulfilled sooner than I had expected, but cast-iron has not been driven from the field as rapidly as might have been expected after the introduction of steel castings. The fact is that you, gentlemen, have not yet learned the whole secret of success; and I do not say this reproachfully, for so many of you have worked energetically at the problem, and have improved in your productions in such a comparatively short time that you deserve praise rather than censure. We engineers do complain—sometimes very savagely—of the delays we experience in obtaining steel castings, and of the loss to us when, after machining for days, we open up such faulty places as necessitate the condemnation of the castings; but in our calmer moments we are fain to admit that our wonder should rather be, not that bad castings are made, but that we occasionally get good ones. Ten years ago my firm commenced using steel castings in lieu of iron ones for such parts as were subject to shock, and were of such plain design as to be easily cast. We then substituted cast-steel for forgings on which the strain per square inch was very light. For these purposes all we required was a material whose ultimate strength was at least 20 tons, and which, before breaking, would stretch at least 5 per cent. At that time our particular sources of supply were Messrs. W. Jessop & Sons, Limited, Sheffield, and Mr. Aesthöwer, of Annen, Germany. The German castings were very sound, and came up to the standard laid down, but went very little beyond it; while those of Messrs. Jessop & Sons were only fairly sound, but they possessed a tensile strength of 30 tons, with an elongation of 20 per cent., and in some cases they were even better than this. With these latter we ventured to go further than we had done before, and we made of them some of the smaller working-parts, which were subject to higher strains, and, eventually the connecting-rods for two engines of considerable size. These rods have now been running for seven years without showing signs of failure. Messrs. Jessop have made crank-shafts of large size of this material, even to 15 inches diameter, I

believe, with most satisfactory results. The machined castings were apparently free from blow-holes, draw-cracks, and even sponginess, while they had all the appearance, when finished, of having been forged. Pieces cut from these castings were found to have as high a tensile strength and as good an elongation as a steel forging, so that, from *prima facie* considerations, a shaft of such material was equal to a forged one, and I am not aware that experience has proved the contrary. To-day we are constructing in the works of my firm (Earle's Shipbuilding and Engineering Company, Limited, Hull, England) engines of 12,000 horse-power, in which the foundations, the columns, the pistons, the thrust-blocks and collars, the eccentric-straps, the cylinder and valve-box covers, and many other minor details are made of cast steel. The saving of weight is such that, taking *one* of these engines—they are twin screws—and half the boilers, that is to say, taking a set of machinery of 6000 horse-power, its weight is 560 tons, against 900 tons for a set of engines, boilers, etc., of 6000 horse-power in 1870, and 1000 tons for the weight of compound engines in 1875, which at that time were deemed the lightest of the kind, and were run at very nearly the same number of revolutions as the modern engine. The ship for which our engines are intended is being built by us, and has a cast-steel stem, a cast-steel stern-post, a cast-steel rudder, and cast-steel propeller brackets, besides other cast-steel fittings for gun and other purposes throughout the vessel. The fact is that our demand for steel castings is to-day large, and, moreover, is growing; and when you, gentlemen, by means of aluminum, nickel, or any other "doctor," will give us sound castings that we can always rely upon, and avoid wasting our money on machining only to discover defects by which the castings are condemned, we will prove even better customers than we now are.

In 1875, Messrs. Vickers, Sons & Co., of Sheffield, were making very large crank-shafts and other shafts; and in the Atlantic service it was becoming customary, when a shaft broke, to substitute one of Vickers's for it. They were, however, a very expensive luxury; and as the wrought-iron shaft generally gave its owners timely warning when it was going to break, and the steel one did not, the latter was treated as untrustworthy, and was not adopted by the majority of engineers or ship-owners. To-day, instead of there being only one firm that can make these shafts, we have a dozen, and at least half a dozen of these whose reputation stands in the first rank. The Siemens furnace now produces steel for shaft purposes at as low

a rate as ruled for the Bessemer in 1875. The substitution of hydraulic presses for steam-hammers, too, has not only improved the quality of these forgings, but most undoubtedly has been the means of reducing their price, and we now have steel crank-shafts, steel propeller-shafts, steel piston- and connecting-rods—in fact, every forging of steel in a war-ship and in the highest class of mail-steamers, and throughout the mercantile marine there is a general tendency to substitute this superior metal for the inferior one. In H. B. M. Navy the shafting is made hollow, in accordance with the plan introduced by myself in 1872, whereby a considerable reduction is effected in weight with an exceedingly small reduction in strength. For example, a shaft 10¼ inches external diameter, with a hole 5 inches diameter through it, is equal in strength to a solid 10-inch shaft, but its weight is only a little over three-fourths that of the solid one. I need hardly say, however, that the difference in cost is considerable; in fact, there is the added cost of boring out the hole. Siemens steel is now but very little dearer than ordinary crown-quality iron, and no dearer than the best quality of iron required by marine engineers, so that all the bolts, studs, nuts, and pins used by my firm are of steel.

The general effect of the substitution of steel for iron in the construction of a marine engine is to save weight, and at the same time to obtain a stronger machine. As the modulus of elasticity of steel, however, is about the same as that of iron, there are many parts of the engine in which no great decrease in scantlings is permissible, but yet on the whole a decrease can be effected, and so the modern engine, composed almost entirely of steel, is lighter than one of iron. The collapse in the *City of Paris* (s.s.) has no doubt shaken the minds of some as to the use of steel castings; but I think it has been proved most conclusively at the trial that the accident was not in any way due to the steel castings, and that these did give way is not perhaps to be wondered at under the circumstances. At the same time I am of opinion that it is unwise, with our present knowledge of the art, to cast columns hollow, *i.e.*, cored throughout their length. It may, of course, be possible to do this successfully, but the risk is great, and the liability to obscure cracks and unseen faults must be great also. At all events, my experience with the material has taught me to avoid such a form as far as possible, and certainly never to make an important part, like the column of an engine, of a hollow casting, more especially one with a thin section.

I do not suppose it is of equal interest to my audience, but nevertheless it is a fact, that very marked improvements have been made in the manufacture of bronze castings and also of bronze forgings. Fifteen years ago a gun-metal casting, with an ultimate strength of 15 tons per square inch, and an elongation of 5 per cent., was considered very satisfactory; but to-day we have the various bronzes, such as phosphor-bronze, manganese-bronze, Stone's bronze, aluminum-bronze, etc., having strengths varying from 20 to 30 tons per square inch, and in some instances even a higher ultimate tensile strength than 30 tons, with an elongation approaching that of steel. Forgings made from some of these metals have proved to be equal to steel in strength, and the cost of these bronzes is very little beyond that of gun-metal. Advantage has been taken of this by the marine engineer to use forgings of these metals in lieu of brass castings, drawn bars in lieu of either brass castings or iron forgings cased with brass, and, finally, to make propeller-blades of bronze instead of cast-iron, steel, or even gun-metal. I will not detail here the gains in speed attributed to the substitution of the bronzes for steel or iron, but content myself with accepting as a fact, and stating the same to you, that bronze propeller-blades being made of thinner section, having keener edges and a smoother skin, give a better result than the heavy cast-iron blades did, or than the steel can possibly give when pitted and roughened by sea-water. The action of sea-water on these bronzes is sometimes capricious, but usually only slight. The English Admiralty, acting upon their own experience, have discarded the use of all the bronzes except phosphor-bronze and ordinary bronze or gun-metal (consisting of 87 per cent. of copper, 8 per cent, of tin, and 5 per cent. of spelter). So far this distrust of other bronzes is not shared by the mercantile marine. I have, however, seen some remarkable failures with certain of the bronzes. For example, a new propeller fresh from the makers was found, after lying on a wharf for a few weeks subjected to severe frost, to be cracked so seriously as to be utterly useless. It is also not unknown to vessels of the mercantile marine to lose one or more propeller-blades without notice; and as indicative of the direction in which engineering opinion may be tending, the latest novelty in propeller-blades is a bronze casting with a cast-steel core, or perhaps you, gentlemen, would prefer to say a steel casting with a bronze skin.

The progress in propeller design and construction has been very slight, although each month seems to produce a new inventor with

a new propeller—at least new to the inventor—that is going to revolutionize the shipping world; but in spite of newspaper paragraphs which are full of hope and promise, if, alas! wanting in scientific and technical knowledge of the first principles of propulsion, the screw-propeller remains without a rival. Its shape has not been altered much, although the very bad forms have gradually died out; and although much has been claimed for forms having a pitch that varies in all kinds of ways, the fact remains that in the navy, as in the mercantile marine, those who have given most thought and observation to the subject prefer a propeller of uniform pitch throughout—in other words, it is desired that its acting surface shall be part of a true helix.

Propulsion by twin-screws has many practical advantages, but it is now adopted mainly from considerations of safety, inasmuch as a ship with two screws is not apt to have both injured at the same time, and, therefore, less liable to a break-down of her whole propelling apparatus than a ship with the single screw. In case of accident to the steering gear, moreover, she can be steered by varying the revolutions of the engines. But there is also a consideration of efficiency: a smaller propeller will suffice for each of the twin-screw engines than would answer for a single engine of their combined power; hence in the case of a deep-draught ship the more deeply immersed twin screws work with a higher efficiency, and in the case of a shallow-draught vessel the same holds good, inasmuch as the twin screws are thoroughly immersed when the single screw of their combined power would be partly out of the water.

Improvements in Feed-Water Supply.

The adoption of steel boilers naturally suggested the advisability of using fresh water, not so much on grounds of cleanliness as of avoiding corrosion. This bugbear, however, like many others, has been shown to have no grounds for existence. Experience has demonstrated that the steel boiler does not corrode any more rapidly than the iron one; and, as a matter of fact, the life of boilers made of steel is likely to be very considerably longer than that of iron ones. The first steel boilers made by my firm twelve years ago are working at their original pressure, under the inspection of the British Board of Trade and Lloyds, and they are likely to continue to do so for many years to come. I am, of course, aware, and would remind you, that this increased length of life is due in no small degree to

the better treatment accorded to them. The use of zinc in the form of cast slabs or rolled sheets, especially in the earlier months of their life, has been the means of preserving boilers in a way not previously known; and the general practice of admitting the feed-water at a comparatively high temperature has also tended to increase their durability. But I repeat that under the same conditions the steel boiler shows no tendency to corrode any faster than the iron one, and the steel boiler has, besides, the advantage over the iron one of not developing blisters in the internal parts exposed to flame, so that the patching of furnaces has become a comparatively rare thing.

The demand for fresh water for feed-purposes has been emphasized since the use of steam of 150 pounds pressure and upwards, especially for vessels making long voyages. It is obvious that the power to generate steam in any boiler must decrease very quickly as the scale on the heating-surface increases, and also that the temperature necessary to produce steam of a certain pressure rises as the density of the water increases; hence, when the waste is made up with sea-water in boilers whose working-pressure is 150 pounds and upwards, the speed of the ship decreases towards the end of the voyage, and, what is worse, the boilers often give out, and require extensive and costly repairs. The consumption of coal is also very much higher than it would be if the boilers were kept clean throughout. Attempts to remedy this evil were first made by carrying a supply of fresh water in the double bottoms of the ships, or in tanks specially provided for the purpose. This expedient helped matters, but was not a perfect success, as the water-supply often contained large quantities of sulphate and carbonate of lime, which formed a hard scale, and which, together with the sea-water, made a worse scale than the sea-water itself—that is, worse to deal with afterwards. This water also cost money, and, what was more objectionable, the carrying of it often reduced the quantity of cargo. Other attempts were made to supply fresh feed to the boilers by means of the donkey-boiler, the steam from it being condensed and pumped into the main boilers. Then, however, the donkey-boiler became the sufferer, and, as a rule, it is not of a type that can be easily cleaned, so that it was only a matter of time for this source of supply to fail, and to become unavailable for several days. Finally, Mr. James Weir, of Glasgow, whose inventions, especially for improving the efficiency of the boiler, are of wide repute, introduced his now well-known evaporator,

which consists really of a small horizontal boiler, ingeniously contrived so as to be easily taken to pieces and cleaned, and the water in it evaporated by the steam from the main boilers passing through a set of tubes placed in its bottom. The steam generated in this boiler is admitted to the low-pressure valve-box, so that there is no loss of energy, and the water condensed in it is, of course, returned to the main boilers. The extra supply is still obtained at the expense of coal, but without any risk of injury and at a more economical rate than was the case when a donkey-boiler was employed. Other engineers have designed and patented apparatuses to do the same work, and it is now getting to be a universal practice for all ocean-going steamers to have feed-evaporators.

Mr. Weir's feed-heater, whereby the feed-water, before entering the boiler, is heated up very nearly to boiling-point by means of the waste-water and steam from the low-pressure valve-box of a compound engine, is another most useful appliance, and was introduced in 1878; it is also coming more and more into use every day, and, in conjunction with Mr. Weir's special feed-pumps, is a very desirable thing to have on every steamship, as the feed-water is delivered to the boiler not only warm, but with such regularity and such very slight risk of cessation, as to render the engineer on watch quite easy on the score of feed-supply. It is claimed for many feed-heaters that they are the means of effecting an economy ranging as high as 15 per cent. I mean marine feed-heaters, where the heating is done by steam, and not by waste-gases in the uptake. It is not obvious how there can be any saving at all, and in my opinion the tens and fifteens per cent. are apocryphal. That steam is better maintained in the boilers with such appliances is undoubted, and that in cases where difficulty has been experienced in keeping up the full pressure, the fitting of a feed-heater has been the means of doing it; so that there must be some saving, although in the nature of "robbing Peter to pay Paul." The fact is, the warm feed assists rather than retards the circulation, and the saving is more in that direction than in any other.

Conclusion.

Improvements have also been made in the engine-room and boiler-room of nearly every steamer in the direction of making the work lighter both at sea and in port, and the occupation of the engineer less severe. Ventilation has been better understood and practiced. The covering of the hot parts by non-conducting material has been

extended, and the substitution of superior non-conductors, such as fossil-meal and asbestos compositions, has unquestionably effected a saving. Steam starting-gears, too, have during the last fifteen years been introduced into all classes of ships, and even the smallest of them may now be found provided with this appliance for the more rapid handling of the engines. Steam turning-gears and steam ash-hoists have also been introduced, and all such fittings must naturally tend to the more economical working of the vessel.

In conclusion, I would summarize by saying that in fifteen years the speed of passenger steamships has been increased from 30 to 40 per cent.; the consumption of coal per horse-power has been decreased from 20 to 30 per cent., or say an average of 25 per cent.; the cost of a horse-power has been decreased by almost the same amount, and in some cases is but a half of what it was in 1875, and the safety, comfort and convenience of the travelling public has been enhanced instead of sacrificed. These successes are largely due to the makers of iron and steel, and the remainder is due to the skill and ability of marine engineers, but it must not be forgotten that the whole has been backed by the enterprise and sagacity of the ship-owners.

ON WELDING BY ELECTRICITY.

BY PROF. ELIHU THOMSON, BOSTON, MASS.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

THE subject of welding by electricity has been so recently and so ably treated by Sir Frederick Bramwell before the Institution of Civil Engineers* as to render evidently superfluous the re-statement on this occasion of many facts and considerations which might be appropriately advanced, had they not been so well discussed in his paper already.

The electrical arts grow rapidly in these days—a fact which is strikingly illustrated in the development of electric railways. In

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* *Proc. Inst. Civ. Eng.*, Session 1889–90, Part iv., vol. cii., p. 1.

welding and metal-working by electricity the progress is very rapid, and many facts concerning this new art have not thus far been brought out in any publication. Moreover, it should be noted that many new improvements are not ready for publication, on account of the necessity for reserve, until the inventors shall have secured the protection to be afforded by patents.

The purpose of the present paper is to call attention to some of the more prominent steps in the progress being made in the development of electric welding and metal-working, it being understood that even while this present paper is in preparation, advances are taking place in the working out of details. It is not, therefore, intended to have this paper take the character of a complete statement, but rather that it shall be a brief review of some phases of the development of the art.*

In this art, the foundation experiments were made about four years ago. The author had the pleasure of describing some of their results, particularly relative to electric welding, in a paper written in the latter part of the year 1886. The appearance of that paper aroused a considerable interest in the subject, which has continued from that time, and it is curious to note how it has caused the revival of early attempts to use the electric arc in metal welding or working, sometimes in conjunction with a magnet for displacing the arc and converting it into a blow-pipe of great intensity, the latter being a procedure due, it is believed, to Werdermann, in the early days of the production of electric arcs from dynamo machines.

It is not my present purpose to deal with any of these arc methods of electric metal-work. Their practicability is doubtful. So far as is known to the author, there are no existing instances of the use in practice of the electric arc for welding, though numerous attempts have been made, which have failed from one cause or another.

The operations with which this paper deals are those in which the heating effect of electric currents traversing a solid metal conductor gradually brings the metal to the working temperature. This temperature is, in the case of easily fusible metals, much below a red heat, and is hence unattended with luminous effects, which, however, appear in the case of metals softened or fused with greater difficulty.

In order to distinguish this process, applied to the welding of metals, from the application of the heating effect of an electric arc

* In illustration of the subject the author exhibited a large number of lantern-views of machines for electric welding.

for like purposes, it has been called the incandescent method of electric welding—a designation which is, in my judgment, misleading, for the reason that lead, tin, zinc, and other metals, fusing far below the temperature of incandescence, are thus welded.

Electric welding occurs between pieces of all metals thus far tried, though the perfection of the joint obtained varies with the metals and with the conditions under which the work is performed. Many different metals unite each to the other, and in many cases the union is such as to possess a strength or tenacity equal to that of both, or of the weaker of the metals joined. In other cases it is not so strong, owing to wide differences in the physical nature of the metals or alloys, or in their tendencies to surface union. Perhaps the incipient alloying at the joint may affect the strength, according to the nature of the alloys possible to be thus formed.

The pieces to be united in electric welding may be in the form of bars of various cross-sections, or in the form of tubes or pipes to be abutted and united, or in the form of plates. It is evident that a pipe is virtually a plate rolled up with its edges joined. The form of the pieces is of little moment, provided that they permit secure clamping for the passage of the current and manipulation. The surfaces to be united are first brought firmly into contact, the pieces being held in heavy conducting clamps for the purpose of carrying a current into them. The current having been put on, heating of the metal between the clamps takes place to a degree which may be regulated at will, or may be automatically controlled. The pressing of the surfaces together unites them, if the joint be a butt-joint, while a lateral or transverse pressure, such as that of hammering or pressing in dies, is used to make a lap-joint. The hammering or pressing is also useful with joints in iron or steel when butt-welding is done, particularly where great strength is required, and the original section at the joint is to be preserved. Such hammering or pressure may be applied simultaneously with the progress of the welding operation, or immediately thereafter, while the metal is still hot.

It is proper here to correct an impression which has gained some currency—namely, that it is the extra resistance caused by the break or limited contact between the meeting portions of metal which gives rise to the heating in electric welding. While this limitation of contact-surface undoubtedly hastens the heating at the joint it is nevertheless a fact that a solid bar, joining the clamps of an electric welding machine, will be heated between those clamps to welding tem-

perature, and may be upset by the approach of the clamps one toward the other. Indeed, this process is actually employed to upset collars on shafts, or to set or fasten in place, by a sort of riveting action, collars which have been placed as rings, more or less closely fitting, upon a bar or shaft. The real cause of the concentration of the heating effect at the joint, or between the clamps, is the relatively greater conductivity of other portions of the welding circuit, which circuit is usually composed of massive copper conductors, kept cool, in the case of large work, by circulation of water. By thus keeping the clamps cool, as well as the conductor in which the welding currents are generated, their conductivity is preserved from running down and there follows a greater accentuation of heating effect at the joint, with the added result of a considerable saving in the electrical energy required to do the work. In the larger work it has also been found that hydraulic pressure may be advantageously employed, both for clamping and making contact with the pieces to be welded or worked and for pressing the pieces together in forming the joint. The pressure can be regulated in this case by a pressure governor to a constant amount for a piece of given size. A scale of pressures corresponding to certain sections and materials can be made beforehand, after which the operation may proceed without skill or knowledge on the part of the workman, whose duty is then confined to replacing pieces in the clamps and turning on the current of pre-determined force.

It is not the purpose of this paper to deal to any considerable extent with the technical details of the construction of apparatus, because there is not time for such a discussion, and because these arrangements are subject to wide modification, in accordance with the nature, size, and other conditions of the work. In the practice of electric welding, the conditions of holding in the clamps, the pressure of the pieces together, and the volume of current required, can be determined at the outset, so that a given piece of work can be repeated without variation of result. The operation is, therefore, capable of being made automatic. The present tendency in the construction of electric welding machinery is towards having its action in most respects automatic, so as to leave little to the skill or dexterity of the operator in charge.

The apparatus first used by the author consisted of an alternating-current dynamo, feeding a comparatively high-potential current to the primary coil of an induction-coil or transformer, the secondary of which was made so large in section, and so short in length, as to

supply to the work currents not exceeding two or three volts and of very large volume or rate of flow. The welding-clamps were attached to the secondary terminals. This first type of apparatus has survived to the present day, and most of the apparatus is constructed on that principle, the secondary coil usually consisting of a single turn only. Other forms of apparatus, such as dynamos constructed to yield alternating currents direct from the armature to the welding clamps, are used to a limited extent. While currents of one direction are quite suitable for electric welding, no special apparatus has been constructed for using them. Currents from storage-batteries are quite suitable for the purpose, but the batteries should be constructed to stand very great rapidity of discharge, otherwise they become very cumbersome and difficult of application.

The practical results are not very different, whether continuous currents or alternating currents are used, though it is to be noticed that in the discussion of Sir Frederick Bramwell's paper the actions of the two classes of currents were spoken of theoretically, to some extent, as different. It has been shown that in conductors of large section the tendency of alternating currents is to keep to the surface of the conductor or to flow in the part of it farthest away from its center. This is undoubtedly true as a tendency, but how far this tendency will be allowed to develop itself will depend on the drop of potential in a given length of conductor, a circumstance which appears to have been overlooked in many of the discussions of this interesting question. In other words, if the fall of potential, as in welding, is, say, two volts in two inches of length of the bar welded or worked, the density of current will be great, even in the center of the section of the bar, but of course much greater near the exterior. But the heating-effect also requires to be greater near the exterior an account of radiation taking place at the outside surface. The practical result may easily be that there is really a uniform heating-effect throughout the section of the bar; and without doubt the conditions could be selected to secure such a result.

In welding the ends of two bars, however, they are frequently rounded, and at the start meet only in the center. In such cases, whether the current be alternating or continuous, the heating-effect begins at the center. A curious fact may here be mentioned. It is found that the distribution of the alternating current may easily be controlled by increase or decrease of self-induction at various parts of the section of the bar by the proximity of magnetizable iron masses. This principle has been applied in a variety of ways to

govern the distribution of the heating-effect, and also to effect the restriction of current to the weld in thick rings. This will be adverted to later.

A glance at some of the typical actions and conditions found to exist in the welding of different metals may not be out of place. With metals such as lead, tin and zinc, the degree of heat required does not, of course, produce light, and the progress of the heating cannot be watched by the eye, as in the case of iron or steel. Therefore, the plasticity, softening or fusion is really the index of the heat which the abutted ends have acquired. If the precaution be taken to shape properly the meeting ends of the sections before applying current and pressure, the joining of lead-pipe sections, end to end, can easily be accomplished electrically, and the joints are very good and sound. The welding occurs best by an actual fusion of metal, a sort of autogenous soldering, or soldering of lead with lead. The meeting edges are best made beveled or tapered so as to expose a smaller section than that of the pipe or bar to the melting action of the current. On account of the poor conductivity, for heat and electricity, of lead and its alloys, comparatively large sections may be worked by moderate current strength as compared with copper.

The metal tin welds electrically with ease, as does also zinc, and even such brittle metals as antimony and bismuth present no difficulties. The welding of aluminum requires special precautions, but can be readily accomplished, and the joints, when properly made, are exceptionally strong and tough. In this case recourse is generally had to the automatic welder, in which machine the conditions prior to putting on the current can be very accurately set. In fact, while by hand-work a considerable proportion of failures will occur in aluminum-welding, by the automatic machine good joints may be made in regular succession. Even magnesium, oxidizable as it is, is readily welded because it melts before reaching the temperature at which it takes fire. In the welding of brass to brass, the conditions for the most successful work vary somewhat with the nature and composition of the alloy, and here it may be remarked that a much greater current-strength is required than in the case of iron of equal section, although the specific electric resistance of cold iron is nearly the same as that of brass, and the latter metal is much more fusible. This requisite of larger current for brass is due to the fact that with brass and similar alloys, such as German silver, only a comparatively small increase of resistance takes place with the increase in

temperature, while with iron the resistance at the welding-heat is a number of times as great as that of the cold metal.*

The temperature-resistance coefficient is, in general, much higher in the case of simple metals than of metallic alloys. The conductivity for heat has also a decided influence on the heating, and the comparatively low heat-conduction of iron assists the work materially. Brass of ordinary composition scarcely seems to become plastic at all before melting. Hence the joints made with it generally show, in part, a real fusion and not merely a softening. In cases where the pressure used is too great, the joint is obtained by the crushing up of the hot metal in a semi-granular state before fusion, but this joint is not generally so perfect as the other. Some of the bronzes, and notably the alloy called aluminum-bronze, show some plasticity before fusion.

The behavior of copper in welding by the electric current may be noted. It is an example of a metal of very high conductivity for heat and for current. Naturally, to weld it electrically, the current-strength required will be much greater for a given section than for metals of smaller conductivity. The heat produced escapes rapidly by conduction, and the low resistance of the metal requires a great current to be passed to cause it to heat. This is compensated for, in large part, by the fact that the copper requires a very low potential, or E. M. F.—less than a volt across the joint. The rate of energy-consumption being the product of the current used by its electro-motive force, and the total energy consumed being the product of this by the time, it will be seen that if a copper weld be made, in a short time the losses by conduction and radiation from the joint can be kept down to a moderate percentage of the total energy. Copper, as is well known, becomes decidedly softened before melting, and is, therefore, often forged hot. Hence, joints may be made apparently without real fusion; or, in other cases, when a lighter pressure for forcing the pieces together is used, a real fusion of portions of the meeting ends may occur. While in the electrical welding process joints are obtainable with most metals without the use of any flux, it is not the less true that a flux is sometimes desirable for good work, particularly with those metals the oxides of which do not melt at the temperature of welding or union of the pieces. Thus a flux should be used with brass, which has a temperature of welding

* Reference is made to the curve in Sir F. Bramwell's paper in connection with which the effect of increase of resistance by temperature in assisting the work is very thoroughly and ably discussed.

and union below that of the fusing-points of oxide of zinc or of copper, while a flux is not an essential in the welding of copper, the melting-point of which is above that of its oxide. In like manner, a temperature which will melt the oxide of iron on the surface of a bar is below that of the welding of soft iron, while such a temperature would ruin tool steel. The latter requires, therefore, a flux to be used with it, melting at a safe temperature and dissolving the scale. The covering action of a flux is also useful in most cases to prevent oxidation. A peculiarity of the electric-welding process, and one which renders it capable of forming joints, even where metals may be coated with oxides infusible at the temperature of fusion of the metals themselves, is the expulsive power exerted to remove metal sidewise from the joint, and thus bring unoxidized, clean surfaces into contact at fusing-temperatures. It is this fact which more than any other accounts for the universal applicability of the process to metals and alloys. For example, the chief obstacle to the soldering or welding of aluminum may be the skin of alumina formed on its surface when hot, which not only protects the metal from further oxidation, but prevents its union with other pieces of metal. The alumina is fusible only at the highest temperatures, whereas the metal fuses at about a red heat. Let two pieces, however, be vigorously pushed together while softening at the joint; then the oxidized surfaces will be thrust aside outwardly from the joint, and the new metal of the interior of the pieces will come together in the cleanest possible state, and unite. The removal of the burr or excluded metal of the joint can be afterward effected with proper tools.

The uniting of iron and various grades of steel is, of course, the most important industrial use of any welding process, and this is particularly true of an electrical one. The behavior of soft iron is particularly favorable to the welding operation. Its great plasticity before fusion renders its working and welding easy and simple. In the case of steel, for the higher grades of which, especially, the ordinary blacksmithing operation is liable to be very uncertain, the electric process is rendered easy and certain for all grades. When it is remembered that the exact conditions of current, conduction, and pressure at the joint may be made definite, and are perfectly under control, it is seen at once that, if the pieces are weldable at all, we can even establish an automatic control of the welding after these conditions have been once determined.

Electric welding machinery is now constructed in which these

features are incorporated, and which is, at the same time, provided with appliances for shaping or hammering the joint, so as to preserve the diameter and increase the toughness of the metal thereat. Where the heat of the welding tends to change the structure or cause a tendency to granulate, hammering may be applied to retain the structure or to renew it during or after the union of the metal process.

There are many kinds of work to which electric welding seems to be peculiarly applicable. It is now in use largely for the joining of sections of wire into one length, and its use for such a purpose is still extending. Joints are regularly made in wires, which are afterward reduced in diameter by the wire-drawing process, various metals, such as copper, iron, brass, etc., being treated in this way. In other cases the electric-welding process replaces the twisted joint formerly made in galvanized telegraph and similar wires—the galvanizing operation either preceding or succeeding the making of such joints. Even the ends of wire cables are readily jointed, the resulting weld possessing a strength not much inferior to the cable itself, and probably about equal to that of any heated or annealed portion. The making of such joints requires, of course, special precautions, but the operation is quite simple otherwise. In the welding and coiling of pipe the electric method seems to be especially expeditious and useful. Coils of pipe are now made continuously, as section after section of pipe is welded on. There is no limitation to the length of pipe which may be so welded and wound as one coil.

There is neither time nor space to go into full descriptions of the details of each operation. The author must limit this paper to brief statements of what has been done in the development of this field of work. The process seems destined to work a revolution in pipe coiling operations. A machine will soon be tested which is intended for welding sections of six-inch extra heavy pipe together endwise. The current developed in this machine may reach 70,000 amperes, though its normal work will demand somewhat less than this. Most of the details of this and the other welding apparatus thus far constructed and put to use have been devised and arranged by Mr. Hermann Lemp, to whom the greatest credit is due for the original character and effectiveness of the machinery itself. Lieutenant W. B. Wood has also had in charge a number of special pieces of work in adapting the welding process to several important applications.

Somewhat akin to pipe-welding is the making of armor-piercing shells (as devised by Mr. Wood) from sections of varied grades of

steel and without screw-joints. The sections are made separately, such as a head or point of high grade steel, a tubular middle section of softer, and a soft back piece perforated for loading; or the shell may be made of two sections only, jointed near the middle. These sections are placed in a welding apparatus of special and most interesting design, where they are welded together to complete the shell body, which is afterward finished outside. Experiments in firing shells made in this way have given satisfactory results, and further developments are being made as rapidly as possible. This, with much other work of like nature, is in the care of Lieutenant Wood. In the making of rings, links, bands, etc., the electric welding operation is particularly useful, and a number of machines specially designed for such work have been put into practical operation. Even rings of small diameters relatively to the weight of section of stock used, are made with facility, though at first it might be thought that the current would pass around the joint and through the solid metal of the ring, instead of the joint or cut ends pressed together. This is true in part; but the actual result is that only in very stout rings is there much heating outside of the joint, and then only enough to make the ring flexible, so that the ends of metal at the joint may be readily moved together. With the alternating currents used in electric welding, not only does the current take the path of least resistance in largest amount, but it also seeks the short path, or the path of least counter-induction or self-induction, and, therefore, will not go around a ring as freely, other things being equal, as along a short, straight path through the joint at one side. Furthermore, if it be desired to check any flow around the ring still more effectively, we have only to introduce into the center of the ring or link a magnetizable core of iron, which effects a great increase of the counter-induction mentioned.

An important matter which may be mentioned in this connection is the facility with which the milder grades of steel may be manipulated in the welding by electricity. The advantage of the superior strength of steel over puddled iron may thus be secured, which allows the use of a cheaper material, and less of it for a given strength. This remark applies to chain-making, in which the substitution of low steels for iron secures the double advantage mentioned, with the other advantages of the electric process over the ordinary ways of working. Considerable attention has, therefore, been given to the development of machinery for chain work, and a small working apparatus has even been made, which takes a wire

from a reel and turns out lengths of chain with electrically welded links; the operation being automatic throughout. A curious feature of electric-welding work is that two or more welds may be simultaneously made in parallel. This fact is utilized in chain-making, where two welds, one at each side of a link, are made simultaneously. There are not yet in industrial operation any chain-machines using the electric process. This step, however, is reserved for the near future. A special, heavy chain-machine will soon be completed.

Thus far the field of work in the manufacture of tools and parts of machines by welding different forms and different qualities of metal together has been but slightly explored, and principally in an experimental way; but it is believed that eventually electric welding will find a large application in such directions.

The size of work which can be done by the electric method is limited only by the size of apparatus and the power to generate current for it. Unquestionably there are natural limits to its applicability and economy, which remain to be determined. It is not to be supposed that this process can or will supplant entirely the other methods of welding, any more than electric lighting supplants gas-, oil-, or candle-light entirely. That it already has enlarged, and will continue to enlarge, the field of mechanical possibility cannot be doubted, in view of the very considerable developments which have taken place in the past two or three years, embracing only the infancy of the art.

In the comparisons between the strengths of electrically welded stock and the results of ordinary welding by the blacksmith, given in Sir F. Bramwell's paper, referred to above, the electric process has shown a somewhat higher percentage of strength, notwithstanding the fact that, in making up the averages, the specimens which broke outside the weld were, in the tests, counted out, and only those which broke in the weld were taken. If the specimens which broke elsewhere than in the weld are counted at 100 per cent. strength, the actual average strength obtained is between 95 and 100 per cent. In similar comparisons as to bending, it was found that the electric welds would not stand bending as well as the others, but when it was remembered that the electric welds were butt-welds, and that in making them a comparatively very short section of the metal at each side of the joint was heated and annealed, while the ordinary welds were lap-welds and involved the heating and softening of a considerable length of the bar, it was seen that the bending in the former case through a given angle would virtually be greater than in

the other case where more of the length took part in the flexure. In such comparisons, as afterward pointed out by Sir F. Bramwell himself, each specimen should be softened by heat to an equal length and afterwards bent under like conditions; otherwise no conclusion can be drawn from bending tests.

It will be readily understood that the heating effect of very heavy electric currents may be applied to other cases of metal-working than welding. Thus, it may be used to heat pieces for soldering or brazing, an operation which only differs from welding in the use of a more fusible metal between the surfaces to be united. In many cases the advantages of control of temperature, quickness of heating, and convenience of holding the pieces will be found greatly in favor of the electric method. In fact, an electric brazing machine has been regularly in use for some time past for bicycle construction, while in other instances such welding takes the place of joints formerly made by brazing with the blow pipe. The application of the electric methods to many tin-plate soldering operations is likely to become industrially, as it has been already experimentally successful.

Again, the use of electric currents of large volume and low potential for locally heating metal to plasticity for shaping, swaging, rolling, spinning, or otherwise giving form to it, seems destined to be applied on a large scale in the future. The author has found it easy to apply the heating currents to metal pieces and to give them rotation or other movement in shaping. As an example may be mentioned the heating of portions or zones of pipe and the revolutions of the pipe itself, or of the tools around it, to raise locally beads or expansions, or form constrictions or depressions, to close or expand the end of the pipe, to cut it off, to spin it into various forms, such as hollow shells or vessels, in fact, to deal with it without cutting tools, at the same time that its temperature may be kept up to plasticity by the electric current. The end of a pipe may be heated, rolled to contract or expand it, and a thread rolled on it, either internally or externally, if desired, at the same time. These operations may be performed while the work is surrounded by hydrocarbon gas or vapor, which prevents scaling of the surfaces. Trials of work of this kind point to future application in various ways.

Among the operations which can be readily performed electrically is that of heating a rivet in place by a current passed through it and heading it while hot. In many trials it has been proved that short, straight sections may be used as rivets, and both heads formed

simultaneously, either as projecting heads or as countersunk heads. The heating action may be so quick as actually to heat the rivet without communicating heat to the plates; or it may be slower, and the plates themselves may partake, in part, of the welding action, so that the rivet welds to the plates. Naturally the current for this work is large, but the potential required is low. The degree of applicability of the process in practice will, it appears, depend on the conditions and requirements in each case. The operation is evidently only a modified upsetting process, in which the section of metal between the clamps or current-conveying blocks is heated. It is, of course, not essential to this operation that the whole length of the rivet be heated, or that both heads be formed at the same time. In fact, the projecting end of a rivet may receive contact from the heading-block as one terminal of the circuit, and the return or opposite terminal may be the plates to be riveted, and this would be the preferable arrangement where the rivet-body was long and the plates to be riveted together were thick. The author has, in addition, devised new riveting operations specially adaptable to the electric process.

Before closing this brief review of some of the phases of electric metal-work, reference may be had to the subject of economy in the work done, and to the power required. In the first place, the conditions vary so greatly that no general statement can be made; but when it is remembered that, for making steam, waste-gases from operations requiring high temperatures are still hot enough, and that the nature of the electric current enables high temperatures to be secured from power developed from steam at comparatively low temperatures, it is easily seen that in many cases of heavy metal-working, power for welding is already at hand, just as power for hammering in heavy forging is obtained, in many cases, from the waste gases of forges. Again, by the electric process the large water-powers can be turned to account in metal-working, and places where fuel is dear and water-power plentiful may, in future, take positions of importance with respect to certain varieties of such work.

Furthermore, where steam is generated for the express purpose of welding and working metals by electricity, the localization of the heat to the exact point required, as compared with heating a considerable length or mass in the fire, with the loss of heat involved in so doing, may offset the heavy losses in the generation of power from steam. It is readily seen that the economy of heating to high temperatures in the fire must fall as the temperature of the metal rises,

the temperature-difference between that which gives heat and that which receives it becoming less and less. In the electric process the reverse is true, aside from increased loss by radiation and conduction as the temperature rises. During heating, the increasing temperature of the metal adds greatly, in most cases, to its electrical resistance, and this in turn increases the electrical economy of the plant, or increases the proportion of the electric energy converted into heat in the work itself.

Considered in relation to the automatic character, the range of adaptability, the convenience and cleanly nature of the work, the perfect control of temperature secured, and the uniformity of result, the electric-welding and metal-working processes are decided advances over the earlier art, and seem destined to a wide application. Already they have given rise to new methods of construction and have produced new articles of manufacture, and they will doubtless advance continually in this respect, in addition to their application to old constructions, or to the replacing of old processes in established manufactures.

Lack of time and pressure of other work have prevented the author from presenting more than a sketch of the developments of electricity in metal-working, and a consideration of many interesting details has been necessarily omitted.

Discussion.

Ewing Matheson, Leeds, England: I have spent some little time in investigating the machines which Professor Thomson sent to London, but indeed no great time is required to convince any one that the electric welding process is practical. Mechanical engineers have been particularly interested with the facility it affords for actual mechanical operations which are called for every day in our factories. Unfortunately, the machines sent to London were intended only for exhibition purposes, and hence the clamps and subsidiary parts were not devised for any particular branch of work. In a recent visit to the electric-welding factory in Boston, at the instance of Professor Thomson's kind invitation to members of this Institute, my time was devoted to observing, not the mere practicability of electric welding, but rather the means for adapting the welding-machines to particular branches of trade. I was very much struck while there with the peculiar appliances for the different shapes of work that we require in every-day operations. Just as in a lathe or planing-machine it is not the time alone that is taken

in the actual cutting, but that consumed in mounting the pieces which counts, so it is with this machine, and some remarkably ingenious devices have been made through which, by an almost automatic operation, pieces of various shapes can be inserted in the machine, welded, and taken out in a marvellously short time—and this not only for the plain butt-welding but also for lap-welding. Perhaps the most ingenious of all the machines was the one for welding the hollow shells which Professor Thomson has shown us in the lantern-views. The soft-steel tube and the hardened steel point were put together not only quickly but perfectly symmetrically—that is to say, the concentricity of the two was not at all lost. If any of the American gentlemen who are present can enlighten us on the commercial application of the machine in this country, such information would prove interesting and valuable. In England I believe there is as yet but one engineering works where the machines are in operation every day in the smithy, but their introduction will no doubt be rapid.

P. C. Gilchrist, London, England: From the very clear statement we have had it is evident that the difficulty in welding almost every metal had been completely overcome, more especially the difficulty of welding different kinds of steel. There are one or two uses to which steel is not yet applied as much as it ought to be —notably to the manufacture of chains. If the cost of making the electric welds in chains is not too high, there can be no doubt that such a method of welding will lead to a very great increase in the consumption of steel for chain-making. While it is against our rules to ask the cost, it is not against our rules to ask for pretty nearly the same thing; if, therefore, the author is disposed to give us the actual horse-power required for making up so many hundredweights of chains of particular sizes, we shall be able to determine the cost quite closely. The use of the new process will depend entirely upon cost. A knowledge of the power required to perform certain specific work will enable us to form a fair idea whether the process may be profitably applied in our works. As one among a good many members to whom the subject is new I venture, in their behalf as well as my own, to tender our thanks to the author for his lucid exposition on the subject.

Elihu Thomson, Lynn, Mass.: The actual horse-power consumed depends upon whether the weld is made quickly or slowly, and therefore it varies greatly. For matters of this kind I can simply refer to the trustworthy facts and figures of Sir Frederick

Bramwell's paper, which is well up to date. He dwells, also, on other interesting matters, such as the comparative fuel-consumption of electric-welding and blacksmith-welding. He finds that the actual fuel consumed in each case is about the same, although the electric welding was not completed without two operations—that is, the hammering was not finished during the first heat. If the hammering had been finished by automatic means during the first heat, the consumption of fuel would have been less in the electric-welding than in blacksmith-welding.

THE WEAR OF METAL AS INFLUENCED BY ITS CHEMICAL AND PHYSICAL PROPERTIES.

BY C. B. DUDLEY, PH.D., ALTOONA, PA.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

In October, 1878, and again in February, 1881, I had the honor to make public, through the medium of the American Institute of Mining Engineers, the results of an extended study of steel rails which had been in service, and which were taken for the purpose from the tracks of the Pennsylvania Railroad Company. These studies appeared in three papers. In the first of these papers the question, what kind of steel is least liable to fracture or disintegration in the track, was the principal one considered. In the second paper the question discussed was, does the power of steel to resist wear increase with the hardness? In the third paper the relation between wear and the chemical and physical properties of the metal was the principal point considered. The general results arrived at, or the conclusions reached, were, as follows: 1. That a mild steel is less liable to fracture, and if properly made, less liable to crushing or disintegration in the track than a harder steel. 2. That the wearing-power of steel in rails not only does not increase as hardness increases, but, on the contrary, diminishes; or, in other words, that a mild steel gives less loss of metal under the same service than a hard steel. These conclusions may be briefly stated in a single sentence—namely, in rails mild steel is not only less liable to fracture and disintegration in the track, but it also gives longer life, or wears more slowly under the same traffic, than harder steel. It is perhaps hardly necessary to add that by mild steel is meant steel

with smaller percentages of carbon, phosphorus, silicon and manganese than is characteristic of harder steel.

My own criticism of this work, after the lapse of ten years, and after all the discussion which followed the publication of the papers above-mentioned, may, perhaps, be fairly summed up in four conclusions:

1. If I had the work to do again I would certainly determine the sulphur in the rails, since all our studies during the past ten years on the influence of sulphur point strongly in the direction of indicating that the sulphur has an important influence on steel, especially in its effect on the carbon.

2. The influence of silicon, and especially its influence from the metallurgical standpoint, seems to be much better understood now than at the time when these studies were begun, and if an ideal formula, representing our views as to the best possible composition for steel rails, was to be made at the present time, the silicon limit would be raised somewhat, possibly to the favorite figure of Mr. Sandberg—namely, 0.10 per cent.

3. It is possible that in the first paper published the influence of the chemical composition on what is commonly known as crushing or disintegration of rails in the track was made more prominent than the facts would warrant. More mature or riper studies would seem to indicate that disintegration or crushing of steel is largely a resultant of lack of soundness in the ingot, and is more mechanical than chemical, except in so far as chemistry may be responsible for the soundness of the ingot. However, upon this point of sound ingots, we have seen little reason to modify the views held for some time, that time is more important in securing sound ingots, especially time in certain critical parts of the process, than any other single element. If our views are correct, sound ingots, with consequently sound rails, can be made from steel of varying composition, provided time is allowed at the right points in the process, and the claim that high manganese and high carbon are essential to secure sound ingots is, in our judgment, not well founded, provided that time enough is allowed to make the steel properly.

4. In all our later studies on the wear of metal we have, as far as possible, avoided a method of deciding which metal is best which attempts to give what may be called absolute results. In other words, loss of metal by wear, per million tons, has some necessary errors in it, and, accordingly, in our later studies we have adopted the method of comparison: Two metals of different composition

are subjected as nearly as possible to the same wear, and the one which wears the faster, by comparison with the other, is regarded as the poorer, as will be explained a little later. It would, perhaps, have been difficult to use this method on rails selected from different portions of the track, the extremes being somewhere about 100 miles apart, but if we had the work to do again a strenuous effort would be made to use the method of comparison. Of course, this is possible where direct positive experiments are made.

One point further in regard to the work already done on this subject. Samples of all the rails which were discussed in the third paper, above mentioned, have been carefully preserved with the idea in mind of some time repeating the work, with the improved means and methods which time and study in the realm of chemical metallurgy should place in our hands. Especially has it been hoped that some methods of chemical analysis would be devised which would enable us to determine, not only the total amount of carbon, phosphorus, silicon, manganese, sulphur, etc., in these various steels, but also how these substances were combined. For example, does the phosphorus in any given rail exist as phosphide, or partly as phosphate, or both? Is the silicon simply alloyed, or chemically combined with the iron? Does the carbon all appear as strength-carbon, or is some of it graphitic, or combined with the iron in such a way as to form a crystalline body which adds nothing to the strength? The ten past years have done something in unravelling these mysteries, but hardly enough, we think, has been developed as yet to enable us wisely to re-examine these rails. It has been our belief for some time that no decided step forward in the chemical metallurgy of steel could be taken until we have solved some of the questions as to how the various impurities which occur in steel exist in the metal, and we may add that it is not from lack of interest in the subject, but from necessary devotion to study in other lines, that our own work in this field, and our possible contributions to knowledge on this subject, have been so small.

Otherwise than as regards the criticisms above mentioned—namely, that the sulphur should have been determined, possibly the silicon limit raised a little, the influence of the method of manufacture on the final product made a little more prominent, and the comparative method used as far as possible in determining the difference between good and poor rails—we have seen little occasion to modify the conclusions stated in the papers as published; that is to say, during ten years past, with our minds constantly directed to the

question of what kind of steel will give the greatest safety and the least wear, not only in rails, but also in almost all constructive uses, we have seen no occasion to change the main conclusion previously reached—namely, that mild steel is not only safer for rails and for other constructive purposes, but also that mild steel gives better wear, or loses less metal under the same traffic, than harder steel. It seems quite probable that this conclusion will hardly be accepted by the mass of engineers, or by those who are engaged in metallurgical industry, and the object of this paper is simply to bring up to date the additional information which has been accumulated on this subject during the past ten years. It is perhaps unfortunate that a larger number of direct experiments have not been made during that time; but progress in such a study as this must necessarily be slow, and methods of making experiments which will give unquestioned results do not always readily present themselves, so that perhaps in some respects the delay has not been so great a misfortune. We are much better prepared to-day to make positive experiments than ever we were before, and we have experiments already started which we hope will throw much light on this subject. We hope we may have the good fortune to bring the results of these experiments to your attention at a later date. Meanwhile, what light have the past ten years thrown on the special subject of the relation between wear and the chemical and physical properties of metal?

1. Since the papers above referred to were published no systematic study of steel rails has been attempted in the Pennsylvania Railroad Laboratory. A few tons each of rails of different specified composition were obtained and put in the track, with the idea of making a positive experiment on wear. Unfortunately, owing to changes of alignment, and in the *personnel* of those who had charge of the experiments, these rails were lost sight of, and no positive conclusions can now be drawn from them. General observation has, however, been constantly maintained, and an occasional analysis of a remarkable rail taken from the track has been made, although these have not at all altered the conclusion previously expressed.

2. It will be remembered by those who are familiar with the discussion which followed the publication of the papers on steel rails, that some trial rails, according to the formula suggested in the first paper, were made in Germany, and inspected by Mr. Sandberg, who made his work the subject of a contribution to the discussion. The place where these rails went into service never came to my knowl-

edge until a little over a year ago. At that time I received a letter from Mr. P. H. Conradson, the chemist of the New York and New England Railroad, stating that he had found in the records of that road that 2500 tons of steel rails, in accordance with the formula suggested in the first steel-rail paper above referred to, had been made at Gutehoffnungshütte, Germany, and had been laid on their road. Mr. Conradson's letter gave the formula on which the rails were ordered, which is practically the formula of my first steel-rail paper, and also states that the records show a number of reports of inspection, signed by Mr. Sandberg, with analyses made by Mr. Magnus Troilius. A number of analyses are also given, taken from the tests of inspection, showing, in general, that the rails came within the limits of the specifications—namely, in no case was carbon above 0.35 or below 0.25 per cent.; manganese was in no case above 0.40 or below 0.30 per cent.; phosphorus was between 0.055 and 0.075 per cent., and silicon varied from 0.01 to 0.08 per cent.; most of them being about 0.05 per cent. The physical tests were also fairly close to the assigned limits of the specifications—namely, they were generally within 75,000 pounds tensile strength per square inch, with not less than 20 per cent. elongation, the limits being from 73,000 to about 80,000 pounds tensile strength per square inch, and 17 to 23 per cent. elongation. The rails weighed 60 pounds per yard, and were put in the track where they could be subjected to the severest service. They remained at this point seven years, and were taken up to be replaced with a heavier rail. Upon receipt of Mr. Conradson's letter, the matter was brought forward with the officers of the road, with the idea in mind of obtaining positive figures as to the tonnage which had passed over the rails during the seven years, but although data on this point were promised, yet, owing to changes in the management, they were never furnished. We are, therefore, without positive data, expressed in figures, as to how these rails wore. The statements, however, of those most familiar with the rails while they were performing their service were to the effect that they were in every sense satisfactory, and that they gave excellent results. The words of the one most competent to judge, quoted by Mr. Conradson, are as follows: "These rails have not changed any during these seven years of continuous use so that it can be detected by the naked eye." It would be gratifying to have positive figures as to the tonnage and loss of metal by wear, but in the absence of these data it is perhaps not too much to say that even if we do not have from it so positive a con-

firmation of our views as could be desired, there is certainly nothing in the facts to controvert the idea that mild steel gives better wear in rails than hard steel.

3. I am aware of the conclusions reached by M. Verschovsky, engineer-in-chief of the Russian State Railways, which were stated in a paper read before the Railway Congress in Paris in 1889. I have not seen the original paper, but simply a translation. Those who have read this paper know that the conclusions reached by M. Verschovsky seem to be directly opposite to the conclusions which we have reached—namely (if we can trust the translation which I have), "the best-wearing rails have the greatest tensile strength with the least elongation;" and again, that the "rails which broke were softer, as far as indicated by tensile strength and elongation, and there is a difference between a hard rail and a brittle one." Still further, as regards the chemical composition, "the best rails contain more carbon and manganese than the brittle ones, and in all cases much more silicon and less phosphorus." In the absence of data as to what constituted the best rails—that is, as to how the best rails were decided on—and in the absence of any positive data of loss of metal by wear, it is, of course, exceedingly difficult to criticise or explain the conclusions obtained by M. Verschovsky. However, one point in the paper seems to throw some light on the possibilities of the case. It is stated that, in consequence of the specifications issued by the government, the steel-works did all in their power to produce a soft steel, so as to insure that the "frozen rails should stand the falling- or drop-test prescribed," and that this result was accomplished, but that, as a concomitant of this soft steel, a new difficulty appeared, which was that after a few months of wear the rails began to crush or flatten at the ends, so that in a short time replacement was found necessary. To our minds this seems to indicate that as the result of the effort of the steel-works to produce soft steel, very unsound or porous ingots were obtained, which produced rails that were not sound and homogeneous, and which would be likely to behave exactly as M. Verschovsky describes. It will be remembered that this difficulty of getting sound ingots with soft steel was largely the burden of the discussion which followed the publication of the steel-rail papers, and it seems fairly probable that the difficulty which M. Verschovsky describes may be entirely due to this cause. If the steel-works, in trying to make the soft steel, so hurried the process that unsound and porous ingots were obtained, we see no difficulty in accounting for the conclusions which M. Verschov-

sky has reached. We ourselves have seen and analyzed steel which was not only brittle, but which also readily disintegrated under blows, and gave both low tensile strength and low elongation, while on analysis it was almost ideal in its proportions of carbon, manganese, phosphorus and silicon to make good rails. The difficulty with the steel was that it was improperly made. The sample which we have in mind was a special blow, made for a special purpose; its history was known completely, and we are confident that few practical metallurgists will disagree with the conclusion that if, at the end of the blow, sufficient time is not allowed to elapse to secure deoxidation, and if the metal is cast while it is still rapidly boiling, a steel will be produced which will be not only brittle, but also of low tensile strength, low elongation, and in every sense unreliable. We are free to confess that we would much prefer a hard, sound steel to a soft, unsound one. Our plea has always been for a sound, homogeneous, mild steel. Of course it is impossible for us to say that the above was exactly the case with the rails tested by M. Verschovsky, but the facts seem to us to point to a possible explanation of the conclusions which he has reached, and if our explanation is correct we should have very little difficulty in agreeing with his conclusions, our answer being, however, that a mild steel properly made is much better than the hard steel which he praises.

4. Several years ago our attention was directed to the question of tires on locomotive driving-wheels, which, as is well known, are made of steel. The question under discussion was the possibility of preparing specifications for locomotive-tires, and, as a preliminary study, examination was made of a number of steel tires which had been in service. It may be stated here that for a number of years past the practical men in charge of the lathe-shops on different portions of the Pennsylvania Railroad where tires are turned off noticed, and have stated in conversation, that they always had to turn off the most metal from the softest tire; that is to say, when the tires come into the shop for re-turning the very hard tires were the ones which had worn the most, the hardness being determined by the behavior of the tool during turning. This, of course, was simply an observation, and very few positive data could be drawn from it. Following this suggestion, however, a number of tires were measured, the method being to take the circumference of two tires on opposite ends of the same axle. These tires, as is well known, are of practically the same size when they go into service, the variation being possibly not more than $\frac{1}{100}$ or $\frac{2}{100}$ inch in diameter between

the two tires on opposite ends of the same axle. They also, as is well known, make the same number of turns, doing the same amount of service, and, barring the possibility of the engine getting a little out of true, have practically the same pressure between the tire and the rail during their service. Again, if the driver-brake comes into account (all driving-wheels do not have driver-brakes), its action is practically the same on the two tires on opposite ends of the same axle, so that it would seem that locomotive-drivers afford an admirable opportunity for the use of the comparative method of making tests on the relative wear of hard and soft steel. In view of this consideration, a number of tires were taped as they came in from service to be re-turned, and in a short time three pairs of tires were found each of which showed a marked difference between the two circumferences. The circumference was measured by putting a tape around the tire. In one case the difference in circumference was 2 inches, and in each of the other cases the difference was $1\frac{3}{4}$ inches. This difference in circumference corresponds to a difference in diameter of from 0.55 to 0.63 inch, or from 0.27 to 0.31 inch in the thickness of the tire itself. In other words, by actual measurement of tires which had been in service on opposite ends of the same axle, which were of the same diameter and circumference when they went into service, in the case of two of the pairs, one of the two tires had lost a little over $\frac{1}{4}$ inch of its thickness more than the other, while in the other pair one had lost nearly $\frac{1}{3}$ inch more of its thickness than the other. In view of this discrepancy in wear under as nearly as possible the same conditions, it was with a great deal of interest that analyses were made of the metal taken from these tires. The results of the analyses are as follows:

Analyses of Unequally Worn Tires, from Opposite Ends of the Same Axles, on the Pennsylvania Railroad Locomotives.

Engine 654.	Least worn tire. Per cent.	Most worn tire. Per cent.
Carbon,	0.594	0.708
Manganese,	1.076	0.938
Phosphorus,	0.039	0.101
Silicon,	0.245	0.143

Engine 136.	Least worn tire. Per cent.	Most worn tire. Per cent.
Carbon,	0.541	0.625
Manganese,	0.880	0.974
Phosphorus,	0.062	0.063
Silicon,	0 253	0.153

Engine 626.	Least worn tire Per cent.	Most worn tire. Per cent.
Carbon,	0.525	0.554
Manganese,	0.512	0.714
Phosphorus,	0.032	0.037
Silicon,	0.169	0.208

The tires of Engine 654 had a difference in circumference of 2 inches; the other two had a difference in circumference of $1\frac{3}{4}$ inches. It is interesting to observe that in every case the carbon is lowest in the least worn tire, indicating the softest steel, so far as carbon is concerned. Again, in two of the three, the manganese is lowest in the least worn tire. In one of the three, there is quite a difference in phosphorus, the lowest being characteristic of the least worn tire, the other two having very slight differences in phosphorus. In two the silicon is highest in the least worn tire, while in the other the difference is the other way. Of course it is impossible to draw any very general conclusion from so small a number of samples as three, but the teaching of these results would seem to be that, in general, lower carbon and manganese and higher silicon are characteristic of tires which give the best wear. It is fair to say, in this connection, that it is not at all impossible that the temper of this steel may have an influence on the wear, apart from the chemical composition. It is well known that tires are "set," as it is called—that is, are fastened upon the wheel-center—by having the tire bored out to a diameter a little less than that of the wheel-center, and then heating the tire to expand it, so that it will take the wheel-center, and then cooling the tire. If, now, the tires are not heated to uniform temperatures, and cooled uniformly, or if they differ somewhat in their carbon, it is probable that one tire would have a different "temper" or "hardness produced by cooling" than the other, and this might have an influence on the rate of wear. Of course no positive data can be given on this point at the present moment. Before leaving the wear of tires, permit me one word further. Confining ourselves to the question of wear, it is difficult to conceive of anything which produces wear in rails different from that which produces wear in tires, since, if we understand rightly, it is a strain between the wheel and the rail which causes the rupture or tearing off of the small particles which we are accustomed to call "wear." It would seem, therefore, that tires afford an admirable opportunity for studying the question of relative wearing-power of hard and soft steel, and the experiments alluded to in the early part of this paper are the experiments on the wear of tires. A method has been devised, we

think, and the number of hard and soft tires compared is sufficiently great, it is hoped, to enable very positive information to be obtained as to the relative power of hard and soft steel to resist wear. It would hardly be wise to enter into the details of the method of these experiments at the present moment, but it seems probable that within six months, or possibly a year, we may have considerably more light on the relative wearing-power of hard and soft steel than we now have.

5. Our studies on wear have not been confined wholly to the behavior of iron and steel under abrasion or rolling-friction. A very large number of experiments have been made on the Pennsylvania Railroad with various alloys used as bearing-metals, and instructive information may be drawn, we think, from these experiments as to the relation between wear and the chemical and physical properties of metal. If the experiments just alluded to may be trusted, and our deductions from them are correct, we find the same thing in the realm of the alloys which we have heretofore claimed with regard to iron and steel,—namely, that those alloys which are least brittle, or, for want of a better word, those alloys which may be called softest give the best wear,—that is, they lose less metal under the same conditions than harder or more brittle alloys. Or, looked at in the light of definite physical properties, those alloys which give the slower wear are characterized by lower tensile strength and greater elongation than is characteristic of those giving the more rapid wear.

The alloys experimented with have been principally the old copper-tin alloy (seven parts copper to one of tin) and alloys of copper, tin and lead, with and without phosphorus and arsenic. It is not intended to say that no other alloys have been experimented with, but these are the alloys which have been most experimented with, and in regard to which we have the most reliable data. The method of experimentation has been to have a certain number of bearings made of a standard bearing-metal, which will be described later on, and the same number of bearings of the experimental metal. These bearings were placed on opposite ends of the same axles, either on locomotive-tenders or on cars, one-half of the standard and experimental bearings being on one side of the car and the other half being on the other side, but in all cases a standard bearing and an experimental bearing on opposite ends of the same axle. The bearings were all carefully weighed before going into service, and after a sufficient lapse of time were taken out and re-weighed. At first an

attempt was made to give the loss of metal by referring it to the mileage, but the method of comparison was ultimately adopted, as giving results free from any possible difficulties introduced by mileage, so that all the results which we obtained are strictly comparative.

The standard bearing-metal is what is known in the market as phosphor-bronze bearing-metal, technically described by the Phosphor-Bronze Smelting Company as the "S Bearing-Metal." This metal contains approximately 79.70 per cent. of copper, 10 per cent. of tin, 9.50 per cent. of lead, and about 0.80 per cent. of phosphorus. It is, of course, a fair question, and one which has not been overlooked, whether the standard bearing-metal gives uniform wear. A large number of experiments have been made on this point, the result being that the average wear of standard phosphor-bronze, compared with the mileage, is best expressed by saying that the phosphor-bronze bearing-metal loses one pound of metal, worn off, for every 18,000 to 24,000 miles of travel. This, it will be observed, shows a discrepancy in the wear of the standard phosphor-bronze bearing-metal, and the fact led us to abandon the method of making comparisons by mileage. The reasons for the discrepancy are not hard to find: First, the pressure per square inch in all tests was not the same, and consequently the wear would not be the same. On the other hand, with bearings on the opposite ends of the same axle, the pressures per square inch are approximately the same. Second, the state of the lubrication in different cars and engines, which is more or less characteristic of different parts of the road, is a very important variable, and undoubtedly goes far towards explaining the differences in mileage above given. This variation in the state of lubrication is not so apt to be characteristic of opposite ends of the same axle as it is of different cars and locomotives. We are inclined to think, therefore, that the assumption that standard phosphor-bronze is sufficiently uniform in its behavior to warrant its being used as the basis of comparison will not lead us into serious error, at least if we confine ourselves to a direct comparison of the loss of metal obtained from standard bearings on one end of the axles and experimental bearings on the other end of the same axles. Usually sixteen bearings of each kind were put in service as a preliminary experiment, and if the metal proved at all favorable on this preliminary trial, a larger trial, embracing fifty or one hundred bearings of each kind, was put in service. The preliminary trials were usually made on locomotive-tenders, where the bearings get the best possible care.

The larger trials were more commonly made on cars. Of course, owing to the exigencies of the service, it sometimes happened that some of the bearings put in use were not returned to be weighed. This was more true where bearings became heated, and were removed at different points along the line, than at the regular inspection points. Whenever, from any cause, a bearing was missing, its opposite was not taken into account, so that in the results given the comparisons are strictly between the same number of standard and experimental bearings on opposite ends of the same axles. Sometimes as high as one-half the bearings in an experimental lot would be lost. In other cases, nine-tenths would be returned. The results of the tests, with the composition, and, so far as our knowledge goes, the physical properties of the various alloys tested, are given below. It is unfortunate that in the earlier tests the physical properties of the alloys were not taken. In the composition, approximately average analyses are given rather than a special analysis of the metal in each test, and it will be observed that there is no allowance made for the small impurities, such as zinc, antimony, iron, etc., which are usually characteristic of the commercial metals used in making these alloys, especially where some scrap is used in making the bearings, as is almost always the case. It will also be observed that in all cases, in expressing the loss of metal by wear, the results are given in percentages of the metal lost by the standard phosphor-bronze :

Copper-Tin versus Phosphor-Bronze.

	Composition, copper-tin. Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	87.50	79.70
Tin,	12.50	10.00
Lead,	none.	9.50
Phosphorus,	none.	0.80

Wear.—First experiment, copper-tin wore 48 per cent. faster than phosphor-bronze; second experiment, copper-tin wore 53 per cent. faster than phosphor-bronze; third experiment, copper-tin wore 47 per cent. faster than phosphor-bronze.

Arsenic-Bronze versus Phosphor-Bronze—First Experiment.

	Composition, arsenic-bronze. Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	89 20	79.70
Tin,	10.00	10.00
Lead,	none.	9.50
Phosphorus,	none.	0.80
Arsenic,	0.80	none.

Wear.—Arsenic-bronze wore 42 per cent. faster than phosphor-bronze.

Arsenic-Bronze versus Phosphor-Bronze—Second Experiment.

	Composition, arsenic-bronze. Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	82.20	79.70
Tin,	10.00	10.00
Lead,	7.00	9.50
Phosphorus,	none.	0.80
Arsenic,	0.80	none.

Wear.—Arsenic-bronze wore 15 per cent. faster than phosphor-bronze.

Arsenic-Bronze versus Phosphor-Bronze—Third Experiment.

	Composition, arsenic-bronze. Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	79.70	79.70
Tin,	10.00	10.00
Lead,	9.50	9.50
Phosphorus,	none.	0.80
Arsenic,	0.80	none.

Wear.—Arsenic bronze wore 1 per cent. faster than phosphor-bronze.

Damascus-Bronze versus Phosphor-Bronze.

	Composition, Damascus-bronze. Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	77.00	79.70
Tin,	10.50	10.00
Lead,	12.50	9.50
Phosphorus,	none.	0.80

Wear.—First experiment, Damascus-bronze wore 8 per cent. slower than phosphor-bronze; second experiment, Damascus-bronze wore 7.30 per cent. slower than phosphor-bronze.

Alloy "B" versus Phosphor-Bronze.

	Composition, alloy "B." Per cent.	Composition, phosphor-bronze. Per cent.
Copper,	77.00	79.70
Tin,	8.00	10.00
Lead,	15.00	9.50
Phosphorus,	none.	0.80

Physical Properties.	Alloy "B."	Phosphor-bronze.
Tensile strength per square inch, pounds,	24,000	30,000
Elongation, per cent.,	11	6

Wear.—Experimental alloy "B" wore 13.50 per cent. slower than phosphor-bronze.

If we interpret the above results correctly, they indicate: (1) that copper-tin wears nearly 50 per cent. faster than standard phosphor-bronze; (2) that arsenic-bronze, containing no lead, wears about 42 per cent. faster than phosphor-bronze; (3), that arsenic-bronze containing 7 per cent. of lead wears less rapidly, the exact figure being 15 per cent. faster than phosphor-bronze; (4) that arsenic-bronze containing the same amount of lead as phosphor-bronze wears but slightly faster, the figure being 1 per cent.; (5) that Damascus-bronze containing as high as 12.50 per cent. of lead wears from 7 to 8 per cent. slower than phosphor-bronze; and (6) that the experimental alloy "B," containing less tin and more lead than any of the other alloys experimented with (the figures being 8 per cent. of tin and 15 per cent. of lead, instead of 10 per cent. of tin and 9.50 per cent. of lead, as is characteristic of phosphor-bronze), wears 13.50 per cent. slower than phosphor-bronze. This last alloy is the only one of which we have the physical properties compared with phosphor-bronze, and it will be observed that it has considerably lower tensile strength, with greater elongation, than the phosphor-bronze. This characteristic of lower tensile strength and greater elongation, it will be remembered, is the same characteristic which has been so often alluded to in the case of steel; namely, the mild steel, which, as is well known, is characterized by lower tensile strength and greater elongation than harder steel, gives the best wear. Here, too, in the realm of alloys, that metal which gives the lower tensile strength and greater elongation, if our experiments can be trusted, gives the slower wear. As has already been stated, it is unfortunate that the physical tests of all the other experimental alloys were not taken; but there is another method of getting at the physical properties of these alloys which is instructive, and although not as definite, is, perhaps, almost as convincing as though the actual figures were given. It is well known that the alloy of two parts of copper to one of tin, which is commonly called speculum metal, is one of the most brittle substances known. If, now, the percentage is changed, and an alloy is made of three parts of copper to one of tin, it is less brittle than the first one. Four, five, six, seven, eight, etc., parts of copper to one of tin give alloys which are in every case less and less brittle, as the tin diminishes, if our data can be trusted. It is also well known that the alloys of four, five, six, and sometimes seven parts of copper to one of tin are used for making bells for various purposes. The alloy of seven parts of copper to one of tin is the copper-tin alloy used as bearing-metal in our experiments. If this

reasoning is correct, the law seems to be that the increase in copper and the diminution in tin, at least within the range of our experiments, give alloys which are less and less brittle in every case. In the experiments which we have made it will, of course, be urged that lead comes in as an important constituent in the alloy, and consequently the query arises, What influence has the lead on the alloy? The influence of the lead has not been worked out in figures to our knowledge, but so far as our observation and study have gone, the addition of lead practically amounts to the same thing as a still further diminution of tin; that is to say, an alloy containing any given percentage of copper and tin will be rendered more ductile and less brittle by the addition of lead to this alloy, which, if our reasoning is correct, is the result which follows a diminution of tin and an increase of copper. I have, in another place, tried to illustrate this subject at some length, and will not, therefore, at this time dwell longer upon it, except to say that in the only case in which we have a physical test of these alloys,—namely, in the case of phosphor-bronze and the experimental alloy "B,"—this law seems to be fairly well illustrated. In the standard phosphor-bronze, the ratio of copper to tin is very nearly eight to one, while in the experimental alloy "B" the ratio of copper to tin is about nine and six-tenths to one. The percentage of lead in phosphor-bronze is 9.50 and in the experimental alloy "B" 15 per cent. The marked influence of these changes on the tensile strength and elongation is very clear. The tensile strength is cut down 6000 pounds per square inch, or about one-fifth, and the elongation is increased to nearly double that of phosphor-bronze. I will add for information that the experimental alloy "B," with a slight modification so as to enable the foundry to use the large quantities of phosphor-bronze scrap which the Pennsylvania Railroad possesses, and which results in giving from 0.10 to 0.20 per cent. phosphorus in the finished bearing,—the percentage of the other constituents being those given above,—is now the standard bearing-metal of the Pennsylvania Railroad, and no information has been obtained, during some six years of constantly-increasing use of this metal, which would controvert the conclusion given above, that the experimental alloy "B" wears more slowly than standard phosphor-bronze. It is possibly hardly necessary to add that we are not able to draw from our experimental work in the realm of alloys any other conclusion than that those alloys which are least brittle, or, expressed in technical language, which have lower tensile strength and greater elongation, give better wear as

bearing-metal than those alloys in which the reverse is the case, or, in other words, that in the realm of the alloys, so far as our experiments have gone, the same thing holds true which we have heretofore found in regard to steel, namely, the softer metal gives the better wear.

A few points further in regard to wear. We are not aware that any attempt has ever been made to formulate the variables on which wear depends, or, in other words, to enunciate a theory of wear, and it is entirely possible that the data in our hands, which are reliable enough to be so used, are not at all sufficient to warrant us in making such an attempt. Our observations, however, have led us to philosophize on this subject, and at the risk of saying something which future experiments may very greatly modify, or possibly show to be fallacious, we will venture to state a few of the variables which enter into wear.

Of course, wear is influenced by the conditions under which it takes place, but it is not our purpose to discuss these variables which may fairly enough be called "concomitant conditions." We will therefore not discuss lubrication, pressure, speed, temperature, rolling friction, or abrasion, nor indeed the nature of the two metals rolling or sliding over each other, but will confine ourselves wholly to the qualities of metal, which, all other things being equal, give least loss of substance by wear under the same service. To our minds we are justified in assuming that at least three elements enter into the problem of wear.

1. That metal which will suffer the most distortion without rupture will wear best. This quality of metal is usually measured or expressed in figures by the well known physical "elongation" or stretch before rupture in the common physical test. Possibly we possess more experimental data on this point than on any other of the variables which enter into wear. If we may trust the data which we have brought forward, and the conclusions drawn from them, in all cases the greatest elongation is accompanied by the best wear; or, according to the law, that metal which is characterized by the greatest power to resist distortion without rupture will wear best.

2. The first variable being obtained in satisfactory amount, an increase in tensile strength will add to the wearing-power of the metal. The diminution of tensile strength, which, according to our data, is characteristic of the better wearing-metals, is not in itself, if we are correct, a desirable quality. It is a concomitant of most

metals that as they increase in power of elongation, or stretch, before rupture, they diminish in tensile strength. If, on the other hand, a new metal could be found, which, with any given elongation, was characterized by a higher tensile strength than some old and well-known metal with the same elongation, the new metal would, if the theory is correct, wear better than the old one. It is not difficult to see why an increase in tensile strength should be valuable in assisting wear, provided the power of distortion before rupture, is not interfered with. Wear, as we understand it, is the tearing off of minute particles, and if in one case it requires more force to tear off the particles than in another, the wear in that case will be slower. We have, we think, a few experimental data which point in this direction. The wear of bearings per thousand miles, is about three times as fast as the wear of axles; in other words, as has already been stated, the standard phosphor-bronze bearing-metal loses about a pound for each 25,000 miles that the bearing moves. The axle under it loses about a pound for each 75,000 miles, but the metal of the axle is from two to three times as strong per square inch, and its elongation is also somewhat higher than the bearing-metal alloy.

3. The third variable which enters into wear, as we look at it, is what may perhaps be termed the "granular structure of the metal." This may, perhaps, best be illustrated by saying that, of two metals which have the same tensile strength and the same elongation, the one which is finer in granular structure will wear the slower. This we think will be evident by returning to our conception of what wear is—namely, the tearing-off of minute particles from the worn body. If, now, at each rupture of a particle of metal, the particle torn off is in one case twice as large as in the other, the wear will be twice as rapid, and we assume that, other things being equal, the granular structure represents the size or fineness of the particles torn off at each operation during wear. We have only a few experimental data on this point. It is generally believed by those who have a chance to make observation that what is known as case-hardened iron wears better then either the wrought-iron from which it is made or than ordinary hammered steel of approximately the same carbon content. It has also been observed that case-hardened metal is always characterized by an extremely fine granular structure, as evidenced by the fracture. This, of course, is only an observation and cannot be taken as proving very much. It is also entirely possible that the influence of what is technically known as "tempering," on wear, may appear in the effect of the temper on the granular

structure. The field is, of course, too void of experiment and too little known to warrant anything more than suggestions.

The relation and interaction, so to speak, of the three variables mentioned above is, of course, quite an unknown field. We are inclined to think that the experimental data which have been obtained point clearly to the conclusion that the increase in tensile strength at the expense of elongation is disastrous, as far as wear is concerned. An increase in tensile strength with an increase in elongation would unquestionably, we think, be valuable. The influence of the structure or granular condition may be even more important than either elongation or tensile strength. Of course the data do not warrant any conclusions, but it seems not at all improbable that the size of the particle torn off, each time that one is torn off, may be the most important variable in the rate of wear. It is also not improbable that if some method of measuring the granular structure of metal, and rightly estimating its influence on wear, were known, this information would go far toward explaining many of the anomalous cases of wear, which, if our experience is worth anything, are almost universal accompaniments of experiments in this field.

The whole subject of the relation between wear and the chemical and physical properties of metal needs study and positive experiment, and it is quite possible that much of what we are accustomed to rely on at the present moment may be upset or overthrown by wider knowledge. The best we can say at present is that, with the light which we have, the highest tensile strength, accompanied by the highest elongation and the finest granular structure, are the physical properties which will probably give the best results in actual service where the metal is subjected to wear, and that that chemistry which will give these results in the finished product, be it in the realm of the alloys, or in the magnificent field of steel metallurgy, or, possibly, in the coming field of a metallurgy based on aluminum, is the best chemistry which we, at the present moment, are able to recommend.

Discussion.

Sir Lowthian Bell, Middlesbrough, England: It happens that the President, Sir James Kitson, and myself are members of the locomotive committee of the North Eastern Railway, a road 1500 miles in length, and therefore affording ample scope for any inquiry bearing upon the question now submitted to this meeting. I entirely agree with Dr. Dudley in the difficulty of assigning a true cause for

the differences which are observed in the wear of rails. Upon the North Eastern Railway we followed at first the advice given by the rail manufacturers, viz., to keep the carbon low, thinking that the softer the steel, the stronger the rails would be, and we continued not to exceed 0.3 to 0.4 per cent. of carbon. Our engineer at the time (a gentleman of great experience, and greatly respected for the use he had made of that experience) and several members of the committee, including myself, frequently discussed the question whether we might not, with advantage to the railway, raise the quantity of carbon in the steel with a view to increasing its hardness, and, as we hoped, increasing its resistance to wear upon the track. At length, at our advice, the Board of Directors consented that our rails should be raised from 0.3 and 0.4 to 0.5 per cent. of carbon, and for some years past they have been guaranteed by the makers to contain that quantity of carbon. We have not, however, had sufficient experience to enable us to pronounce positively as to the benefit of the change. Though the question may seem a very simple one, it is, in fact, replete with difficulty. For purposes of comparison the conditions to which the rails are exposed should be exactly the same in every case, but in this lies a great practical difficulty. Dr. Dudley cites the wear of rails during a certain number of years upon the New York and New England Railroad. If we were to collect the statistics of our own line in that way we should obtain very discordant results, not only because the traffic upon our line has assumed very much larger proportions, but because we have greatly increased the weight of our locomotive-engines, which, so far as wear is concerned, is perhaps of much more consequence. Having this constant change of conditions in view, I was anxious to establish some rapid mode of ascertaining the relative wear of rails of different compositions, and accordingly the superintendent of our locomotive-department constructed a machine with which, by the rapid revolution of a wearing-surface,—like a kind of grindstone,—we might, in a few weeks, compare the results of actual loss in weight of the rail. Perfect success has not yet attended our investigations with this contrivance, but I am not going to relinquish them on that account.

I have heard with great surprise that the carbon in American locomotive-tires is as high as 0.6 to 0.7 per cent. So far as I know, we have no tires running upon the North Eastern Railway of England which contain much more than half that quantity. Our locomotive-tires run at about 0.45 per cent. and our wagon-tires at 0.35

per cent. of carbon. If an analysis of such high-carbon tires as those mentioned by Dr. Dudley were brought before our locomotive committee, we should feel somewhat alarmed as to the possible results of their use. However, it is perhaps premature to declare any opinion very strongly upon that subject until we have had more experience.

We keep a record of all our broken rails. Our line, as I have mentioned, is about 1500 miles long, and of that stretch something like 1400 miles are laid with steel rails. Whenever a rail is fractured it is brought into the chemical laboratory of the company and submitted to an accurate analysis; yet I agree entirely with Dr. Dudley, that all the chemical information we have secured up to this time has failed to elucidate the cause of the fractures. I am inclined to believe that it may be due rather to the existence of that critical condition or that critical point of the rail—arising perhaps from too sudden or too slow cooling—which depends on questions in no way related to chemical composition.

THE INSPECTION OF MATERIALS OF CONSTRUCTION IN THE UNITED STATES.

BY GEO. H. CLAPP AND ALFRED E. HUNT, PITTSBURGH, PA.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

THE great advantage to be gained by a careful inspection of all materials used in construction, in regard to safety and permanency, and to accuracy of workmanship, has been longer recognized in Great Britain and on the Continent than in America. This is probably owing to the way in which great enterprises have been "rushed" in the United States. Time and cheap first cost, in too many instances, have been considered of more importance than absolute safety and durability; the structures being afterwards strengthened if this was found necessary.

This has been most noticeable in railroads built through "new" country, where the object in view has been to reach the terminal point and get to work. These "loose" methods are now being very largely done away with, and a large proportion of the structures lately built are not only carefully inspected during the process of

manufacture of the iron or steel, but also during the working and assembling at the bridge shop, and while the erection is going on; the manufacturers, in many cases, by the terms of the specifications, not being relieved from the chance of rejection until the final acceptance of the finished structure after erection. The reasons calling for this increased care in inspection, such as increase of traffic and rolling loads on railroads, heavier and more substantial buildings, etc., have been so thoroughly discussed before the different engineering societies that it is unnecessary to take them up here, the object of this paper being to show the different requirements that iron- and steel-makers in America are called upon to meet, and which British and Continental makers as well have occasionally been called upon to fulfil for American orders.

The practice of having structural material inspected has increased a great deal within the last few years in the United States, until at present the very largest share of all structural materials for bridges, boilers, or buildings is now ordered subject to inspection, and there are several firms who make the inspection of iron and steel a specialty. Formerly a great many railroad companies, when they required any inspection at all, sent out some of their office-force, frequently young men with absolutely no knowledge of iron and steel whatever, men who could not explain the terms of the specifications under which they were expected to work. Their inspection was on a par with their ignorance, and consisted principally in making trouble for the mills. Several railroad companies now maintain regular corps of inspectors, who are under the direct supervision of the chief engineers, the general superintendents, or the superintendents of bridges and buildings. Of the private inspecting firms, the successful ones are mainly those whose members have served their apprenticeship either in iron- and steel-works or at the large bridge-works; and as they have their own men constantly located at all of the principal structural and rail-mills, most of the railroad companies find it more economical and convenient to employ them. Being constantly at the mills, they also become more thoroughly acquainted with individual peculiarities at the various locations, and can thus work with less friction. Several of the larger works have provided offices exclusively for the inspectors, where there are desks at which they can do their clerical work, and, if permanently located at the works, keep their order- and report-books locked up.

The growth of inspection has also necessitated the use of accurate and rapid means of measuring and calculating, so that all well-

equipped testing-rooms are now provided with standard micrometer-gauges, calipers, and steel rules, tables of diameters and areas, frequently tabulated in such a way that the percentage of reduction of area can be read at a glance. Another almost universal equipment is the "Thacher Slide Rule," with which probably most of the members are familiar, as it was fully described in a paper read some years ago before the British Association for the Advancement of Science, by Mr. W. F. Stanley.

As regards the requirements of testing, there is a growing tendency among engineers towards uniformity, largely brought about by the publication of "Standard Specifications" by such well-known engineers as Theo. Cooper, C. C. Schneider, J. A. L. Waddell, and others; Mr. Cooper's specifications, in particular, having been adopted by a great many railroad companies and by County Commissioners for highway work.

The general specifications given below are an average compiled from those mentioned above, and from the specifications of some of the largest railroads, all strains being referred to in terms of pounds per square inch, instead of tons per square inch, as the former has become the universal custom in the United States.

General Specifications.

Complete facilities for inspection of material and workmanship must be given by the contractor, and specimens for testing, and the necessary labor, shall be furnished by him, without charge.

The acceptance of any materials or manufactured members by the inspector shall not prevent their subsequent rejection, if found defective after delivery, and any rejected material shall be replaced by, and at the expense of, the contractor.

Material to be tested shall be of full-sized section wherever practicable, or in bars or strips of at least 0.50 square inch in sectional area, and, where possible, should be at least 18 inches long, and have two opposite sides left as they came from the rolls. Test-pieces should not be cut from short crop-ends, as they almost invariably contain flaws that cause erroneous results.

A duplicate specimen of each bar or strip to be tested, except in the case of full-sized bars or members, shall be bent cold until it commences to rupture, or through an angle of 180° around a cylinder having a diameter specified for each kind of material, or until the sides are in contact. The bending must stop at the first indication of fracture.

Test-specimens must not be annealed, heated, hammered, forged, or otherwise treated, except where specified in the contract; but shall fairly represent the quality of material to be tested.

The testing-machine used by the contractor shall be compared with a standard testing-machine, and if the results vary, the contractor's machine shall be corrected until the tests show that the machines accord.

Wherever full-sized wrought-iron eye-bars are tested, the material shall be allowed a reduction of 1000 pounds in ultimate strength, and of 2 per cent. in reduction of area, for each additional inch of sectional area over $4\frac{1}{2}$ square inches, down to a minimum of 46,000 pounds per square inch, and 15 per cent. in reduction of area. Such bars shall have an elongation of 10 per cent. in their whole length when the sectional area is under $4\frac{1}{2}$ square inches, and of 8 per cent. when the sectional area is over $4\frac{1}{2}$ square inches.

In the selection of eye-bars or tension-rods for full-sized tests to destruction, the inspector shall select from fairly representative bars of the whole lot, and shall cause to be made under his personal supervision, three eye-bars or rods of each size and lot to be tested; from these bars he shall select one for testing. All of the test-bars must be made in the same way as the lot which they are to represent.

If the full-sized tests are satisfactory, the bars shall be paid for at the contract price, less their scrap value, by the owners of the bridge or parties ordering the tests to be made. If they do not stand the specified tests, they will be considered rejected material, and will be solely at the cost of the contractor.

Specifications for Wrought-Iron.

All wrought-iron shall have an elastic limit of not less than 25,000 pounds per square inch, and must be tough, fibrous and uniform in character, neutral in quality and completely welded in rolling, and must have a workmanlike finish. It must be free from injurious seams, especially such as run across the fiber, and must have no blisters, buckles, cinder-spots, rough or imperfect edges, or deep creases from tool or roll-guide marks.

The finished iron must be straight, and out of wind, and must not have over 3 per cent. of variation between the actual and the estimated weights.

All wrought-iron, when nicked and broken at a temperature of over 50° Fahr., or when pulled apart in testing-machines, must show a clean fibrous structure, except in the case of the webs of shape-iron,

which may be allowed to exhibit as much as 30 per cent. crystalline in the fracture.

The finished iron must show no evidence of butt-welds, and the piles for the wide bars must have the top and bottom covers, and for bars of over one inch in thickness, at least one other bar in the center of the pile double-rolled and extending over the entire length and width of the pile. Wherever bars of less width than the entire pile are used, the joints shall be well broken in the piling. Material cut too close to the crop-ends will not be accepted.

All wrought-iron used in tension, except wide plates, shall be made from "double" or "single" rolled-iron piling-pieces.

Wide plates may, and preferably, should have a few alternate layers of muck-bar laid crosswise and extending completely across the pile.

Wherever single-rolled iron and iron made from piling scrap, or from re-rolling old rails, is to be used, the items in which it will be permissible shall be individually specified in the contract.

Specifications for Tension-Iron.

All material subject to alternate compression- and tension-stress shall be considered as in tension.

Kind of Material.	Tensile Tests.				Cold-Bending Tests.	
	Limit of Elasticity per sq. inch.	Ultimate Strength per sq. inch.	Per cent. Elongation in 8 ins.	Reduction per cent. at Fracture.	Angle of Bend.	Diameter of Cylinder around which the Specimen is to be bent.
Bar Iron.						
Round bars up to 1½ in. diameter	26,000	50,000	18	25	180°	Upon itself until sides come in contact.
Round bars up to 2⅜ in. diameter	26,000	50,000	15	20	160°	One diameter of bar.
Square bars up to 1½ in. per side	26,000	50,000	18	25	180°	Upon itself until sides come in contact.
Square bars of less than 4½ sq. in. sec. area	26,000	50,000	15	20	180°	Diameter equal to the sides of bar.
Flat bars of less than 4½ sq. in. sec. area	26,000	50,000	15	20	180°	Diameter equal to two times the thickness.
Bars of more than 4½ sq. in. sec. area	26,000	50,000	12	18	140°	Two diameters of bar.
Channels and Beams.						
Test from web	26,000	47,000	12	18	140°	Diameter equal to thickness of specimen.
Test from flange	26,000	49,000	15	20	160°	Do.
Angle-iron	26,000	49,000	13	20	140°	Do.
Other shape iron	26,000	48,000	14	18	120°	Do.
Plate Iron. Tests to be taken with the fiber.						
Under 18 inches wide	26,000	50,000	15	20	160°	Diameter twice thickness.
18 to 36 inches wide	26,000	48,000	12	16	100°	Do.
36 to 54 inches wide	25,000	46,000	10	10	90°	Do.
Over 54 inches wide	25,000	46,000	8	8	90°	Do.

All iron must come up to the requirements expressed in the foregoing table, in test-specimens cut from the finished material.

Earlier specifications made a difference in the requirements between material to be used in compression and in tension, with the intention of ordering for compression-members, in the case of iron, a poorer quality, not as high in tensile strength and ductility, and in the case of steel, a harder material much higher in tensile strength and with considerably less ductility, than for tension-steel. The great difficulty, however, in keeping these various grades separate, and the serious error of having the mills furnish one class of material to be used in the place of the other, has occasioned the abandonment, by most engineers, of the differing requirements for tension and compression members. However, some engineers of late have been ordering three grades of steel, which can readily be kept separate in ordering, namely, the material for the trusses of the highest ductility and tensile strength, the latter ordinarily running from 65,000 pounds to 75,000 pounds; the material for the floor-system of medium steel of greater ductility and of lower tensile strength, from 60,000 to 70,000 pounds; and for rivet-steel, material of not over 60,000 pounds tensile strength. The great majority of specifications in the United States, however, only require two grades of steel, that for the main members of the structure being all of one kind, the only exception being the softer steel used for rivet-rods.

Specifications for Rivet-Iron.

Rivet iron shall be subject to the same requirements as tension-iron of the same section, and shall further be capable, without cracking or serious abrasion, of being heated to a good forging-heat, and made up either by machine or hand-work into rivets, and of again being heated to a good red heat, forged as in riveting, allowed to cool, and upon being nicked and cut out of the work, must show good, tough, fibrous structure, without any crystalline appearance. Rivet-iron shall especially be required to be neutral in character and tough in fibre *after being riveted*, and to flow well in riveting.

Specifications for Cast-Iron.

All cast-iron used in compression must be of best quality—tough gray iron—and must be able, in bars of 1 inch square and 5 feet length, and when supported by knife edges 4 feet 6 inches apart, to carry a load of 500 pounds applied at the center, without rupture, and without a deflection of more than 2 inches.

The specifications above stated give substantially all of the requirements in regard to iron, excepting that some engineers permit the use of a little steel scrap in piling, while others will condemn material showing the slightest indication of steel having been used. The use of steel will require much more elaboration, owing to the different requirements of engineers, some merely specifying "steel" with certain physical requirements, while others prescribe the chemical requirements and the method of manufacture as well, making the specifications very rigid and difficult of fulfilment; in fact, some of the recent specifications have been so "stiff" that the manufacturers have thrown up the contract after getting started.

Specifications for Steel.

The method of manufacture of the steel, whether by the Bessemer open-hearth, or crucible processes, shall be distinctly specified in the contract for each item of material.

Each ingot designed to be used in the contract shall be plainly marked with the blow- or heat-number in the case of Bessemer or open-hearth steel, and afterwards this heat- or blow-number shall be stamped upon each piece of finished material rolled from it, and a portion of a full-sized ingot, from each heat or blow to be used, shall be rolled into a test-bar $\frac{3}{4}$ inch in diameter. The method of obtaining the $\frac{3}{4}$-inch round test-bar shall be the same with all of the heats or blows tested, and shall represent in the $\frac{3}{4}$-inch round the amount of work in the finished product as far as is practicable.

These bars are to be truly round and finished at a uniform dark orange heat, arranged to cool uniformly and without any danger of hardening, from cool draughts of air, being laid on damp ground, or the like, and if any of these bars prove unsatisfactory in tests, the whole blow or heat represented by such defective bars shall be rejected.

Each heat or blow of steel to be used shall be tested first in the $\frac{3}{4}$-inch round test-bar, both in tensile and bending tests required, and afterwards in specimens to be cut from the finished material; provided that the number of such tests shall not be over four in all for each melt or blow, including the test in $\frac{3}{4}$-inch round.

The tests first made on the $\frac{3}{4}$-inch test-bars must exactly conform to the requirements of the specifications. The tensile tests made from the finished steel will be accepted if they give results within 4 per cent. of the requirements of the specifications.

If drillings taken from any portion of the finished material, or

from the $\frac{3}{4}$-inch round test-bars do not satisfactorily answer the chemical requirements of the specifications, the whole charge or blow is to be rejected, unless it shall be proved that the defect was caused by the particular ingot or bloom having been "burned" in heating.

No steel shall be used for structural purposes which contains over 0.08 per cent. phosphorus, and the average percentage of phosphorus of the steel of any given structure, or on any given order, shall not exceed 0.07 per cent.

No steel shall be struck with a hammer or worked while at a black heat.

A variation in cross-section or weight of rolled material of more than 3 per cent. from that specified, may be cause for rejection.

Finished bars must be free from injurious flaws and cracks, and must have a workmanlike finish.

Test-specimens shall bend through an angle of 180 degrees, and close down upon themselves without showing a crack or flaw, and similar bars shall have an ultimate strength of from 58,000 pounds to 68,000 pounds per square inch, and an elastic limit of not less than 36,000 pounds per square inch. Tensile test specimens shall elongate at least 23 per cent. in 8 inches, and show a reduction of area of at least 42 per cent.

Test-pieces shall be of at least $\frac{1}{2}$ square inch sectional area, and cut lengthwise from finished bars or shapes, or either crosswise or lengthwise from plates.

All steel eye-bars must be annealed in a properly constructed furnace, and all eye-bars must show the same degree of tenacity and ductility, proportioned to the size of the bar and different manner of testing, that the steel from the same blow or melt did in the various specimen tests on which it was accepted at the rolling-mill.

Specifications for Rivet-Steel.

All rivet-steel shall be subject to the general specifications given above, and shall further be capable, without cracking or serious abrasion, of being heated to a good forging-heat, and made up either by machine or hand-work into rivets, and of again being heated to a good red heat, and forged or pressed as in riveting, and allowed to cool; and upon being nicked and cut out of the work it is in, must show a good, tough, silky structure with no crystalline appearance. Rivet-steel especially shall be neutral in character and pure in composition, and shall flow well in riveting.

Rivet-steel should not have over 0.15 per cent. carbon, and must

not have an ultimate strength of over 60 000 pounds per square inch, and must have an elongation of at least 25 per cent. in 8 inches.

Steel rivets should not be heated above an orange color, much less than the heat put upon iron rivets, which are heated nearly to a lemon color, often, without apparent injury.

Government engineers and many other engineers are requiring that rivet-rods shall be bundled in lots of a ton weight; that the steel of each melt shall be kept separate, and have a metal tag securely fastened to each bundle, with the melt-number of the steel stamped upon it; this in lieu of each rod being stamped with the melt-number, as with bars of larger section.

Rarely have specifications for steel demanded the tests of the modulus of elasticity (although many engineers have asked for these tests, as a matter of record), as it has been found by practice that the modulus of elasticity varies but slightly for very considerable differences in the quality of structural steel—that is, steel from 55,000 to 80,000 pounds tensile strength runs very closely alike in modulus of elasticity, and can be safely estimated as 30,000,000, whether the steel is made by the open-hearth or the Bessemer acid or basic process.

In addition to the unannealed specimens cut from the finished material, it is becoming a widely established practice to require a certain number of annealed specimens of the finished materials to be pulled. This, especially, is required with eye-bar material, to prevent the acceptance of material which would be too soft fairly to fulfil the requirements of the specification, but which by cold-rolling —that is, allowing the finishing passes to be made on the metal at a very dark red-heat—will show a considerable increase in tensile strength, the metal being hardened thereby so far as readily to answer the requirements of the specifications. At the same time, when such bars are sent to the bridge-shops, and made into eye-bars and thereafter carefully annealed, according to the requirements of all bridge specifications, and pulled in the testing-machine, the results will be found to fall far below the specifications, and what should be expected of the material from the tensile tests of the finished material. By having a certain number of specimens cut from the full-sized bar and annealed, this practice on the part of the mills is prevented.

Whenever a test-specimen does not fairly represent the average

lot of the material, it is the practice of most inspectors to allow a second test to be made, when, owing to flaws, the first test is manifestly not fair. This is frequently occasioned by cutting the test-specimens from the ends of bars which have been cut too close to the crop-end. In such cases the second finished material test-pieces are taken by cutting actually into the center of bars to be furnished for the work, thus ensuring that the test-pieces are average specimens.

The requirement for maximum content of phosphorus allowed in material has been specified differently for steel manufactured by the various methods, the highest maximum, from 0.08 per cent. to 0.10 per cent., being allowed in acid open-hearth steel. Most engineers are now requiring the maximum in all cases to be less than 0.08 per cent., and for Bessemer not over 0.05 per cent. phosphorus. Some engineers are requiring the average of Bessemer to be less than 0.04 per cent., claiming that only Bessemer steel of this degree of purity can at all equal in uniformity open-hearth steel. For basic open-hearth steel the requirements still vary, and basic steel has not been in use long enough to have the practice settled as to what should be the requirements of the specification. We believe, however, that the limit will be placed at 0.04 per cent. for basic open-hearth steel. Our experience so far with basic open-hearth steel for structural purposes has been that when the carbon runs down to 0.15 per cent. if the basic operation has been successfully carried out in the melting-furnace, the percentage of phosphorus will be below 0.04 per cent., and that if the content of phosphorus is above this, unless the higher phosphorus comes from very high-phosphorus final manganese additions, it is liable to be non-homogeneous, and the steel is liable to show unsatisfactory results under tensile tests; and this is an evidence that either the slag was not completely basic during the melting-operation, or that the temperature in the open-hearth furnace was unsatisfactory, and the working irregular, since in the contrary case the phosphorus would have been reduced below 0.04 per cent. before the carbon had been reduced to 0.15 per cent.

Many engineers have inserted in the requirements of the specifications that the contractors, in placing their bids, shall specify the maximum and minimum limits through which the carbon and manganese shall vary in the steel to be furnished, and also the maximum limit of the sulphur, phosphorus, and silicon.

A large amount of the steel which is now sold subject to specifications has accompanying the invoice a chemical report of the percentage of carbon, manganese, and phosphorus of each melt used in

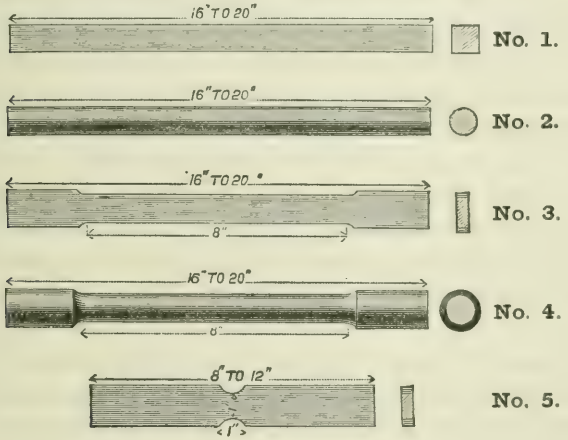
the shipment, which report is furnished by the manufacturer. These analyses are certified by the chemist of the works. Many engineers require, in addition to this, reports of check-analyses, made by inspectors or chemists to be selected by them, of one sample from each five or ten melts.

Test-Pieces.

Fig. 1 shows the standard shapes of specimens for tensile tests of both iron and steel.

Some of the railroads adopt test-pieces of 2, 3, 4 and 5 inches between shoulders, but the majority are now coming round to

Fig. 1.



Shapes of Specimens for Tensile Tests.

No. 1.—Square or flat bar, as rolled.

No. 2.—Round bar, as rolled.

No. 3.—Standard shape for flats or squares, edges smooth and true, fillets $\frac{1}{2}$-inch radius.

No. 4.—Standard shape for rounds.

No. 5.—Government shape for marine boiler-plates only. Not recommended for other tests, as results obtained by this shape are generally in error.

the above standard shapes, as drawn by Wm. Kent, M.E., some years ago, and since adopted and recommended by Prof. R. H. Thurston, in his *Materials of Engineering*, and by several other writers on the testing of iron and steel. The sizes recommended by Mr. Kent are:

For tool-steel, either $\frac{1}{16}$ inch (1.75 centimeter) diameter, or $\frac{3}{8}$ square inch (2.44 square centimeters) area; in other metals, either $\frac{3}{4}$ inch

(1.9 centimeter) diameter, or .44 square inch (2.84 square centimeters) area.

These sizes are adapted to a 50,000-pound testing-machine, which is the size of machine most commonly used in the United States, although mills putting in new machines are largely adopting the 100,000- or 200,000-pound sizes.

The form of test-pieces generally adopted in testing iron at the mills is the parallel-side strip about 16 to 20 inches long, as it is found to give satisfactory results, and is much cheaper to prepare than the turned section. Of course, large squares and rounds have to be turned down, and it is found that in so doing the best and strongest part of the metal is turned off, and only the weak center left, which point we do not think is recognized so generally as it should be; and frequently iron, which in full-size section would show 45,000 to 48,000 pounds, runs down to 43,000 to 45,000 pounds in the turned section. For this reason we request in our own specifications, that when possible "two opposite sides of the test-piece should be left as they came from the rolls."

One advantage in the straight-side test-piece is that, no matter where it breaks, the elongation in 8 inches can be taken; while in a turned specimen, 8 inches between shoulders, if it break at the fillet, there is no way of measuring elongation, and the record is incomplete.

It has become a very general practice among engineers, in specifying the quality of steel to be used, to require a preliminary test of the material, the test-piece for which shall be of such a size as fairly to represent the amount of work done on the finished material. This test-piece is taken in the same manner in all cases, being cast directly from the ladle into a small ingot 4 inches square and 1 foot long, usually when the ladle is about half-emptied. These test-ingots are then treated as nearly as possible in a uniform manner as regards time of reheating, rolling, and allowing to cool, special care being observed in this last step that there may be no tempering caused by the bars being exposed to draughts of cold air or from being laid on damp or wet places. At a great many mills troughs are placed near the rolls, filled with dry sand, on which the bars are laid to cool.

With these precautions the test-specimens are supposed, in each case, fairly to represent the quality of the material; and the requirements of the specifications are, that the steel of these preliminary test-bars shall exactly answer the specifications, further provision

being made that tests of the steel of each melt shall be made on bars cut from each largely varying section into which the steel of each melt is rolled, an allowance being made in many cases of 4 per cent. In some specifications the amount of variation is left to the discretion of the inspector, with the statement—"A reasonable allowance shall be made in these tests taken from the finished material for the variation in section."

The intention of the engineers who draw up their specifications in this way is to obtain for their structures steel as closely as possible *uniform in its chemical composition and uniform under tests made under like conditions.*

The specifications usually state that the steel shall be cast into test-ingots of 4 inches square, of a length of about 1 foot, and that the test-ingot shall be cast in a mould as uniformly as possible in method, when about one-half of the metal has been cast out of the ladle, the ingots to be rolled usually upon a 9-inch mill to bars of $\frac{3}{4}$-inch round. This test is usually spoken of in America as the "preliminary $\frac{3}{4}$-inch round test."

This system of requiring a "preliminary $\frac{3}{4}$-inch round test" from which the original selection of the material will be made has been objected to by many of the large contractors and steel-manufacturers. They claim that the $\frac{3}{4}$-inch round test does not fairly represent the quality of the material, in that a small 4-inch square ingot is a very difficult ingot to obtain sound, and that flaws are, therefore, very apt to occur in the $\frac{3}{4}$-inch round test-specimens.

This, however, has not been our experience, and this statement we make from a very large experience with the $\frac{3}{4}$-inch round test. Our records show no more flaws in the $\frac{3}{4}$-inch round tests than in the tests cut from the finished material. Of course, where such flaws do exist, second or further specimens are pulled, until a sound specimen is obtained. Again, this objection seems to us to be begging the main question, which is upon the advisability of a uniform preliminary test, and not as to whether the test-ingot for this test should be cast in a 4-inch ingot mould or in a larger mould for the sake of securing a sounder ingot.

A further objection is that these $\frac{3}{4}$-inch round test-pieces are very liable to be cold-rolled, or to be hardened by too rapid cooling.

This argument, it seems to us, is one that only refers to careless work. If proper precautions are taken to have these $\frac{3}{4}$-inch round test-pieces rolled and afterwards cooled uniformly, they can, we believe, be obtained much more uniform than the average of test-pieces cut from the finished material.

The argument of the extra expense of the $\frac{3}{4}$-inch round test does not seem to us to be valid, if the advantages to be obtained from it are taken into consideration, as the test is one readily obtained at nearly all of the large steel-plants; and the bulk of the test-rods can be used for rivet-steel, or for other purposes for which the steel has a ready sale.

The real question at issue with reference to the preliminary $\frac{3}{4}$-inch round test seems to us to be *whether it is advisable to have the steel for any structure uniform chemically or uniform physically*—that is, whether plates and angles of $\frac{3}{8}$-inch thickness rolled from ingots of the same size, as thick eye-bars of $1\frac{1}{2}$-inch to $2\frac{1}{2}$-inch thickness ordinarily are, shall be made from the same quality of steel, and show the variation in tests due to the variation in work upon the different sections, or whether the thicker sections shall be made of higher-carbon steel, which will answer the same tensile requirements in the $1\frac{1}{2}$-inch to $2\frac{1}{2}$-inch thick bars that the softer lower-carbon steel does in the $\frac{3}{8}$-inch thick bars. In the one case the steel for the structure would be practically homogeneous chemically if the steel were accepted upon a preliminary uniform method of testing; and, in the other case, steel in a given structure would be non-homogeneous, much harder, and having a considerably higher percentage of carbon in the thick sections than in the thin.

At present the majority of engineers, we believe, require uniform steel chemically, and the preliminary test is, we believe, the "fashion," although the question is a moot one, and still deserves careful consideration. From a comparison of several thousand tests of both acid open-hearth and acid Bessemer steel, selected from our records of tests, made by us, of American structural steel for bridges, which we have inspected, the steel tested being as fairly representative as possible of steels ranging from 55,000 to 85,000 pounds tensile strength per square inch, we found the following average variations between the $\frac{3}{4}$-inch round test and the test of the finished material:

Plates, Angles, Channels and Beams.

	Per cent.			
Elastic limit,	5.15	lower in finished material	test than in	$\frac{3}{4}$-in. round test.
Tensile strength,	1.11	"	"	"
Elongation in 8 in.,	5.76	"	"	"
Reduction of area,	7.18	"	"	"

Tension-Bars.

	Per cent.			
Elastic limit,	5.08	lower in finished material	test than in	$\frac{3}{4}$-in. round test.
Tensile strength,	3.07	"	"	"
Elongation in 8 in.,	5.63	"	"	"
Reduction of area,	10.58	"	"	"

In getting these averages 4300 tests were actually tabulated and fully as many more were roughly compared. The steel was to fill a specification as follows:

Elastic limit, at least 38,000 pounds per square inch.
Tensile strength, from 65,000 to 75,000 pounds per square inch.
Elongation, at least 18 per cent. in 8 inches.
Reduction of area, at least 35 per cent.

This particular specification was taken, as the steel was considered fairly representative of the average structural steel in use in this country. The average of the $\frac{3}{4}$-inch round tests was as follows:

Elastic limit, 42,580 pounds per square inch.
Tensile strength, 69,625 pounds per square inch.
Elongation, 25.95 per cent. in 8 inches.
Reduction of area, 52.85 per cent.

The plates, angles and channels gave the following average:

Elastic limit, 40,390 pounds per square inch.
Tensile strength, 68,850 pounds per square inch.
Elongation, 24.44 per cent. in 8 inches.
Reduction of area, 49.56 per cent.

Bars ranging from 4 inches $\times$ $\frac{1}{2}$-inch to 8 inches $\times$ $2\frac{1}{2}$ inches gave the following average:

Elastic limit, 40,420 pounds per square inch.
Tensile strength, 67,480 pounds per square inch.
Elongation, 24.46 per cent. in 8 inches.
Reduction of area, 47.25 per cent.

One engineer states, in all of his recent specifications, that "steel may be made by the open-hearth or by the Bessemer process, but no steel shall be made at works which have not been in successful operation for at least one year;" and "steel made by the Clapp-Griffiths process will not be accepted."

He, however, gives his preference to open-hearth steel, and in his latest specifications, published this last summer, in giving the chemical requirements, says that Bessemer steel must in no case exceed 0.05 per cent. phosphorus, and that the average must not exceed 0.04 per cent., while in open-hearth steel he allows individual heats to run up as high as 0.10 per cent., and gives 0.08 per cent. as the average.

To show the changes in requirements that have taken place during the last ten years, we quote from six specifications drawn by the

same engineer, which have been considered "standard," and frequently referred to by engineers who have not regular specifications of their own.

1st. 1879.—Open-hearth steel for tension-members.

Not less than 50,000 pounds elastic limit.
Not less than 80,000 pounds tensile strength.
Not less than 12 per cent. elongation in 8 inches.
Not less than 20 per cent. reduction of area.

2d. 1881.—Open-hearth or Bessemer steel.

Tension-Members.

Elastic limit, 40,000 to 45,000 pounds per square inch.
Tensile strength, 70,000 to 80,000 pounds per square inch.
Elongation in 8 inches, at least 18 per cent.
Reduction of area, at least 30 per cent.

Compression-Members.

Elastic limit, 50,000 to 55,000 pounds per square inch.
Tensile strength, 80,000 to 90,000 pounds per square inch.
Elongation in 8 inches, at least 12 per cent.
Reduction of area, at least 20 per cent.

3d. October 1, 1882.—Steel must be open-hearth.

Tension-Members.

Elastic limit, not less than 40,000 pounds per square inch.
Tensile strength, not less than 70,000 pounds per square inch.
Elongation in 8 inches, not less than 18 per cent.
Reduction of area, not less than 45 per cent.

Compression-Members.

Elastic limit, not less than 50,000 pounds per square inch.
Tensile strength, not less than 80,000 pounds per square inch.
Elongation in 8 inches, not less than 15 per cent.
Reduction of area, not less than 35 per cent.

4th. December 1, 1885.—No special method of manufactur specified, but all steel of same class must be made by same process.

Tension-Members.

Elastic limit, not less than 40,000 pounds per square inch.
Tensile strength, not less than 70,000 pounds per square inch.
Elongation in 8 inches, not less than 18 per cent.
Reduction of area, 42 per cent.

Compression-Members.

Elastic limit, not less than 50,000 pounds per square inch.
Tensile strength, not less than 80,000 pounds per square inch.
Elongation in 8 inches, not less than 15 per cent.
Reduction of area, not less than 35 per cent.

5th. June 15, 1887.—Open-hearth or Bessemer steel made at works in successful operation at least one year. Clapp-Griffiths steel will not be accepted.

Phosphorus must in no case exceed 0.10 per cent., and must average 0.08 per cent.

Tension-Members.

Elastic limit, not less than 40,000 pounds per square inch.
Tensile strength, 67,000 to 75,000 pounds per square inch.
Elongation in 8 inches, at least 20 per cent.
Reduction of area, at least 42 per cent.

6th. Summer of 1888.—Steel made as under 5th.

Open-hearth steel—phosphorus not to exceed 0.10 per cent., and must average 0.08 per cent. Bessemer steel—phosphorus not to exceed 0.05, and must average 0.04 per cent.

Tension-Members.

Elastic limit, not less than 38,000 pounds per square inch.
Tensile strength, 63,000 to 70,000 pounds per square inch.
Elongation in 8 inches, at least 22 per cent.
Reduction of area, at least 45 per cent.

In specifications Nos. 1, 5, and 6, there is no mention made of compression-steel, and it was furnished of the same quality as the tension-steel.

The specifications show a tendency to give a decided preference to open-hearth steel, as well as a gradual lowering of the requirements in tensile strength and increase in elongation and reduction of area.

The latest specifications require a drifting-test for steel plates, angles and shapes—a $\frac{3}{4}$-inch hole, punched as in ordinary work for riveting, the center of the hole being $1\frac{1}{2}$ inches from the edge of the specimen, to withstand the operation of drifting, by blows of a sledge upon a drift pin, to at least $1\frac{1}{2}$-inch diameter without fracture. The advantage of this test is that it can be made very much more cheaply than ordinary tensile tests, or even bending-tests, where the sheared edges have to be planed off in order to give the steel a fair oppor-

tunity to show its ductility. Many of these tests can be made without much expense on the short crop-ends which are necessarily made in cutting the material to length, and the advantage of the many tests in proving the homogeneity of the steel is apparent.

Tempering-tests have also been introduced in recent specifications, and steel that has been heated to a dull cherry-red, and quenched in water at 60° to 80° Fahr., according to the different specifications, must bend 180° around a pin varying from 1 to 2 diameters, without sign of fracture.

This change has been of great benefit to the travelling public, as it ensures a greatly superior quality of material, in which the chances of failure due to sudden changes in temperature, or to careless heating and cooling, either at the mill or at the bridge-shop, are reduced to a minimum.

The effect of temperature on steel has now become very well known, and needs no discussion here. With iron, however, it is different, and a great variety of opinions exist as to the effects of temperature upon this form of metal, varying from negative to positive.

In a paper read before the Engineers' Society of Western Pennsylvania, at Pittsburgh, Pa., October 18, 1887, by Jos. Ramsey, Jr., Chief Engineer of the C. H. & D. R. R., a series of tests, showing the effect of temperature ranging from 0° Fahr. to 60° Fahr., was discussed, and the conclusion arrived at was that the effect of freezing was to decrease the ductility *under impact*, making the iron more crystalline and brittle. This conclusion was combated by several of the members, and other tests were brought forward to show that the effect was so slight as to be practically *nil*. Engineers are, however, pretty well agreed that bending- and nicked-bending tests should not be made on iron that has been exposed to a temperature below freezing, without first allowing it to become warm, reaching, say, a temperature of about 60° Fahr., or, as the mill men express it, "allow the frost to get out of it."

About two years ago an instance came under our observation of some 6-inch channels that had been "lying out in the cold" over night, at a temperature of probably 10° to 15° Fahr., which, when struck with a hammer on the web, split up from 6 inches to 18 inches. While subsequent tests on the same channels showed that the material was not strictly first-class, they also showed that the iron, when warmed, developed from 50 to 75 per cent. fiber, whereas the cracked channels were 75 to 100 per cent. crystalline, as was also

shown by further nicked-bending tests made at the cold temperature. Instances of this kind would tend to show a molecular change due to cold, and a change great enough to affect seriously the strength of the iron if exposed to sudden shock at low temperatures, as in the case of the derailment of a train on a bridge.

From a consideration of such facts as these, our inspectors have for the last three or four years been making all bending- and nicked-bending tests at a temperature of 60° Fahr., unless otherwise directed, proceeding on the assumption that during the greater portion of the year, in this latitude, the temperature is above the freezing-point—the average being 52° Fahr.—and that a uniform temperature is necessary in order to get comparative results of the fibrous quality of wrought-iron. We also recommend to engineers that where the iron will be subject to extremely low temperatures, a series of fracture- and bending-tests should be made at freezing temperatures, and that reasonably ductile results under such treatment should be required.

The effect of temperature upon tensile and compression-tests is not at all so marked as it is on tests under impact, for with whatever rapidity the strains have been put upon specimens by tensile or compressive apparatus which can measure the strains applied, the work and flow of the metal has so increased the temperature of the parts as not seriously to modify the results at ordinary temperatures, even though the experimental bars be immersed in freezing-mixtures while being tested.

Planing all sheared edges in structural work is being required more and more in this country, and for this purpose many of the larger steel-works are procuring fast rotary planers, but the average structural mill is not equipped as finely as are the large structural mills in Great Britain. This probably is due to the fact that for many purposes we use universal-mill plates, and there has not been the necessity for planing so many sheared plates here as abroad.

Steel pins are required to be carefully annealed after being rolled and before being turned up, and pins of over 6 inches in diameter are in most cases drilled through their longer axis with holes from $\frac{1}{2}$-inch to $1\frac{1}{2}$-inch diameter, with the intention of testing the soundness through the entire length. Test-specimens for pins are ordinarily cut from the ends of blooms which have been forged into sizes convenient for the purpose.

Numerous instances could be cited to show the unreliability of Bessemer steel for structural purposes. One of the most marked, however, that has come under my notice will suffice.

A 12-inch I-beam weighing 30 pounds to the foot, 20 feet long, which had been rolled of a new section to replace 42-pound iron beams, on being unloaded from a car, broke in two about 6 feet from one end. The Bridge Company wrote about the accident as follows:

"In unloading the 12-inch beams, one about 20 feet in length, in dropping, broke completely in two, about one-third the distance from one end. There is apparently no defect in the beam at this point, nor anything to account for the breakage. The chemical analysis and tests appear good. We are at a loss to account for the breakage, and it shakes our confidence in the whole subject of steel beams."

The analysis of the test-ingot of the Bessemer blow from which the beam was made was as follows:

	Per cent.
Carbon,	0.18
Manganese,	0.60
Phosphorus,	0.072

We directed that the broken piece be shipped to us, and on receiving it, found the fracture to be exactly at right-angles to the flange, and almost as true as if it had been cut in a planer. Although rusty, the fracture appeared to be finely crystalline, and showed no evidence of any flaw which would have caused the break.

Drillings taken from various points in the fractured surface showed:

	Per cent.			Per cent.	
No. 1.		Flange.	No. 4.		Web.
Carbon,	0.18		Carbon,	0.18	
Manganese,	0.68		Manganese,	0.68	
No. 2.			No. 5.		
Carbon,	0.18		Carbon,	0.18	
Manganese,	0.67		Manganese,	0.67	
No. 3.			No. 6.		
Carbon,	0.18		Carbon,	0.17	
Manganese,	0.68		Manganese,	0.67	

	Per cent.	
No. 7.		Flange.
Carbon,	0.17	
Manganese,	0.69	
No. 8.		
Carbon,	0.17	
Manganese,	0.68	
No. 9.		
Carbon,	0.17	
Manganese,	0.68	

An average sample of the nine sets of drillings gave:

	Per cent.
Carbon,	0.18
Manganese,	0.68
Phosphorus,	0.09
Sulphur,	0.033

Tensile test of the original $\frac{3}{4}$-inch round test-bar gave:

Elastic limit,	39,160 lbs. per sq. in.
Tensile strength,	68,360 " "
Elongation in 8 inches,	29.25 per cent.
Reduction of area,	52.86 "
Character of fracture,	Silky, cupped.

Of the beam cut as close to the broken end as possible:

	Flange.	Web.	Flange.
Elastic limit,	40,580 lbs. per sq. in.	45,240 lbs. per sq. in.	41,660 lbs. per sq. in.
Tensile strength,	62,140 " "	68,540 " "	64,110 " "
Elongation in 8 inches,	23.75 per cent.	23.62 per cent.	27.50 per cent.
Reduction of area,	53.38 "	56.86 "	51.79 "
Fracture,	Silky.	Silky, cupped.	Silky, pitted.

The analyses and tensile tests made do not show any cause for the failure.

The cold- and quench-bending tests of both the original $\frac{3}{4}$-inch round test-pieces, and of pieces cut from the finished material, gave satisfactory results; the cold-bending tests closing down on themselves without sign of fracture.

Numerous other cases of angles and plates that were so hard in places as to break off short in punching, or, what was worse, to break the punches, have come under our observation, and although makers of Bessemer steel claim that this is just as likely to occur in open-hearth as in Bessemer steel, we have, as yet, never seen an instance of failure of this kind in open-hearth steel having such a composition as that given below:

	Per cent.
Carbon,	below .25
Manganese,	" .70
Phosphorus,	" .08

Mr. J. W. Wailes, in a paper read before the Chemical Section of the British Association for the Advancement of Science, in speaking

of mysterious failures of steel, states that investigation shows that "these failures occur in steel of one class, viz., soft steel made by the Bessemer process."

Bessemer steel-makers may claim that in a general paper of this kind it is unfair to criticize their process. There is some justice in this; but at the same time it seems proper to state the reasons for the growing tendency among engineers to rule out Bessemer steel for tension-members and all of the more important structural purposes. This tendency, we think, is fully as strong in Europe as in America, and the report of the Secretary of this Institute, and of the Iron and Steel Association of the United States, in their records of the statistical progress of new steel-plants and processes, show a most remarkable growth of open-hearth furnace-building.

THE DEVELOPMENT OF AMERICAN BLAST-FURNACES, WITH SPECIAL REFERENCE TO LARGE YIELDS.

BY JAMES GAYLEY, BRADDOCK, PA.

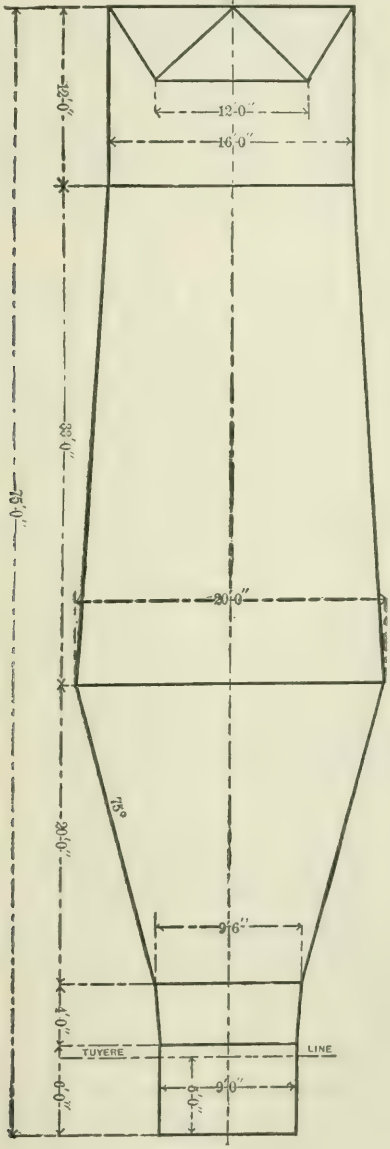
(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

THE development of blast-furnace practice in America in the direction of large yields is mainly the history of our working since the year 1880, as the advancement that has been made in the last decade is greater than that in the third of a century previous. A new era in the manufacture of pig-iron began in 1880 with the putting in blast of the Edgar Thomson furnaces. These furnaces at once leaped to the front as pig-iron producers, and have maintained that position—with but one brief interruption—ever since. I shall, therefore, confine myself mainly to a description of these works, showing the changes in design and practice by which these results have been achieved.

In order to show more clearly the progress that has been made since 1880, I shall refer briefly to the best work that was done in the ten years previous to that time. The Struthers furnace, in Ohio, was one of the first to attract attention in the matter of large outputs. This furnace was 55 feet high, 16 feet in diameter of bosh, 9 feet in the hearth, and about 8 feet 6 inches at the stock-line. The fuel was raw coal. This furnace, in December, 1871, made 1602

tons* of iron, and in January, 1872, made 1642 tons. The best out-

Fig. 1.



Lucy Furnace, No. 2. 1877.

put in a single week was 400 tons. By March, 1876, the product

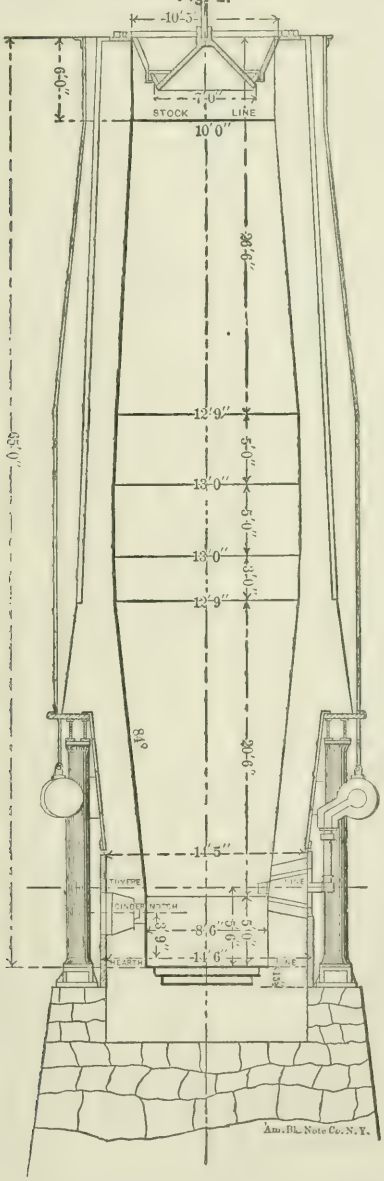
* All tons are long tons of 2240 pounds, and all temperatures are Fahrenheit.

had been increased to 2032 tons. The furnace was blown by one engine with an air-cylinder of 72 inches diameter and 4 feet stroke. The only change made from 1871 to 1876 was the addition of a second blowing-engine. The size of the hearth was much in advance of the construction usual at that time, and was, in fact, as large as would now be put in a furnace of the same size. The good results obtained at this furnace were largely due to the fact that the manager, Mr. Thomas W. Kennedy, broke away from the traditional practice of regulating the quantity of blast by the pressure-gauge, and substituted instead the revolutions of the engine—a practice which obtains to-day.

A record of No. 1 furnace of the Isabella furnaces, located at Pittsburgh, has been furnished me, and, as these furnaces have since become famous in the annals of pig-iron manufacture, a return of their work at that time will be found interesting. I have not been able to secure the lines or detail construction of this furnace, but the general dimensions were: Height, 75 feet; diameter of bosh, 20 feet; cubic capacity, about 15,000 feet. The furnace was "blown in" in January, 1876, and continued in blast until May, 1880—making a total output of 117,575 tons of pig-iron, or an average of 2264 tons per month. The consumption of coke per ton of iron averaged 3000 pounds; the temperature of the blast ranged from 1000 to 1100 degrees, with a blast-pressure of 4 to 6 pounds. No record was kept of the amount of air blown. Also located in Pittsburgh are the Lucy furnaces, owned by the firm of Carnegie, Phipps & Co. These furnaces ranked with the Isabella as the largest producers in America. I have been so fortunate as to obtain the lines of one of these furnaces, the Lucy No. 2, built in 1877, which I show in Fig. 1. The general dimensions are: Total height, 75 feet; diameter of bosh, 20 feet; diameter of hearth, 9 feet; cubic capacity, 15,400 feet. This drawing shows a bell 12 feet in diameter, but Mr. H. M. Curry, who was then manager of the works, advises me that the bell generally in use was 11 feet in diameter. During a part of the blast a bell 12 feet in diameter was tried, but, not proving to be advantageous, was abandoned. In the construction of this furnace the noticeable features are a narrower hearth and a wider top than are now put in furnaces of the same cubic capacity; but at that time it was considered an excellent shape, and certainly did produce some excellent results. As early as 1878 this furnace had made a monthly output of 3286 tons on a coke-consumption of 2793 pounds per ton of iron, and in one week shortly afterwards it made 821 tons. The

furnace was "blown in" in September, 1877, but after running a

Fig. 2.



Edgar Thomson Furnace "A." 1879.

few months was blown out, in order to repair the bosh-walls, which, for some reason, had rapidly given away. In the following March

the furnace was again started, and continued in blast for nearly thirty-four months, making in this period an output of 92,128 tons, or an average of 2737 tons per month, on an average coke-consumption of 2865 pounds per ton of iron. For the first twelve full months the output was 33,552 tons on a coke-consumption of 2850 pounds. The amount of air blown was 16,000 cubic feet per minute, which entered the furnace through six 8-inch tuyeres; the temperature of the blast was 915 degrees, and the pressure at tuyeres 5 pounds. The ore-mixture yielded in the furnace 60 per cent. of iron. The work at this furnace was unquestionably the best, all things considered, that had been accomplished prior to the starting of the Edgar Thomson furnaces.

Furnace "A" of the Edgar Thomson works was originally a charcoal furnace at Escanaba, Mich., but was removed and erected on its new foundations in 1879. The dimensions of this furnace, Fig. 2, are as follows: Height, 65 feet; diameter of bosh, 13 feet; diameter of hearth, 8 feet 6 inches; cubic capacity, 6396 feet. Six tuyeres, 4 inches in diameter, were used; and as they projected 7 inches inside the crucible, the efficient diameter of hearth was 7 feet 4 inches. The tuyeres were placed 5 feet 6 inches above the hearth line. In the *Transactions of the American Institute of Mining Engineers* (viii., 348), a description of the "blowing in" and subsequent working of this furnace is given by Mr. Julian Kennedy, who was then manager of the furnaces. The following points of construction are emphasized in his paper: The interior lines make very small angles with each other,—so small, in fact, that the arc of a circle drawn from the top to the tuyeres will not deviate by more than 2 inches from the lines as given. Particular attention was paid to rounding the angles. The bosh is located about midway in the furnace, making the bosh-wall very steep. The batter of this wall was $1\frac{3}{8}$ inch to the foot, which is equivalent to an angle of 84 degrees. The furnace was lined throughout with small bricks. The stove equipment consisted of three Siemens-Cowper-Cochrane stoves, 15 feet in diameter by 50 feet in height. This furnace was "blown in" in January, 1880. The ore-mixture consisted of Tafna, M'Comber, Pilot Knob and Sommorostro ores, yielding in the furnace 54.5 per cent. iron. The output of the first full week was 442 tons, and reached 537 tons for the fourth week. The output and consumption of coke for the first few months are as follows:

TABLE I.—*Showing the Working of Furnace "A," Fig. 2.*

Months.	Tons.	Pounds of Coke per Ton of Iron.
January, 1880.....	1,634	2.675
February, "	2,236	2,306
March, "	2,762	2,306
April, "	1,586	2,482*
May, "	2,226	1,945

* The furnace was stopped for four days.

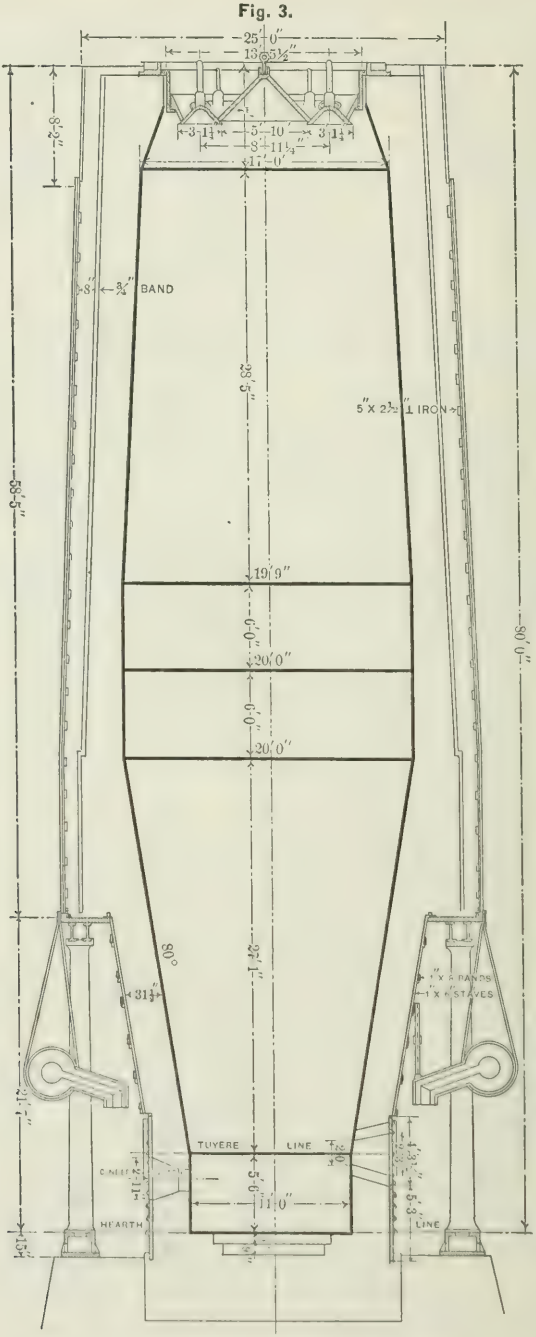
In June, the furnace was changed to making spiegel, and has been used almost exclusively ever since for making the various grades of spiegel and ferro-manganese. The average weekly output for the month of March was 644½ tons, one day having been lost.

The best week's output was 671 tons. The blast was heated to an average temperature of 1050 degrees, the utmost that the stoves would furnish: the pressure at the tuyeres was 6¼ pounds. The engine equipment consisted of two vertical blowing-engines, one of which was new, with a blowing tub 84 inches in diameter and of 4 feet stroke; the other, a small and inefficient one, had been brought from Escanaba. But it so happened that Furnace "B" was in course of construction, and one of the engines belonging to this furnace was in good running order; consequently, the Escanaba engine was retired from active service, and these two larger engines were used to supply Furnace "A."

The volume of air forced into this furnace was 15,000 cubic feet per minute, or as much as was used elsewhere for furnaces of more than twice the capacity. The results obtained were surprising, and it is no wonder that the pig-iron-making world regarded them with an astonishment amounting, in many cases, to incredulity. Considering the cubic capacity of the furnace, the rate of driving was certainly excessive, and that the results on fuel were so low, as compared with the subsequent consumption on larger furnaces where the same practice was employed, is mainly due to the narrow furnace-stack. These fuel-results, as I shall presently show, were much lower than any obtained from the larger furnaces in the next five years, and, had a distinction been then made between rapid driving on the one hand and excessive driving on the other, many American furnaces would have reduced their outlay for fuel enormously.

In Fig. 3 is shown the second furnace, "B," erected at these works, of which the general dimensions are as follows: Height, 80 feet; diameter of bosh, 20 feet; diameter of hearth, 11 feet; cubic

Fig. 3.



Edgar Thomson Furnace "B." 1880.

capacity, 17,868 feet. The brick-work of the stack was held together by 8-inch iron bands supported by staves of T-iron, forming a crinoline. The bosh-walls were banded in the same manner. The walls of the hearth were surrounded by solid cast-iron plates, securely bolted, no cooling-plates of any kind being used. The stock was distributed at the top by a double bell, in which the central cone remained stationary, while the outer conical ring was lowered, casting the stock towards the wall and center of the furnace. One feature of this construction, differing from that of other furnaces then using coke for fuel, was the large hearth, providing more space for combustion. As originally planned, the hearth was 9 feet in diameter at the bottom, and sloping up to the tuyeres, but, before building, Mr. Kennedy changed the plan, so that the in-walls of the hearth would be straight, making the diameter 11 feet. There was, moreover, an increased number of tuyeres, eight being used, and an increased elevation of the tuyeres above the hearth-level, all of which modifications were necessary for rapid driving and large yields. In the anthracite district large hearths had been tried at a much earlier day. As early as 1854 the Thomases, at the Crane Iron Works, in the Lehigh Valley, had tried enlarging their hearths by pulling back the tuyeres until a diameter of 11 feet was reached. From this experiment they obtained such excellent results, both as to output and quality of metal, that they afterwards adopted as a standard that the diameter of the hearth should be one-half the diameter of the bosh. But while the hearths of various furnaces had been enlarged after they were in blast, yet no American furnace up to that time had been constructed with so large a hearth as this one at the Edgar Thomson works. In another respect this furnace was well prepared by its designers for a high productive capacity, viz., in its equipment. Fire-brick stoves of the most approved type were erected. Substantially-built blowing-engines were provided, and they were rendered efficient by an ample supply of boilers—a point in which other furnaces were then sadly lacking. At the same time, all the flues and mains were constructed sufficiently large, and in the most substantial way. In fact, no furnace previously erected had been planned on such a liberal basis; consequently, large yields were to be expected.

The furnace was put in blast in April, 1880. In the following month an output of 3718 tons was made, and the next month showed 4318 tons; thus fully justifying the claims of its designers by eclipsing all previous records. The detailed record of the work

done by this furnace during the earlier months of the blast is as follows:

TABLE II.—*Showing the Working of Furnace "B," Fig. 3.*

Months.	Tons.	Pounds of Coke per Ton of Iron.
April, 1880.....	2,723	2,536
May, "	3,718	2.574
June, "	4,318	2,344
July, "	4,345	2,706
August, "	4,601	2,811
September, "	4,221	2,757
October, "	4.722	2,736

It will be noticed that the consumption of fuel increases as the summer months advance. These works, being located on flat land along the Monongahela River, are more particularly affected than elsewhere by the contents of moisture in the atmosphere, it being nearly twice as much in summer as in winter. While this fact has always had an important bearing on our fuel-economy, yet in this case another feature of the practice tending to increase the fuel-consumption was the increased rate of driving; the volume of air blown having been gradually increased to 30,000 cubic feet per minute, engine measurement, taxing alike the efficiency of the engines and the stoves. However, a high productive capacity was aimed at, and an output of 4722 tons in a month was certainly marvellous at that time, and was either regarded as incredible or accounted for on the supposition that the furnace had been transformed into a cupola by melting large quantities of scrap. But the furnace-records show that the yield of ore-mixture was 55 per cent. iron. The weight of limestone was 25 per cent. of the weight of the ore. An analysis of the cinder showed:

	Per cent.
Silica, ...	32.31
Alumina, ...	13.20

The limestone contained a very small quantity of magnesia. The blast entered the furnace through eight bronze tuyeres of 5½ inches diameter, and was heated to a temperature of 1100 degrees. The silicon in the iron averaged about 2 per cent. The rapid wear of the furnace walls, through the use of such a large volume of air, gradually increased the consumption of coke to over 3000 pounds per ton of iron. At the end of the first twelve full months the out-

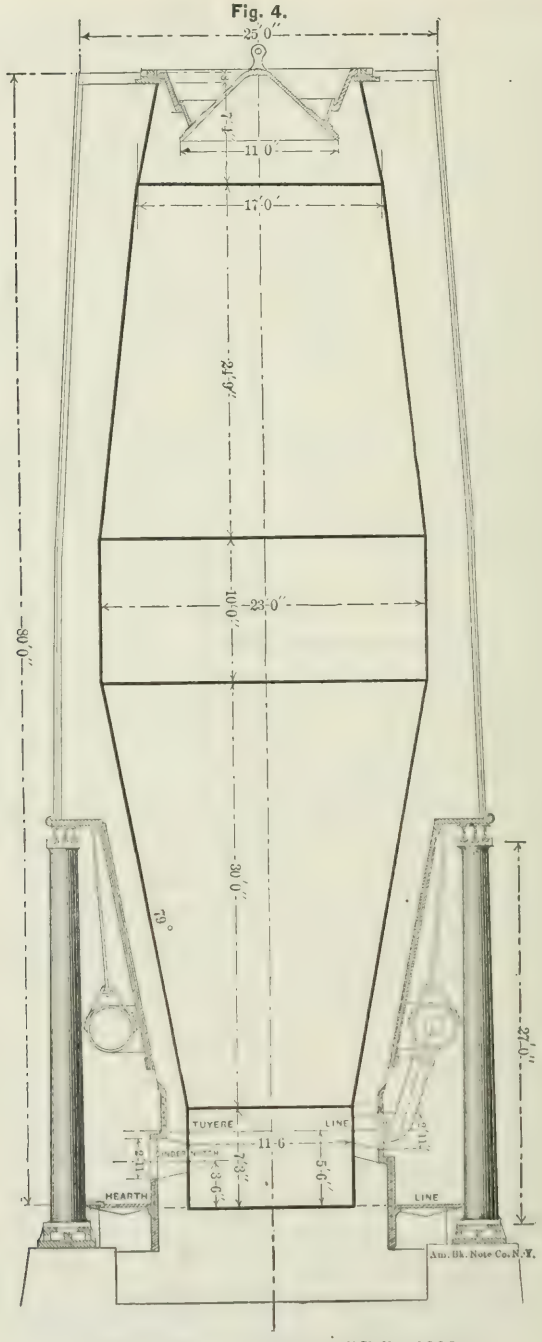
put was 48,179 tons, on an average coke consumption of 2859 pounds per ton of iron. The second year showed an average consumption of 3200 pounds of coke, with a decrease in yield. The upper brick of the stack having given way, the furnace was blown out after a blast of two years and five months, having made a total product of 112,060 tons, on an average coke consumption of 3149 pounds per ton of iron.

The results obtained in this blast determined several important changes in construction. It was seen that the crinoline structure afforded inadequate protection to the brick-work of the stack, in consequence of which it was torn down and replaced by an iron jacket; also, that the bosh-walls should be protected so as to preserve as far as possible the original lines, and the hearth be surrounded with water-cooled plates. The double bell was also found to possess no special advantage, and was abandoned. These changes were introduced in the furnace next erected at the works. But, before leaving this furnace, I wish to emphasize the bearing that the practice of rapid driving, begun on Furnace "A," and further developed on this one, had on the general practice of this country. The large outputs obtained from this furnace by the use of a large volume of air, was a matter of common knowledge; the practice of fast driving soon became the accepted one, and with our national ardor it was prosecuted enthusiastically. In every direction, engines that had been running along for years at a methodical gait, were oiled up and started off at a livelier pace; new boilers were added; the old iron hot-blast stoves, not supplying sufficient heat, were torn down and replaced by the more efficient fire-brick stoves. At many works rapid driving degenerated into excessive driving. True, the outputs increased; so also did the consumption of fuel, and that at a surprising rate, until it was thought well-nigh impossible to produce a ton of iron with 2600 pounds of coke.

Mr E. C. Potter, of Chicago, sounded the note of warning in 1885, and great credit is due to him for demonstrating that large outputs and low fuel-consumption are compatible. Since then a marked change in our practice has taken place, and the volume of air blown has been greatly diminished.

Although the practice of rapid driving has been much decried, yet in many ways it has resulted beneficially. It has brought in an equipment of hot-blast stoves, boilers, engines, etc., sufficient to accomplish a large amount of work without a constant strain on every part—a condition very rare prior to 1880; and it has also

Fig. 4.



Edgar Thomson Furnace "D." 1882.

developed a construction of the furnace-stack, by which larger outputs from a single lining can be obtained, with less irregularity in the working.

Fig. 4 shows a furnace of different construction from either of the preceding, it being my purpose to present successive types of construction rather than changes in the same furnace. We find this furnace constructed with special regard to the better protection of the brick-work of hearth and bosh. The general dimensions are as follows: Height, 80 feet; diameter of bosh, 23 feet; diameter of hearth, 11 feet 6 inches; stock-line, 17 feet; bell, 11 feet; cubic capacity, 21,478 feet. The bosh is placed at about the center of the stack, making very steep walls. The hearth is also made wider by 6 inches than in furnaces previously described. The hearth-walls are surrounded by cast-iron plates with a coil inside for the circulation of water. Around the bottom of these plates is a gutter, through which waste-water from the cooling-plates flowed, affording better protection to the bottom of the hearth. Above this row of plates, at the tuyere-breasts, is another circle of cooling-plates, partially inserted in the brick-work. The walls of the bosh are encased in a jacket of wrought-iron, $\frac{1}{2}$ inch thick. This jacket is bolted to the mantle. The bosh-walls inside of the jacket were made but $22\frac{1}{2}$ inches thick, so that the cooling effect of the air-currents on the jacket would prevent any very rapid wear of the brick-work.

It is a fact well known in our practice that the best records are obtained within the first year of a blast, as the walls are not then affected to any great extent by wear. With the better protection afforded by more recent construction, this difference has not been so marked. This furnace was put in blast in 1882, and in the second month, made the large yield of 6045 tons, on a coke consumption of 2617 pounds. In the first twelve full months the output was 65,947 tons, on an average of 2570 pounds of coke per ton of iron, thus exceeding, by over 11,000 tons, the best output that had previously been obtained in the same time from any furnace at these works, and with a much smaller consumption of fuel. The record for the best month during this period was 6131 tons, on a coke consumption of 2387 pounds per ton of iron. The amount of air blown was 27,000 cubic feet per minute, which was heated to an average temperature of 1000 degrees. The pressure of blast at the tuyeres varied between 9 and 10 pounds. After a blast of seventeen months' duration this furnace was blown out, having made a total

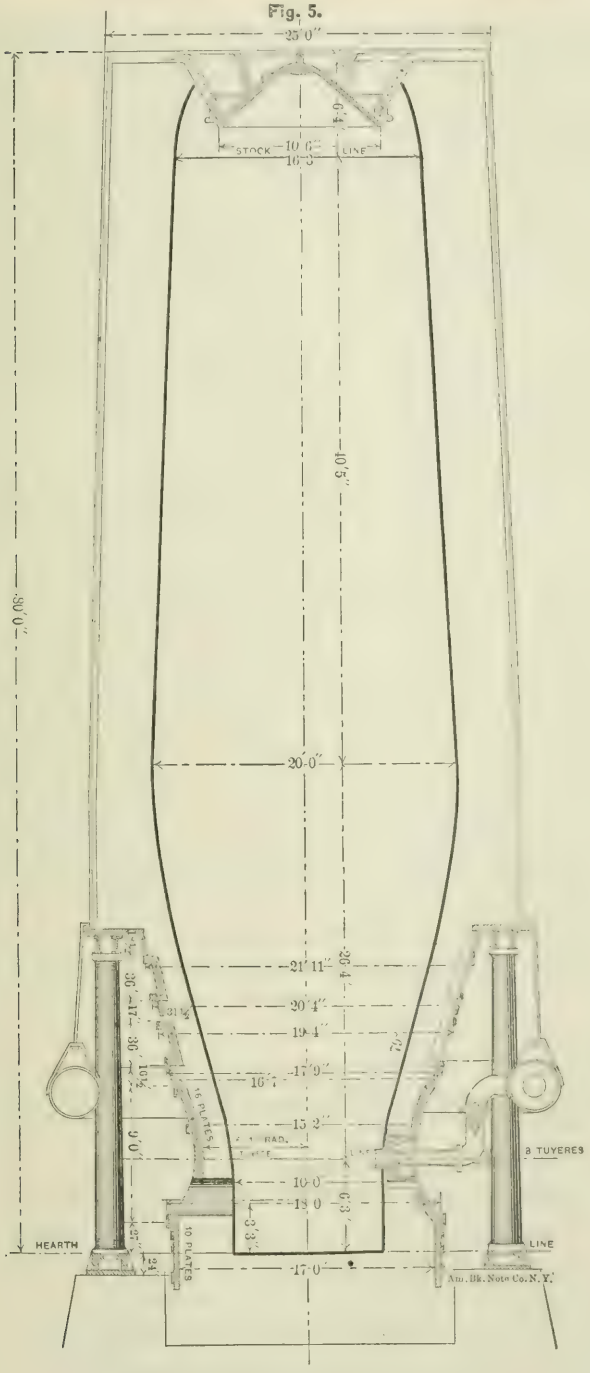
output of 90,317 tons, on an average coke consumption of 2613 pounds per ton of iron.

For what reason this furnace was blown out I do not know, since the consumption of fuel, even in the last two months of the run, did not increase more than 100 pounds above the average. The furnace-records afford no explanation of the cause. As a matter of fact, the results were much better than any obtained for a number of years afterwards. They serve, also, to show how very materially a slight protection afforded to the furnace-walls affects the output and the consumption of fuel.

As illustrating the effects of an excessive volume of air, I may mention a furnace put in blast two years later—of practically the same dimensions and protected to the same extent, but blown with 33,000 cubic feet of air per minute—which made an average output of 5000 tons per month, on a coke consumption of 3000 pounds per ton of iron. This is the largest amount of air that I know to have been forced into any furnace. The fuel consumption is what might have been expected, as the furnace was driven at such a rapid rate that the imperfect reduction of the ores in its cooler parts necessitated the completion of the reduction at a temperature sufficiently high for the CO_2 to react on the carbon.

In the next drawing, Fig. 5, is shown a different arrangement for maintaining the bosh-walls. The lower part of the hearth is surrounded by cast-iron plates. On top of these rests a water-cooled corset-jacket, fastened together with links, and banded at the upper and lower ends by iron bands, 6 inches wide and 1 inch thick. The wrought-iron bosh-jacket is dispensed with; and, in its stead, are two bands of water-cooled plates, covering the greater portion of the brick-work. These are held in place by bands at the top and bottom, and also by being inserted a short distance into the brick-work. The tuyere-fittings are unique. The arrangement, as shown in the drawing, consists, first, of a large oval block, made of cast-iron, and water-cooled. Fitting into this, with sufficient space left for a clay packing, is a large bronze cooler, oval-shaped at the outer end, but with a circular opening at the inner end for the insertion of the tuyeres. This cooler was supported by trunnions resting on the block, by means of which the nose could be either elevated or depressed. The tuyeres were shaped to make a ball-joint connection with the cooler, so that a horizontal delivery of the blast could be obtained with any inclination of the cooler. The adjustment was such that the tuyeres were placed in a staggered position, there being as much as 6 inches be-

Fig. 5.



Edgar Thomson Furnace "C." 1885.

tween the center-lines of any two adjacent tuyeres. The reason given for thus placing the tuyeres was "to obtain a greater area of contact between the fuel and the air." No such benefit, however, was derived, and the tuyeres were subsequently placed on the same level.

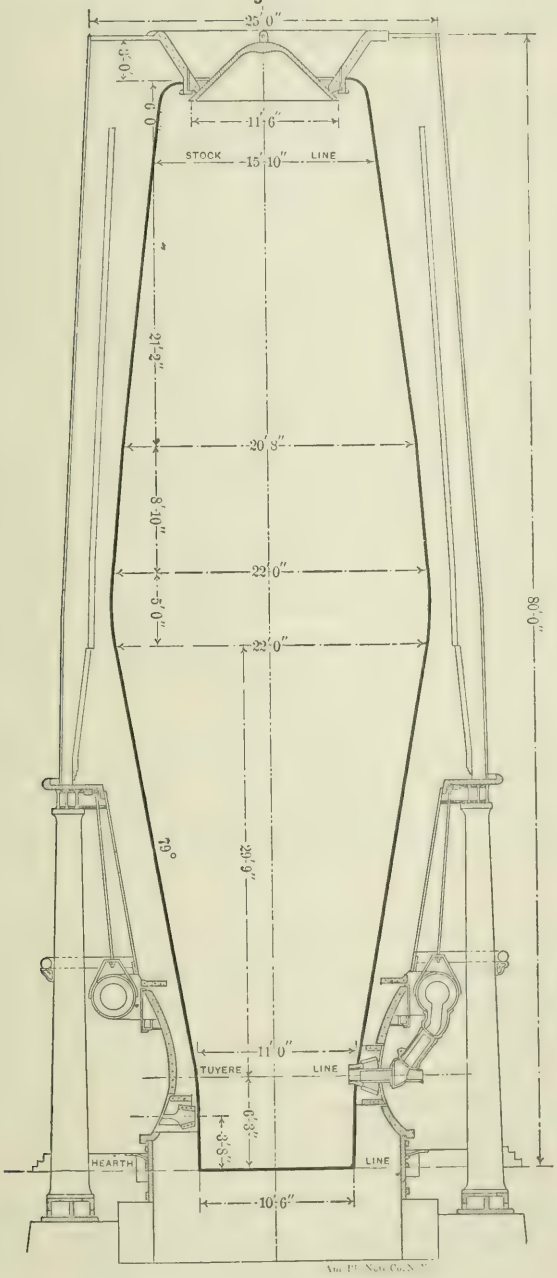
The dimensions of the furnace are shown in the drawing. The cubic capacity was 16,680 feet. In February, 1885, the furnace was "blown in." The volume of blast was rapidly increased until, in the following month, it reached 31,000 cubic feet per minute. The blast entered the furnace through eight tuyeres, 7 inches in diameter, and was heated to an average temperature of 1200 degrees. The pressure at the tuyeres was $8\frac{1}{2}$ pounds. The average monthly output from March to August inclusive was 5122 tons on a coke consumption of 2874 pounds per ton of iron.

As this was the third blast for the same furnace (a companion furnace to that shown in Fig. 3), a comparison with the record made at the same relative time on the first blast will show the effect of different rates of driving on the fuel-economy. In the first blast the coke-consumption was 2600 pounds, with a volume of air of 24,000 cubic feet per minute; in this blast it has risen to 2850 pounds, through an increase in the volume of air of 7000 cubic feet.

On taking charge of these works in October, 1885, I endeavored to reduce the volume of blast in order to diminish the consumption of fuel. The reductions were made slowly, until the volume was 28,000 cubic feet per minute. Many attempts were made to reduce it still further, but as each attempt was invariably met by great irregularity in the working of the furnace, further efforts in that direction were abandoned. The volume of blast was therefore maintained steadily at 28,000 cubic feet, and the temperature, which before had been quite variable, was maintained more uniformly. At the same time the cinder, which was quite basic, was made less so.

As a result of these changes the output increased to an average of 6050 tons per month, on a coke-consumption of 2400 pounds per ton of iron; but after a three months' trial the furnace was suddenly shut down, with a full working burden, for a period of 21 days, on account of labor troubles. After it had resumed, and the bad working incident to such a shut-down had been remedied, the furnace made an average monthly output of 5700 tons; but the consumption of coke had advanced to 2600 pounds, and continued at that figure till the end of the blast. On account of the upper lining being destroyed, the furnace was blown out after a blast of twenty-three months.

Fig. 6.



Edgar Thomson Furnace "D." 1885.

The total output for the blast was 118,000 tons, on a coke-consump-

tion of 2705 pounds. For the first twelve full months the output was 64,998 tons, on a coke-consumption of 2677 pounds per ton of iron.

In Fig. 6 is shown a combination of Figs. 4 and 5 in the construction of the cooling-jackets of Furnace "D" in 1885. From the top of the corset-jacket down, the construction is an exact duplicate of Fig. 5, just described. But between the corset and wrought-iron jackets a row of water-cooled iron plates is inserted in the brickwork. There are eight sections in the circle, all being connected together, and the water was fed from the high-pressure main. The advantage of a high-water pressure on this kind of plate is shown by the fact that not a single one was lost during the entire blast.

The interior shape of this furnace differs very materially from those previously shown, having been made to conform, as nearly as possible, with a furnace at the South Chicago works, which was then giving excellent results in regard to both output and consumption of fuel. The bell is increased in diameter, and the upper walls are drawn in, making a less diameter at the stock line. The bosh is 22 feet in diameter, the total height 80 feet and the cubic capacity 18,950. There are eight tuyeres, each 5 inches in diameter.

This furnace was "blown in" in September, 1885, and in one week from starting, the volume of blast was 27,000 cubic feet per minute, and was at this figure when the furnace came under my management. Within the next two months I reduced this volume to 22,000 cubic feet, which was heated to a temperature of 1300 degrees.

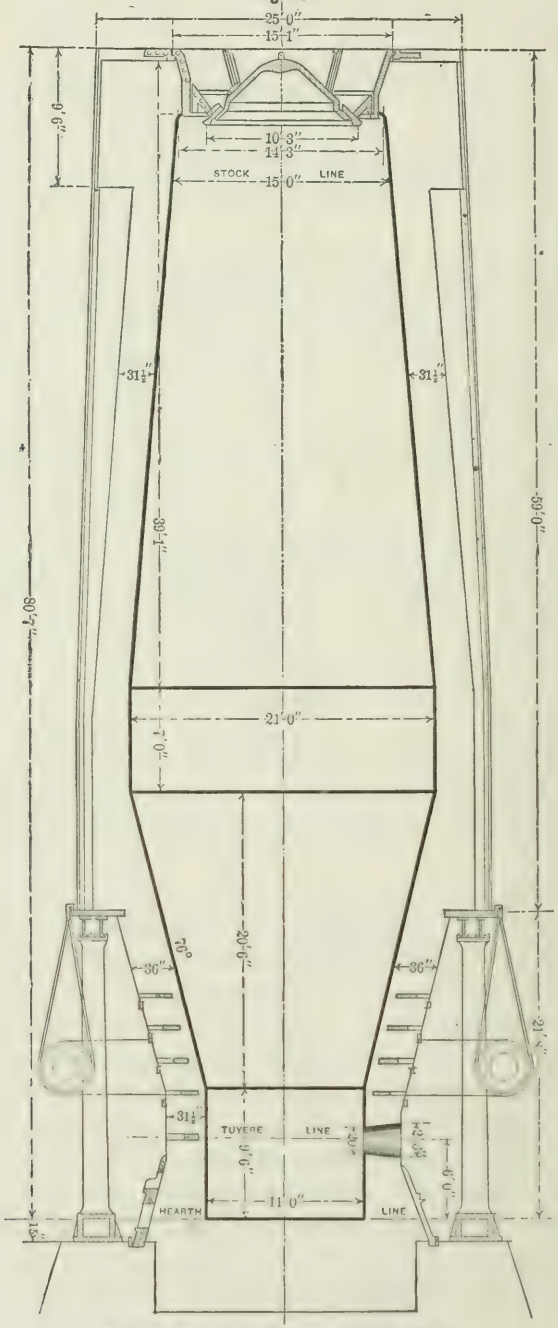
The product for October was 6320 tons, and for November 6306 tons, on a coke-consumption of 2396 pounds. In December, the output on the decreased volume of blast increased to 6451 tons, and the consumption of coke had fallen to 2172 pounds. During the months of January and February, 1886, we were unfortunately compelled to shut down for thirty-three days. In the month of March the output was 6352 tons, on a coke-consumption of 2105 pounds. For the first full twelve months the output was 74,475 tons, on an average coke-consumption of 2250 pounds per ton of iron. During the last year that the furnace was in blast, the volume of air was gradually increased to 27,000 cubic feet per minute. The inwalls were much worn, and the grade of iron had become variable; but with the increased volume of air, the working

became very uniform. In the last 8 months of the blast, the consumption of coke had risen to 2500 pounds. After a blast of twenty-six months, the furnace was blown out, having made a total output of 150,374 tons on an average fuel-consumption of 2342 pounds per ton of iron. At no time in the blast did the coke consumption reach 2600 pounds, except in the third month before blowing out. The best output obtained in any single month was 6730 tons.

While these records were much better than any previous ones at the same works, it was evident that had the bosh-walls been entirely protected with cooling-plates, still better results would have been obtained. When exterior cooling-jackets only are used, the preserving of the furnace lines depends wholly upon the carbon coating which is deposited upon the brick-work as the latter wears away; in fact, were it not for this protection, the bricks would last but a very short time indeed. When this coating increases in thickness, the furnace works far more regularly and economically; and in a furnace considerably worn, the thickness of this carbon wall is to a great extent the measure of its economical working. We have repaired bosh-walls where only 6 inches of the brick-work remained; but inside was found this coating, 18 to 20 inches thick, firmly set and far more difficult to penetrate with a drill than the firmest brick-work. But while the thickness of this coating may be increased at will, yet it can only be done at the expense of fuel; and so unstable is the formation, that the expensive operation of building up—extending over months—may be dissipated in a week or two, by running the furnace on an acid cinder.

Fig. 7 shows the same furnace as Fig. 5, with the bosh entirely protected by cooling-plates inserted in the brick-work. A comparison of the two drawings will show the changes in construction. The hearth has been widened to 11 feet diameter. The diameter of bosh is also increased 1 foot, and the stock-line is reduced 15 inches. The hearth is protected by water-cooled plates, which by their slanting position, admit a greater thickness of brick-work at the bottom. These plates are practically automatic, as under an expansion of the brick-work they slide up. In some cases they have raised as much as 3 inches. In the bosh are 4 rows of bronze cooling-plates, eight plates forming the circle, and each plate having two water-courses. In the two lower rows each plate is fed separately; in the upper rows, two plates are connected together. We have found, from quite a number of tests, that the water

Fig. 7.



Edgar Thomson Furnace "C." 1887.

circulating through these plates increased 10 degrees in temperature. Cast-iron cooling-plates are also placed between the tuyeres, the number of the latter being reduced to 7, each 6 inches in diameter. The cubic capacity of the furnace is 17,230 feet.

This furnace was "blown in" in March, 1887, and in April made 5591 tons, on a coke consumption of 2105 pounds. In May and June the output was small, a stoppage occurring on account of a shortage in the coke-supply. In July the output increased to 6243 tons, and in September reached 7064 tons, on a coke-consumption of 1993 pounds. The volume of air blown was 24,000 cubic feet per minute, heated to an average temperature of 1200 degrees. This furnace continued to do excellent work until the following December, when a stoppage of $4\frac{1}{2}$ months occurred, pending an adjustment of the wage-question. In the nine months that the furnace had been in blast previous to this stop—and including the stoppage on account of coke-shortage—the fuel-consumption did not exceed 2160 pounds per ton of iron. In May operations were resumed. While the output soon reached the former figures, yet the coke-consumption was rarely reduced for any one month—even to the end of the blast—below 2300 pounds. In the period extending from October, 1888, to January, 1890, the average monthly output was 7000 tons, on a coke-consumption of 2350 pounds. During this period the volume of air had been increased to 27,000 cubic feet per minute. The temperature of blast was still maintained at 1200 degrees. The average pressure of blast at the tuyeres was 8 pounds. On account of the brick-work in the bosh being very much worn, the furnace was blown out after a run of 2 years, 7 months and 17 days—exclusive of the time the furnace was banked. The output for the blast was 203,050 tons, on an average coke-consumption of 2342 pounds per ton of iron. The output for the first twelve full months was 72,554 tons, on a coke-consumption of 2230 pounds. For the second twelve months, during which no stoppage occurred, the output was 83,219 tons. The best output made in any one month was 7680 tons.

It will be seen by the preceding description that the results both in output and economy of fuel have been materially affected by the long stops that the furnace was subjected to. Under a continuous run, much better results would have been attained.

The advantages of this system of cooling the bosh were very marked, and although quite a number of plates were lost from vari-

ous causes during the blast, yet when a new one was inserted, the furnace invariably showed an improvement in its working.

In the next drawing, Fig. 8, is seen the same construction of hearth and bosh applied to one of our larger furnaces. In fact, the furnace just described was remodelled after this one; but for the purpose of comparing two consecutive blasts on the latter I have placed it earlier. This furnace was built in 1885–86. The total height is 80 feet; the diameter of hearth, 11 feet; the diameter of bosh, 23 feet. The bell is 12 feet in diameter, and the stock-line 16 feet. The cubic capacity is 19,800 feet. The construction of hearth and bosh is the same as in Fig. 7. There are 7 tuyeres, each 6 inches in diameter.

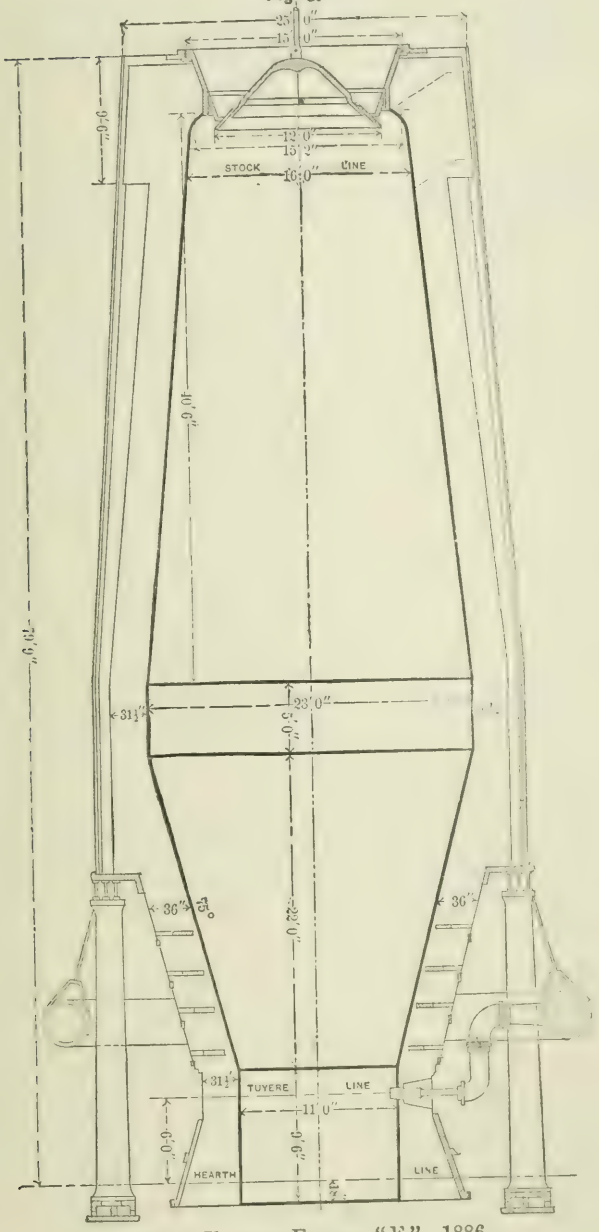
The furnace was started in October, 1886. The record for the next three months is as follows:

TABLE III.—*Showing the Working of Furnace F, Fig.* 8.

Months.	Tons.	Pounds of Coke per Ton of Iron.
November, 1886.....	6,735	2,128
December, "	7,494	2,105
January, 1887.	8,398	1,935

From January to May inclusive, the average monthly output was 8150 tons, on a coke-consumption of 1980 pounds. The volume of air blown was 27,000 cubic feet per minute, which was heated to an average temperature of 1200 degrees. The pressure at the tuyeres varied from 9 to 10 pounds, and was generally the latter figure when the outputs were largest. In June, the furnace was banked for eight days on account of scarcity of coke, as previously mentioned. As this furnace had been doing such excellent work, the others were banked first, in order to give this one, if possible, a continuous blast; but the supply of coke failed, and although the stop was comparatively a short one, yet, as a result of it, the coke-consumption increased to 2300 pounds. The monthly output from this time, up to January, 1888, was 7400 tons, when a second stoppage on account of labor troubles occurred. The length of this stop was 67 days; and, although the product was exceptionally large, even to the end of the blast—reaching in some months to over 8000 tons—the consumption of coke increased to an average of 2450 pounds.

Fig. 8.



Edgar Thomson Furnace "F." 1886.

It is greatly to be regretted that our work does not show an uninterrupted blast; since our records show, in nearly every case, a

decrease in output and an increase in fuel-consumption from 150 to 200 pounds per ton of iron, as a result of each "banking-up."

As a portion of the upper lining gave way in August, 1889, the furnace was blown out. The last full month (July) in blast showed an output of 6491 tons. The furnace was in blast—exclusive of the two stoppages I have mentioned—2 years, 7 months and 10 days, and made in that time 224,795 tons of iron, on an average coke consumption of 2317 pounds. The output for the first 12 full months was 88,940 tons on 2150 pounds of coke. The efficiency of the cooling-plates on the bosh-walls was very marked in this case. The exterior brick-work was in as good condition as at the beginning of the blast, neither a crack, nor a brick forced out of position, showing anywhere. After the stack was cleaned out a careful inspection was made. The bosh-walls were found to be in good condition. A measurement at the bosh-line showed that the furnace had widened out 18 inches, but with such uniformity that the widest variation shown by the gauge did not exceed 2 inches. From the bosh-line to the top of furnace the wear was very much greater. The furnace was relined without delay, and after a two weeks' drying out was put in blast on September 25, 1889, just seven weeks from the time of blowing out. The construction is the same in every particular, except that the diameter of the bosh is reduced to 22 feet, and the stock-line to 15 feet, 6 inches. The lining runs straight from bosh to stock-line. This change reduced the cubic capacity to 18,200 feet. The same number and size of tuyeres are used. The volume of air blown is 25,000 cubic feet per minute, a reduction of 2000 cubic feet from that used in previous blast. Beginning with October, the record of monthly outputs and coke consumption to the present time is as follows:

TABLE IV.—*Showing the Working of Furnace F, Fig.* 8.

Months.	Tons.	Pounds of Coke per Ton of Iron.
October, 1889	6,521	2,450
November, "	9,097	1,897
December, "	10,603	1,756
January, 1890	10,536	1,737
February, "	8,954	1,859
March, "	9,941	1,845
April, "	10,075	1,847
May, "	10,035	1,884

The best output for any one week is 2462 tons. The tempera-

ture of blast averages 1100° and the pressure 9½ pounds. The temperature of the escaping gases is 340°. An average composition is:

CO, 27.5 per cent.	$\frac{CO_2}{CO} = 0.42.$
CO_2, 11.7 "	

Counting the time the furnace was running in the first blast, and up to the end of May, 1890, in the second blast, including also the time spent in relining, the period covered is 3 years and 5 months; and in that time this furnace has made an output of 301,205 tons, a record which is unparalleled. In the month of May, this, together with a companion-furnace of precisely the same dimensions, made an output of 20,192 tons, on an average coke-consumption of 1882 pounds per ton of iron. The ores used are from the Lake Superior region, and yield through the furnace 62 per cent. of iron. The proportion of limestone carried is 28 per cent. of the ore-burden, and about 1200 pounds of cinder are made per ton of iron. The average analysis of the cinder is as follows:

	Per cent.
Silica,	33.00
Alumina,	13.00

The stone used contained but a very small quantity of magnesia. The silicon in the iron averages 1.60 per cent., the iron being run into ladles and sent to the converting-mill direct. The suitableness of the metal for this "direct process," as we term it, will be appreciated from the fact that under the rigid system of foundry-grading —by the fracture—the average percentage of grades Nos. 1 and 2 from all of our furnaces for the first five months of this year was 95 per cent., and the analyses of 100 ladles, filled consecutively from the furnaces, show average sulphur-contents of 0.023 per cent.

I shall not undertake to say what these furnaces will accomplish on an uninterrupted blast; but I believe the day is not far distant when we shall be able to show a record of 300,000 tons from a furnace in three years, and on a single lining.

In the period covered by the last decade there are three steps in the development of American blast-furnace practice that might be mentioned—first, in 1880, the introduction of rapid driving, with its large outputs and high fuel-consumption; second, in 1885, the production of an equally large amount of iron with a low fuel-consumption, by slow driving; and third, in 1890, the production of nearly double that quantity of iron, on a low fuel-consumption, through rapid driving.

Table V., on page 913, gives a summary of the preceding data.

TABLE V.—*Abstract of Results.*

Designation of Furnace.	Year in which Furnace commenced the Blast.	Cubic Capacity.	Volume of Air per Minute.	Total Output from Blast.	In First Twelve Full Months.			
					Output.	Average Daily Output.	Average Coke Consumption.	Capacity for one Ton of Iron per day.
		Cubic Feet.	Cubic Feet.	Tons.	Tons.	Tons.	Pounds.	Cubic Feet.
Isabella,.....	1876	15,000	.....	117,575	28,000	76	3,000	197
Lucy No. 2, Fig. 1..	1878	15,400	16,000	92,128	33,552	91	2,850	169
Edgar Thomson, A, " 2..	1880	6,396	15,000	.....	.....	*71	2,400	90
" " B, " 3..	1880	17,868	30,000	112,060	48,179	132	2,859	135
" " D, " 4..	1882	21,478	27,000	90,317	65,947	180	2,570	119
" " C, " 5..	1885	16,680	†31,000	118,000	64,998	178	2,677	90
" " D, " 6..	1885	18,950	22,000	150,374	74,475	204	2,250	92
" " C, " 7..	1887	17,230	24,000	203,030	‡72,554	198	2,230	87
" " F, " 8..	1886	19,800	27,000	224,795	88,940	244	2,150	81
" " F, " 8..	1889	18,200	25,000	.....	⁂113,000	310	1,920	59

* Estimated.

† After running nine months the volume of air was reduced to 28,000 cubic feet.

‡ The second twelve full months, by reason of a continuous blast, show an output of 83,219 tons on 2396 pounds of coke.

⁂ Estimated from record made to date.

NOTE :—On the completion of the twelve months in blast, the record for Furnace F, Fig. 8, blast of 1889, shows an output of 113,526 tons, and an average coke-consumption of 1892 pounds.

Discussion.

Sir Lowthian Bell, Middlesbrough, England: If the creation of the Iron and Steel Institute had been followed by no more important results than the opportunities it has afforded those occupied in a common pursuit, of comparing the results of their individual experience, it has done good work. It has taught us, among other things, that any knowledge we may impart to each other must not be regarded as a mere gift, but rather as an exchange, the value of which cannot be estimated by the sum of the two, but by entirely new factors, requiring the united experience and knowledge possessed by the giver and the receiver.

Speaking from my own experience, this has been amply illustrated in the frequent and, to myself, most beneficial relations which I have been fortunate enough to establish with a large number of the leading manufacturers in the United States of America, as well as with numerous other friends in various countries of Europe. Lastly, I am indebted to my connection with this Institute for the opportunity of again visiting the United States, and seeing once more those gentlemen from whom I have received so much attention, and towards whom I feel myself drawn by ties of the warmest friendship.

It is unnecessary to assure the author of the paper before us how grateful the members of this body must feel for the thoroughness of the work he has performed. Iron manufacturers for some time have been accustomed to follow, as closely as the nature of the operation permits, the action of the blast-furnace, and to study the laws which regulate that action. The results, however, which have been communicated to us this morning may well suggest the idea whether in our previous estimates we have not overlooked some of the conditions involved in smelting the ores of iron, for we have not only to consider the low consumption of fuel mentioned by Mr. Gayley, but the hitherto unparalleled large production of iron from one furnace. Thinking that it might interest others as well as myself, I have examined the figures given us by the author, and compared them with those of furnaces working under very different conditions, and I hope to be able to show you that it is not necessary to look for any new law to account for what, at first sight, seems little short of incredible.

The problem before us is to account, if we can, for a furnace producing about five times as much iron as we have hitherto made in Cleveland from our poorer ore and about twice as much as has been

obtained from our English hematite-furnaces of like dimensions. And this has been accomplished in the particular case before us with an expenditure of 16.80 cwts. of coke per ton of iron of a high average quality.

The consumption of coke per ton of iron depends on its quality, or on the more or less perfect manner of its combustion, or on the amount of work it has to perform.

Now, so far as concerns Connellsville coke, which was used in the case before us, nothing can be attributed to superiority of quality; for instead of containing 5 to 7½ per cent. of ash, like the coke of Durham, it has rarely less than 10 per cent., and sometimes more.

The completeness of the combustion must be judged by the amount of carbonic acid in the furnace-gases. When we compare the heat obtained from each unit of coke in a Cleveland furnace with that obtained in the Pittsburgh furnace (estimated as nearly as I have been able from Mr. Gayley's data), we find a somewhat higher duty for the former—partly because, in the gases of the Pittsburgh furnace, the proportion of carbon as carbonic acid to that as carbonic oxide is slightly below what is often found in the gases of a Cleveland furnace; partly because the temperature of the blast of the Pittsburgh furnace is not quite so high; and partly because of the higher ash in the American than in the English coke. The effect of these conditions is evidenced in the following figures:

Heat developed by one unit of coke:

	Cleveland. Calories.*	Pittsburgh. Calories.*
Carbon burnt to carbonic oxide, ...	2032	1898
Part of above burnt to carbonic acid, . . .	1672	1626
Heat in blast, ...	727	617
Totals, . . .	4431	4141

Showing a difference of 7½ per cent. in favor of the Cleveland furnace. It may be remarked that the relation between carbon as carbonic acid and carbon as carbonic oxide in the Cleveland furnace is as 1 to 2.11 against 1 to 2.35 in the Pittsburgh furnace. On the other hand, the actual quantity of carbon as carbonic acid is 5.97 cwts. in the former against 4.80 cwts. in the latter. As the generation of carbonic acid is almost entirely due to the oxidation of carbonic oxide

* Centigrade units: one calorie equals 3.968 British thermal units.

by the act of reduction of the ore, the difference in quantity of carbonic acid in the Pittsburgh furnace-gases must result from its partial return to the condition of carbonic oxide, involving thereby a loss of about 5600 calories per ton of iron. I believe that 5.97 cwts. of carbon as carbonic acid, per ton of iron, is about the full quantity capable of being produced by the reduction of the ore.

The furnace with which this Pittsburgh furnace has been compared performed its duty under an entirely different set of conditions, as the following two columns of figures show:

Working-Conditions.	Cleveland.	Pittsburgh.
Cubic contents of furnace,	25,500	18,200
Temperature of blast,	704° C.	593° C.
Coke consumed per ton of iron,	19.99 cwts.	16.80 cwt.
Limestone " "	11.00 "	9.03 "
Ore " "	48.00 "	32.26 "
Weight of blast per ton of iron,	87.15 "	71.20 "
Weight of gases, "	119.47 "	100.10 "
Temperature of gases, "	250° C.	171° C.
Weekly product of iron per 1000 cubic feet of furnace-capacity,	21.57 t'ns	128.00 t'ns*
Slag produced per ton of iron,	28.00 cwt.	10.71 cwt.
Calories, " "	88,577	69,569

To appreciate the full significance of these figures, I have prepared another table in which the difference in the various factors are met by corresponding variations in the appropriation of heat, the equivalents in coke being also given:

Heat-Requirements.	Cleveland. Calories.	Equivalent Coke, Lbs.	Pittsburgh. Calories.	Equivalent Coke, Lbs.
Reduction of peroxide of iron,	33,108	=7.47	33,108	=8.00
Reduction of metalloids in pig iron,	4,174	=0·94	2,631	=0 64
Dissociation of carbonic oxide,	1,440	=0.33	1,460	=0.35
Fusion of pig-iron,	6,600	=1.49	6,600	=1.59
Evaporation of water in coke,	275	=0.06	83	=0.02
Decomposition of water in blast,	2,380	=0.54	2,333	=0.56
Expulsion of carbonic acid from limestone,	4,070	=0 92	3,094	=0.75
Reduction of this carbonic acid to CO,	4,224	=0.95	3,488	=1 42
Fusion of slag,	15,400	=3.48	5,890	=0.84
Carried off in gases,	7,166	=1.60	5,013	=1.21
Lost by radiation, convection, tuyere-water, etc.,	9,740	=2.21	5,869	=1.42
Totals,	88,577	=19.99	69,569	=16.80

* Average of 3½ years, 2333 tons per week.

A very important loss of heat may be involved in that carried away in the escaping gases; and the lessening of this source of loss bears directly on the large make of the furnaces referred to by Mr. Gayley, and accounts for a large portion of the saving effected by the hot blast. Thus, when furnaces 48 feet in height were blown with cold air, nearly 25 per cent. of the effective heat of the coke—say 9 cwts. per ton of pig—was carried off in this way; but by heating the blast to 485° C. one-half of this heat was saved, and by doubling the capacity of the furnace the loss of heat carried away in the gases was further reduced to less than 2 cwt. of coke. I have heard the large make and fuel-economy of these Pittsburgh furnaces attributed to the facility with which the Lake Superior ore is reduced. In my opinion, this is doubtful. In my experiments I have made many comparative trials of the readiness with which different ores part with their oxygen. Of all, I believe the ore of our Cleveland hills retains its oxygen with the greatest tenacity; but nevertheless the reduction, so far as it is effected in the upper zone, is accomplished during the descent of this ore through the first 10 feet of an 80-foot furnace. I have purposely qualified my remark as to the amount of reduction effected in these upper 10 feet, because I think that about 25 per cent. of the oxygen is retained until the region of the tuyeres is reached.

One cause of the high temperature at which the furnace-gases escape is the undue rapidity with which they rise through the columns of descending solids. From this it might be inferred that the rapid driving of the Pittsburgh furnaces would aggravate the loss from this cause, whereas the contrary is the fact, seeing that the amount of gas in a given time is three times as great in them as in those of Middlesbrough; and if we take account of the difference in the cubic capacity, about four times as much gas is passing hrough a given space in the American as in the English furnace. To account for this apparent anomaly is the only remaining question. To do this, we may neglect the chemical action going on in the interior of the furnace, and consider only the cooling of the gases as effected by their meeting during their ascent the descending column of cooler solids.

Let us suppose a tall vessel filled with small balls, say of calcined Cleveland ore. A current of hot air is passed through them, and in time the air will flow out of the top at about the temperature at which it enters below, the slight difference being due to loss through the sides of the vessel. We now commence to withdraw the balls

from the bottom, say at the rate of twenty per minute, adding the same number at the top. The escaping current becomes slightly cooler, which means that a loss of heat at the top is still going on, but the balls withdrawn are still at a high temperature. Next, we increase the rate of withdrawal to 200 per minute, by which the gases fall, let us say, to atmospheric temperature; but we now find that the 200 balls withdrawn are no longer heated to the desired temperature, which shows that we have exceeded the heating capacity of the current of hot air, and we are obliged to retard the rate of withdrawal until the desired elevation is obtained. It follows from this experiment, that there is a given rate somewhat between 20 and 200 balls per minute which gives the best result.

In an old blast-furnace of 6000 cubic feet capacity, smelting Cleveland ore, 36.7 tons of iron were made weekly per 1000 cubic feet of capacity; the temperature of the escaping gas was 451° C., the blast being 485° C. The equivalent loss of coke in the escaping gases was 5 cwt. After increasing the furnace-capacity to 15,500 cubic feet, the loss from this cause was reduced to 2½ cwts.; but instead of obtaining 36.7 tons of iron per week per 1000 feet of capacity, we got only about 22 tons; the heat-intercepting power of our furnace-charges being capable of reducing the temperature of the escaping gases to 310° C.

It is quite reasonable, however, to suppose that materials might be introduced at the top of the furnace which would be capable of intercepting heat more effectively than Cleveland ironstone. I believe that your compact Lake Superior ore, containing one-half more iron than ours, possesses this property, and in consequence your Pittsburgh furnace-gases escape at a temperature of 171°, which I estimate represents a loss of only 0.93 cwt. of coke per ton of pig.

If the rate of driving in the Pittsburgh furnace represents the position of equilibrium which I have sought to describe, you will lose heat and waste coke by driving either more slowly or more quickly than you are now doing.

Edgar S. Cook, Pottstown, Pa.: In view of the vast advances in the production of pig-iron within the last ten years (not all due, by the way, to improved construction and equipment), he would be a bold prophet, indeed, who should put himself on record by prophesying that the end had been reached. Though progress in the future may not be made with such giant strides as in the past, there is ample scope for earnest endeavor in bringing the average

work of very many furnaces up to the high standard of the Edgar Thomson furnaces, as made public by Mr. Gayley.

At the present time, when the author has so ably treated of the large and splendidly equipped furnaces which use coke-fuel and burdens of the rich Lake ores, the advances of smaller furnaces using anthracite-fuel may not be without interest. In 1872, the State of Pennsylvania produced 968,453 short tons of anthracite pig-iron and only 388,011 short tons of coke-iron, out of a total product for the United States of 2,854,558 tons. The Pennsylvania furnaces in 1880 made 1,237,930 tons of anthracite and 801,817 tons coke-iron, out of a total product of 4,295,414 tons; while in 1889 the furnaces situated in the anthracite-district of Pennsylvania are credited with a yield of 1,793,639 tons, and the coke-furnaces 2,371,652 tons, out of a total of 8,516,079 short tons for the whole country. Previous to 1882, little or no coke was used in admixture with anthracite. Since 1883, the use of a portion of coke has become almost universal in the anthracite-districts. Many furnaces, though in close proximity to the anthracite-fields, use coke-fuel exclusively, so that of the 1,793,639 tons credited to the anthracite districts for 1889, only 344,358 tons were made from anthracite exclusively. Thus it would appear that the anthracite-iron industry is a decaying one, soon to disappear from the State, unless the coal-railroads adopt a more liberal policy for the future than has characterized them in the past. Coke-fuel costs more per ton in the districts referred to than coal. The preference shown to the former as a blast-furnace fuel is due to the greater ease with which the furnace is controlled and the less risk of costly scaffolds and chills. No one, not practically acquainted with the subject, is likely to appreciate the difference in the actual management of furnaces using anthracite coal and coke respectively. The difficulties attending the former greatly exceed the latter. Mr. Gayley, no doubt, can speak feelingly on this subject; and he may be willing to attribute part of his deserved success in coke-practice to his severe training in the anthracite part of the business. At the same time, if the anthracite-furnace is successfully and economically handled, the profits are greater or the losses smaller than if the more expensive coke is used.

The development in furnace-practice east of the Allegheny mountains, within the last ten years, will compare favorably with the advancement made in the West, notwithstanding that the yearly product of any single furnace sinks into insignificance when compared with the vast outputs obtained at Pittsburgh and Chicago. As the

Edgar Thomson furnaces fairly represent the progress in coke-practice, so the Warwick furnace, at Pottstown, Pa., may be taken to illustrate the improvement in anthracite-practice, although it is by no means alone in the work. I assumed the management of this furnace in December, 1877. It had been built and first blown in during 1875, and run with indifferent success, averaging about twenty tons of iron per day. The furnace was originally of 16 feet bosh, and 55 feet high. The bosh diameter was afterwards reduced six inches, and the height increased six inches, the capacity being about 5500 cubic feet. Previous to 1878, furnaces of this size were making from 150 to 250 tons per week, the latter figure being considered rather exceptional. From $1\frac{3}{4}$ to 2 long tons of coal per ton of iron was about the average fuel consumption. The Warwick furnace in 1878 produced 17,148 tons, of 2268 pounds,* an average of nearly 330 tons per week, using 2912 pounds of coal per ton of iron. The ore-mixture yielded 45.4 per cent. iron, and the furnace was blown with 8300 cubic feet of air per minute, heated in iron-pipe stoves to 855 degrees Fahr. For 1879 the product was 19,387 tons, or about 373 tons per week, with 2912 pounds of coal per ton of iron. The ores yielded 44.7 per cent. iron, and the volume of blast was 8300 cubic feet of air per minute, heated to 850 degrees Fahr. The campaign terminated in January, 1880, the furnace having run 107 weeks, and made 37,300 tons of pig. In starting, we blew in on an old in-wall. This gave way directly above the boshes, and gas penetrated to the red brick shell, necessitating blowing-out.

At the time, this work was quite a revelation, so far as the use of anthracite-fuel was concerned, and was the subject of wide remark, occurring, as it did, about when the Lucy furnace, at Pittsburgh, was showing the possibilities of coke and creating a stir in the iron circles, not only of that vicinity, but of the country at large as well. Comments were numerous, and some, of course, unfavorable. It was asserted that no furnace brick-work could stand such over-driving, although the difficulty had always been with anthracite-furnaces *to drive* them, and that the manager, with youthful ambition, was striving to make a record for himself at the expense of his company. As our president and directors were more than satisfied with the financial result, under the most unfavorable trade-conditions hitherto experienced, these criticisms did not create much discourage-

* An allowance of 28 pounds is made for "sandage" on each long ton of iron. All shipments, as well as all products of pig-iron, are therefore given in tons of 2268 pounds.

ment. To illustrate the other side, I cannot do better than quote from a letter of Mr. P. L. Weimer, a gentleman widely and favorably known in blast-furnace and engineering circles, published in the *Bulletin* of the American Iron and Steel Association. In part, it is as follows: "That the record above given is exceptionally good none will deny, and that it demonstrates the value of anthracite as a furnace-fuel is equally obvious, but there are several features especially worthy of consideration." Referring to the year 1879, he continues, "the output, which averaged 37,283 tons per week, is greater than the average of coke-furnaces of equal size using the rich ores from Lake Superior, and will compare in proportion to size with any coke-furnace. The ores averaged 44.7 per cent., while the lake ores employed by coke-furnaces average 60 per cent. of iron."

"Second. The fuel-consumption with ores yielding less than 45 per cent. is noteworthy; 2912 pounds per ton of iron."

"Third. The small amount of air per pound of fuel; 73.63 cubic feet"

"Fourth. The moderate heat of blast employed; 850 degrees."

"Fifth. The apparently excessive amount of stoppages."

(In explanation, I would say that these stoppages were largely due to cleaning boilers, no extra set being built at that time. We stopped two weeks each year for boiler-cleaning.)

"Sixth. The regularity of the furnace-operation, neither the maximum nor minimum weekly output varying from the average more than 12 per cent. Only 6.5 per cent. of the iron made was other than Number 2 and 3 mill, and but 1 per cent. was mottled and white."

"Seventh. The variety of ores employed, necessitating corresponding changes in flux."

It will be well to bear in mind that at this period chemistry was but sparingly employed. Only a few furnaces, Warwick among the number, then made use of the help afforded by analysis. The furnace was relined, no particular changes being made in the construction, and blown in during the latter part of March, 1880. We started off well enough, making about 400 tons per week, but the blast was of short duration. During this year the adverse critics of the improved practice apparently had the best of the argument. The manager was placed in an embarrassing position. Though firm in the belief that all would come out right in the end yet, as "success is the best evidence of success," he could not then offer the convincing proof.

In 1875 the furnace had been built with bosh-walls 5 feet thick, strengthened by upright wrought-iron buck-staves. In the repairs of 1880 this construction was retained, as light bosh-walls were then unknown, or, if considered at all, were regarded as a hazardous experiment. With the increased volume of air blown and lower fuel-consumption—I never run by pressure of blast, but always by volume—the brick-work melted rapidly, forming huge pockets or cavities above the tuyeres. These filled with finely divided ore and coal-dirt, and this mass, slipping at once into the crucible, chilled the furnace. Had we been as well informed then as in later years about the use of the oil blow-pipe, and of a device, employed only at our furnace and termed a tuyere slagging-valve, the mortification of a chilled furnace could have been avoided, although it is not probable that the furnace could have been profitably run. The only way I can account for the absence of the same trouble in the previous blast is that for the first month of that blast the furnace had been excessively gray, making vast quantities of kish. The bosh-walls had become heavily coated with a graphite covering, fortunately, evenly distributed. This had protected and preserved the brick-work and kept the boshes in good shape. When the furnace was blown out the existence of this coating had been plainly evident. It had proved a blessing in disguise, when I recall the anxiety and worry caused by the overheated condition of the furnace.

Not to prolong unduly this ancient history, recounting the trials of 1880, I will simply say that the bosh was finally filled with vertical coils of 1-inch pipe, encircling the whole circumference and extending about 10 feet above the tuyeres. These coils were supplied with water, which was kept in constant circulation. The pipes were built in the 5-foot wall. The exterior being in good condition, was not disturbed. The pipes were placed against the melted brick face and from 12 inches to 18 inches of new brick protected them from the contents of the furnace. This was the first instance, in our vicinity at least, of a water-cooled bosh, and many were the curious comments upon our placing the furnace in a spring of water. In December, 1880, the furnace was blown in and continued in blast 245 weeks, making 101,335 tons, or 413.6 tons per week. This was an unusual total for an anthracite-campaign. It may or may not have been reached by larger furnaces: I cannot say. Many difficulties were encountered, but patience and close application overcame them. For a few months, in 1881, we used one-eighth coke and seven-eighths anthracite; but as this small percentage of coke

only added to the cost, without yielding any corresponding benefit, it was discontinued. We ran on anthracite exclusively until January, 1882, when we began using one-fourth coke and three-fourths coal, which proportion we have continued until the present time. For the entire blast of 245 weeks, the ore-mixtures averaged 50 per cent. iron; fuel per ton of iron, 2912 pounds; air per minute, 8750 cubic feet, heated to 865° Fahr. The best average for three months shows 600 tons of iron per week with a fuel-consumption of 2212 pounds per ton of iron. The furnace was repaired in the fall of 1885. New hot-blast stoves were built, four sections of the Durham pattern with an aggregate heating-surface of 6240 square feet replacing two small stoves of only 3600 square feet of aggregate heating surface. The thick bosh-walls were dispensed with, and 22 inches of brick strengthened by a one-half inch wrought-iron jacket were adopted instead. The bosh-coils or water-cooled pipes were retained, being placed between the jacket and the brick-work. The capacity of the furnace remained almost 5500 cubic feet. We blew in during December, 1886, and blew out on May 28, 1889. During the campaign of 180 weeks we made a total of 107,595 tons of iron, an average of nearly 600 tons per week. The ores averaged 53 per cent. iron; the fuel 2665 pounds per ton of iron, and the air 11,050 cubic feet per minute, heated to 884° Fahr. The largest yearly product was 32,738 tons; the smallest, 30,045 tons. The best month's record showed 679 tons of iron for each week, made with 2464 pounds of fuel per ton, or 1 ton of iron per day for 55.7 cubic feet of furnace capacity, thus slightly exceeding the driving of the Edgar Thomson furnaces. For the largest year's product of the campaign, 61.2 cubic feet of furnace capacity made 1 ton of pig-iron per day, and for the entire campaign, including stoppages, etc., the average is 64.4 cubic feet of capacity per 1 ton of iron per day, which will compare favorably with the best coke-practice, considering the difference in the ores and fuel used. As a suggestion merely of the capabilities of anthracite, I would add that for a single week of this blast 702 tons iron were made, so that 55.0 cubic feet of furnace capacity produced 1 ton of iron per day. The Warwick furnace, however, makes a specialty of mill-iron, low in silicon and phosphorus. The silicon runs from 0.4 to 0.6 per cent. in gray forge grades, and phosphorus 0.30 to 0.35 per cent. The mottled and white irons made will not exceed 3 per cent. of the total product of any blast. No. 1 foundry will amount to 7 per cent., and the remaining 90 per cent. is gray forge of open and close grain.

Up to June, 1889, no change had been made in equipment or size of furnace since 1875, except the enlarged hot-blast stoves and an extra set of boilers to avoid stoppages for cleaning, one set being idle all the time. During the summer of 1889 the old furnace-stack, brick banded with iron, was torn down and a new steel shell erected. The enlarged furnace is 70 feet high and has a 16-foot bosh and a crucible 8 feet 9 inches in diameter. Its cubic capacity is about 8400 feet. A large Weimer engine and extra boilers were added to the equipment. On October 6, 1889, we blew in and have been in blast for nearly one year. The product will approximate 40,000 tons, the average fuel-consumption to date being 2464 pounds per ton of iron, with an ore-mixture of 55 per cent. iron, and 14,000 cubic feet of air per minute, heated to 900° Fahr. The largest record for any one month thus far averages 823 tons per week, with 873 tons as the best week. The lowest fuel-consumption for any five consecutive weeks is 2268 pounds per ton of iron. For the first year's work of the enlarged furnace, 76.4 cubic feet of furnace-capacity has made 1 ton of iron per day, the largest single week showing 67.4 cubic feet of capacity per ton of iron per day, which is somewhat above the average of the smaller stock. Before the campaign is completed we hope to improve the record. Heretofore the second year of a blast has shown the best results. Whether the same will hold good with this blast remains to be seen.

A THIELEN, Ruhrort, Germany: In offering a few remarks on Mr. Gayley's valuable paper, I would point out the strongly contrasting conditions under which we have to work our blast-furnaces in Germany, more particularly in the northern district, along the lower Rhine and the Elbe. On the lower Rhine the ores at our disposal contain about 45 to 48 per cent. of iron. They consist principally of the Dutch bog-ores and various grades of fine burnt pyrites or "Blue Billy," and are largely used in the manufacture of basic pig. If we should attempt to work our blast-furnaces so as to attain a monthly production of 10,000 tons, like the Edgar Thomson furnace "F," we should have to put through them a very much larger quantity of material than suffices in Pittsburgh. If for the Edgar Thomson furnace we assume an output of 10,000 tons of pig-iron, derived from 63 per cent. ores, and allow for a limestone addition of 25 per cent., then the entire blast-furnace mixture will amount to 14,000 tons of ore and about 3500 tons of limestone, making in all 17,500 tons. In Germany, on the other hand, if we take the average of our ores at 47 or 48 per cent. of iron, the ore-

burden alone will be 22,000 tons, and to this we must add 33 per cent. of limestone, or 7100 tons; hence we should be obliged to put 29,100 tons of material through the furnace to get 10,000 tons of product. In view of the leanness of our ores, if we should limit ourselves to the same quantity of material as goes through the Edgar Thomson furnace, our output would not exceed 6200 tons.

But there is another essential difference of conditions which prevents our following the example of the American blast-furnace. The native ores which we have to depend on are, as a rule, very fine. More than 25 per cent. of the ore in our blast-furnaces is of a much finer size than peas, and sometimes as fine as dust. With this material we should be unable to work furnaces of such heights as are common at Pittsburgh. We are limited, as long as we do not get other ores, to our low furnaces. At no great distance from our lower Rhine district there is a great body of oölitic ore, which we hope to be able to use to our great advantage when our projected canal is built, or as soon as we can induce the government to lower its rates on rail-freight.

The quantity of coke we require in Germany is considerably less than that used at Pittsburgh. With only 48 per cent. of iron in our ores, as against 63 per cent. in the ores of the Edgar Thompson furnace, we do not use more than 17 cwts. of coke per ton of pig.

Another reason for our not being able to follow the American lead is that the managers of our Bessemer and basic converters demand at least 1.8 per cent. of silicon in the pig, whereas American steel-workers are satisfied with 0.6 or 0.7 per cent. In striving to emulate American example in this respect, we encountered insuperable difficulties. Without entering into details, I will only mention that our converters use direct-metal in contrast to the usual American plan of using cupola-metal, which has a higher initial heat. We have to compensate for the lower temperature by keeping the silicon up to 1.8 per cent. I might add another point: We would not be quite satisfied with the costly luxury of relining our furnaces every three years. Thus far we have managed to keep our furnace-linings in for thirteen to seventeen years; at the present time we have several furnaces which are working in their sixteenth or seventeenth year, and they still give satisfactory results.

E. Windsor Richards, Middlesbrough, England: This history of the development of blast-furnace practice in and about Pittsburgh is a most interesting one. Mr. Gayley's paper, and Mr. Cook's statement yesterday, show that there has been an abundance of ex-

perience in blowing-in and blowing-out blast-furnaces. What are the objects, and what are the inducements for this excessive driving? The author shows us that the cost of doing so is enormous. Labor is dearer in America than in England, so that as much work must be got out of it as possible. But I believe that the charger at a Cleveland (England) blast-furnace handles as many tons of material per day as flesh and blood can do, and keep it up all the year. If, therefore, the production of a blast-furnace is to be doubled, twice as many men must be employed, and so it seems difficult to understand how this excessive driving can reduce the cost of labor very much. It is probable that an explanation may be found in the satisfaction felt by the owner in a margin of profit which is so pleasant to see that he directs his blast-furnace manager to give him tons of pig—of course, at the lowest possible cost—but yet to give him tons of pig. Then, there is another reason for this fast driving, which to those engaged in it has a great fascination, and is a constant stimulus to exertion—I refer to the practice of trying to beat the record. Captain Jones was a man of great ability, industry and energy, and we who knew him well sorrow to-day that he has passed away from us. I had the pleasure some nine years ago of visiting with him the South Chicago works. At that time they had a new and magnificent plant of blast-furnaces. Mr. Potter had arranged a most powerful equipment of stoves, blast-engines and several batteries of boilers. Captain Jones sent home the word, "Those boys at Chicago intend to beat our record." And they did. Then the boys at the Edgar Thomson works made their powerful plant still more powerful, so as to beat the boys at Chicago. And so it went on, until, in 1885, Mr. Gayley tells us that they forced in the enormous volume of 31,000 cubic feet of air per minute at a blast-pressure of 9 pounds per square inch, and overdid it to such a degree that they burned 2677 pounds of coke per 2240 pounds of iron produced, and shortened the short and merry life of the blast-furnaces. The members of the Institute will be curious and interested to know who has the record now—whether Mr. Carnegie or Mr. Potter. If it is Mr. Carnegie, he will probably be content with 100,000 tons of pig-iron per furnace per annum. If not, then we shall hear of more water-blocks, more fire-brick stoves, more blast-engines, more batteries of boilers. But the furnaces themselves will have to be of a very greatly increased capacity to work economically, having boshes from at least 23 to 30 feet diameter, provided, always, that there is a sufficiently comfortable margin of profit in the transaction.

It is interesting to study the internal lines of the furnaces described in this paper, and to note where and how they differ from the English forms, so as to bring out clearly what has, in my judgment, enabled the American practice to produce what are really astonishing results. We are at once struck by the much larger diameter of the hearth of the American furnace, and it is primarily due to this enlargement of the hearth that the increased output is obtained. But this of itself would have been of no great advantage if the pressure and volume of the blast had not been at the same time very largely increased. Look at the form of the furnaces shown in Figs. 2, 3 and 4, and you will see that they have very steep and narrow boshes, which my experience has led me to regard as an extremely bad form—so bad, indeed, that a furnace with a steep and narrow bosh of 34 degrees, like No. 3, could not work well and make a regular quality of iron at a rate that we must now term, I suppose, slow driving. Indeed, such a form necessitates fast driving so as to keep the materials in motion and prevent scaffolding. Steep and narrow boshes favor the formation of an arched scaffold much more than a flatter bosh. The latter gives an arch of larger radius, much more difficult to form. This arched scaffold causes great trouble when the slip takes place. No doubt the English bell, 11 to 12 feet in diameter, is better and much simpler than the complication which is shown in Fig. 3. It is gratifying to see that Mr. Gayley has discarded this arrangement in the later forms. The adoption of the 11-foot bell gives a 2-foot space all around for the material to pass. A diameter of throat of 11 to 12 feet is a good proportion for a 22-foot bosh. I am not at all in favor of small boshes. The descent of the material in the cylinder is made difficult through close packing, so that the diameter from the throat downward to the top of the bosh should constantly increase, and so release the material from a too close contact, and permit the free passage of the gases. This form, at the same time, tends to increase the diameter of the bosh, giving greater capacity to the furnace, and favoring a greater production with less forcing. Mr. Gayley has evidently recognized this, for in speaking of furnace "F," Fig. 8, he says, "from the bosh-line to the top of the furnace the wear was very much greater," and a little further on, "the lining runs straight from bosh to stock-line." I advocate more the form of Fig. 8—making a 15-foot throat, and tapering down to a very large bosh, so that the materials unpack themselves, and the gases pass readily through. The angle of the bosh should not, in my opinion, exceed 75 degrees, and may, with advantage, be as low

as 70 degrees if the bosh is about 22 feet in diameter. Now let us examine the last furnace which the author has illustrated, Fig. 8. The lines show a return to the English form, with the exception that the tuyeres are placed at a higher level. I would suggest that if the two great works before named still purpose continuing to compete for the record, the one which adopts the greater cubic capacity will win.

Mr. E. C. Potter, of Chicago, has already contributed so much in bringing about the brilliant results now recorded, that this first day of October, 1890, will mark an epoch in the blast-furnace practice, and will in the future be pointed to and cited as the goal attained after years of thought and labor by many who have devoted time and vast sums of money to reach it. It would be of great service in this discussion if Mr. Potter would give us the results of his experience; for it must be evident that, being at such a distance from supplies of fuel, he would give much attention to economy in that direction. There seems at the present moment to be a lull in the race of the rivals. They appear to have obtained as much out of 20,000 cubic feet of blast-furnace as can be secured with economy; and we have before us the fact that a furnace using ores which carry 60 per cent. of iron, produces 2000 tons of pig-iron weekly, with the very low consumption of 1920 pounds of coke per 2240 pounds of pig made—a result which reflects the greatest credit on the owners and managers.

A very interesting feature arises out of this rapid driving. We know that a blast-furnace settles down in the course of a couple of months to a certain rate of driving, by which it yields a regular quality of iron. Take the case of a furnace making Cleveland iron, 60 per cent. of which is of No. 3 quality, with a yield of 20 cwts. of iron to 20 cwts. of coke; now, if that furnace be driven without increasing the coke, the quality of the iron will immediately fall off, and No. 4 forge-iron is likely to be the product, and with yet a little faster driving it falls to white iron. But the American practice of doubling the production with nothing exceptional in the temperature of the blast, makes a ton of Bessemer iron with considerably less than a ton of coke—a result, I believe, never before attained. We must presume that this economy of fuel was an accidental outcome of fast driving. At what point of production was this observed? I have constructed several blast-furnaces, some with hearths 10 feet in diameter, and some with 11 feet, and have made Bessemer pig-iron for steel purposes up to 1000 tons weekly, without remarking an

special economy—indeed, rather the contrary, although the temperature of the blast was maintained at about 1400 degrees Fahrenheit.

Is it more desirable or more economical, all things considered, to drive a blast-furnace at a speed of 2000 tons per month, than to obtain the production from two furnaces? Of course we must remember that only the shell of the blast-furnace is to be considered. The number of stoves and blast-engines and boilers will be the same in either case, so that it is simply a question of the outer shell of the furnace. And so, I repeat, is it desirable, all things considered, to drive the furnaces at such a rate—the author showing us that the cost of so doing is enormous?

When a furnace is producing 2000 tons per week, is the iron as uniform in quality as when the product is limited to 1000 tons? In my belief it is, because I have found when going at 600 to 1000 tons per week, that the quality was as good in making 1000 tons as in making 600. And so one might infer that the quality would be as good when making 2000 tons as when making 1000. But some may ask, if this be so, why has it become necessary to adopt the system of storing several casts of molten iron taken from two or more furnaces in a large vessel termed a mixer, before the iron is considered fit for the Bessemer converter?

E. C. Potter, Chicago, Ill.: I am confident that Mr. Windsor Richards has no intention of precipitating a fight between the Pittsburgh boys and the Chicago boys. I would assure him that the rivalry is a most generous one, and that each of us, I think, has profited by our mutual comparisons of work. We have a record of furnace-practice at Chicago which will compare favorably even with the magnificent work of the Edgar Thomson furnaces. The figures are not quite so large, yet, considering the size of the furnace, they may furnish food for reflection as to whether very large furnaces are the ones with which we may expect to obtain the highest possible economy. The record to which I refer was not accomplished by one of the furnaces at South Chicago, I regret to say, but by a furnace at the Union plant of the Illinois Steel Company. The furnace is comparatively small, containing only about 8800 cubic feet capacity. In a single week this year it has made 1414 tons of Bessemer iron, and in a month over 5800 tons; and this rate has been sustained for several months, showing that it is not merely a spurt. The fuel-consumption for the month per (long) ton of pig was 1680 pounds, or 57 pounds less than the lowest monthly average (for January, 1890) which Mr. Gayley has obtained.

The question that presents itself at once to my mind in considering the performance of this furnace, and not this one alone, but all the fifteen furnaces now owned by the Illinois Steel Company, is that in proportion to their cubic contents, the smaller furnaces have invariably shown much the better results, both for fuel-consumption and output. The performance of the furnace I have just cited shows a product of nearly $2\frac{1}{4}$ tons of iron per day for each 100 cubic feet of capacity. To equal this, Mr. Gayley's furnace should make over 12,000 tons per month, an output which even the most enthusiastic American can hardly consider quite possible in the light of our present knowledge. The lesson to be learned from these records (and they are borne out by all our practice in Chicago and the West) would seem to be that a furnace of moderate size, say from 10,000 to 12,000 cubic feet capacity, must be the one to which we should look for the best results, economically considered. Personally, I admit, I have not quite reached this conclusion yet. We have now under construction at South Chicago a plant of new furnaces of the largest size, the same size, practically, as those of Mr. Gayley. They are 85 feet high, 21 feet in bosh-diameter, and have a cubic capacity of about 18,000 feet. Until we have operated these furnaces and determined just what they can do in comparison with smaller ones, I would withhold my judgment. In considering size, locality must of course play a large part. In the case of the Edgar Thomson plant, located near the coke-region, where the fuel-saving is not so important an item as in the West, a very large furnace, with a very large output, which reduces the labor-cost per ton and the item of general expenses, is undoubtedly the best one for that locality; but for the West, and for Chicago particularly, fuel-saving is and must be the paramount feature of our furnace-practice.

I have followed with much interest Mr. Gayley's experiments with his furnace-lines, and also the remarks of Mr. Richards on the same; and my judgment is in entire harmony with Mr. Richards in his criticisms on the lines. We have done considerable experimenting upon furnace-lines at South Chicago, more particularly in the direction of lowering the bosh of the furnace. The furnaces described in my paper before the Iron and Steel Institute * four years ago, were of the high, steep-bosh kind. They gave good results in driving and fuel-economy, but as Mr. Richards had rightly foreseen, during the first 6 months of their life, or until they had cut their own lines, they occasioned us great trouble with sticking

* *Journal of the Iron and Steel Institute*, No. 1, 1887, p. 163.

and hanging and scaffolding. During the first 6 months of the second year we got our best work out of these furnaces. When relining became necessary, we lowered the bosh from 35 feet above the hearth-level to 26 feet, all other dimensions remaining the same. In this way the angle of the bosh was flattened from 81° to 72°. The relined furnaces behaved very well as regards scaffolding and hanging, never giving an hour's difficulty on that score, but they absolutely refused to be driven, and consequently we did not get as good fuel-results as we did from the rapid-driving furnaces. This, of course, was a fatal feature for Chicago practice. To overcome it we have this year lined a furnace upon a still further modification of the lines, by leaving the bosh-height the same, 26 feet, but widening the hearth from 11 to 12 feet, thus restoring our old bosh-angle of 81°. This furnace seems to have realized our expectations, and has been working very smoothly from the start, and driving with reasonable rapidity and getting very good fuel-consumption. It is now working side by side with one of the first-described furnaces, and is making from 100 to 150 tons per week more than the latter. I am inclined to believe, therefore, that we have very nearly solved the question of furnace-lines for the material with which we have to work.

G. J. Snelus, Workington, England: It has been of great advantage to us Englishmen to have had advance copies of Mr. Gayley's paper placed in our hands before we started on our ocean voyage. Many of us have thus enjoyed an opportunity of carefully considering the paper; speaking for myself, I was certainly delighted with it, nor can I express my thanks too warmly to the author and to the owners of the works for the magnificent way in which they have laid their results before us. The wonderful yield, fully three times what we are getting from furnaces of nearly or quite equal capacity in England, and the remarkable economy, a saving of twenty per cent. in fuel, are, I venture to say, points which we Englishmen cannot afford to overlook. I have endeavored to compare the several factors in the Pittsburgh furnace-practice with our own English practice, and I will now attempt to review them as rapidly as possible.

First, with regard to the shape of the furnace, I am pleased to see that the Edgar Thomson furnace, "F," Fig. 8, which has given these wonderful results, is a return to the lines of most of our good English furnaces. Mr. Richards has already called your attention to this, and I can only add that if you were to place our West

Cumberland lines upon Fig. 8, you would find a coincidence in all particulars except that our furnace is smaller. The angle of the bosh is identical—75 degrees. The position of the bosh is identical. The proportion of the hearth to the bosh and to the height of the furnace is about the same. The mere shape of the furnace does not, therefore, seem to account for the improvements which have been attained.

Next, as to the fuel, Sir Lowthian Bell has stated that our fuel is of better quality than that at the Edgar Thomson furnace. If I cannot say quite as much for our fuel on the west coast, yet it is not much worse; we have in West Cumberland a coke which contains about ten per cent. of ash and from three-quarters to one per cent. of sulphur. This is also about the average of Connellsville coke I believe, so that we may put aside fuel as not contributing in any great measure to the wonderful result.

Then, as for ores, we are told that they yield 62 per cent. of metallic iron, whereas our Cumberland ores yield on an average perhaps 56 per cent. Here is a point of advantage which we should not overlook, for not only does the ore contain more iron, but less earthy matter; and this, I take it, is a point of great importance in producing large quantities of material quickly, because we have so much less slag to get rid of and correspondingly less fuel to deal with. As far as the composition of the slag is given, I find very little difference between it and our Cumberland slag.

With regard to the composition of the gas, I find it practically the same as at most of our furnaces in England, so that no point of advantage is to be sought there.

The temperature of the blast, we are told, is 1100 degrees Fahrenheit, which is about the temperature that we regularly get at our own works; hence, no advantage exists there. But with regard to the pressure of the blast, the Edgar Thomson furnace enjoys a very important advantage. In West Cumberland, we have for many years been blowing with a blast-pressure of six pounds to the square inch, but in Pittsburgh they have from nine to ten pounds. This, in my opinion, is a very great advantage in getting large outputs. It is quite evident, also, that a much larger volume of blast is blown at these furnaces than we are accustomed to in England. It has been a common idea among our English iron-masters that by increasing the volume of the blast and driving faster we wasted fuel. I shall refer to that point later. Clearly, as regards large outputs, the

American furnace has a decided advantage in the higher pressure and greater volume of blast.

In respect to the radiation of heat from the furnace, there must obviously be an advantage in the American practice, and I cannot help thinking that it is one of the essential points which account for the gain and saving of fuel. A furnace which is producing as much iron with one casing as we are with four, is bound to radiate very much less heat than we do with four furnaces.

Looking to the product, we see that the iron is low in silicon—a little over 1¼ per cent. Here, again, I believe, there is a distinct advantage. Some years ago I had the opportunity of proving this, in working some of that wonderful material, the Campanil ore, which contained only about five per cent. of silica. It was very porous, allowing us to drive our furnaces faster and get less slag and produce iron with less than our usual amount of silica. For several years I drove our furnaces on a 1¼ per cent. silicon iron, which was taken direct to the Bessemer works. Unfortunately, the Campanil ore is becoming very scarce, and we in Cumberland can no longer afford to use it, but have had to fall back on local ores which are more siliceous. Our experience, however, was conclusive, that in making a non-siliceous iron we generally realized a saving in fuel. But the American saves coke not only by avoiding the reduction of much silicon, but by making an iron which also contains less carbon than our own. It is a point on which, unfortunately, the hands of the English metallurgist are tied; for our engineers declare that when they go about in our works they like to see No. 1 pig with plenty of graphite in it. If they come across a pig which even slightly approaches white, they immediately wonder whether we are not making very bad steel; and so, to keep up the reputation of our steel, we often run counter to our own ideas of what would be the best economy.

Finally, the characteristic rapid driving of the furnace is, I think, an important factor in its economy, and yet in England our experience with rapid driving has shown a greater consumption of fuel. I never could account for this anomaly. I have always been an advocate of rapid driving, but my difficulty has been that I could not drive with any real rapidity. The men would not be driven; the manager would not be driven, and the furnace would not be driven, because we did not have the plant and could not get it. But, given the mechanical means for driving the furnace rapidly, and, I

believe, you must necessarily drive it economically. I certainly am in favor of the practice which we have had put before us.

As for the durability of the furnace-lining, I believe in a short life and merry one. If we gained nothing but the saving of coke, amounting to four hundredweights to the ton, or 22,000 tons of coke per annum on this particular furnace (Fig. 8), surely, that alone would supply sufficient money to re-line the furnace two or three times over. But I think there is a saving of wages as well as of coke. An economy in labor and management must assuredly result from getting out of one furnace what is elsewhere obtained from four. If I had to build myself, I would certainly prefer to erect one furnace to do the work rather than two, even though two might yield the same output, and I would do it even if the two furnaces could be put up at the cost of one.

After having reviewed all these points, I yet fail to perceive sufficient cause in either plant, material, or mode of working to account for the final achievement. I cannot help thinking that this result has been mainly due to the wonderful spirit of emulation among the proprietors, the managers and the men; and I for one would only be too glad if we Englishmen and our managers and our men could catch the same spirit.

Julian Kennedy, Pittsburgh, Pa.: Mr. Gayley's paper is very suggestive in showing the great effect produced by very small, or apparently very small, changes in the lines of the blast-furnace. In the early days of furnace-driving, when the results were not as good as they are now, but when we tried just as hard to get them, our fast driving was not due altogether to the desire to drive fast. We often tried a little slower rate of driving, but then found that the furnace would not work, and so we were obliged to drive as we did. I ascribe a great deal of this to the difference in the lines of the furnace then from what they are now. These differences do not seem very great, but still they made a marked difference in the result. Mr. Gayley tells us that in his endeavors to reduce the blast in one of his furnaces, he found it impracticable to drop below 28,000 cubic feet per minute on account of the resulting irregularities in work, until, by making an apparently trifling change in the dimensions of the top of the furnace, he was able to effect a further reduction of 3000 cubic feet per minute. I have no doubt that if the tops of his furnaces were restored to the old dimensions of bells and stock-lines, he would again be obliged to blow harder. Thus all through the paper we see evidences of the great effect of very small changes in

the furnace-lines. When this modern system of fast-driving was begun, our theory and practice were to use a wide top, and to our own detriment we carried that practice to an extreme. The accepted theory at that time was that the proportion between the diameter of the bell and stock-line should be a geometrical one—that is, if a $10\frac{1}{2}$-foot stock-line was right for a 7-foot bell, we considered that an 18-foot stock-line would be suitable for a 12-foot bell, whereas to-day the stock-line would be made only $15\frac{1}{2}$ feet. The record throughout impresses me with the important bearing of the furnace-lines upon the work.

I am also inclined to think that Mr. Gayley is substantially aided in his splendid work by his fine equipment of blowing-engines. These powerful engines are regulated by automatic governors, which can be set at any desired number of revolutions, and which will maintain the engines at that speed whether the blast-pressure be six pounds or sixteen pounds, thus insuring uniformity in the volume of blast entering the furnace.

Mr. Gayley has succeeded in reducing his coke-consumption to a point that is undoubtedly very satisfactory, and it is questionable whether it will ever be brought much lower. His labor-item seems as low as it could be in smaller furnaces aggregating the same production, and compared with these the cost of management for some departments must be less. If Mr. Gayley succeeds in obtaining an average life of 300,000 tons to a lining, as I have no doubt he will, and a campaign which is not interrupted by labor-troubles or other extraneous causes, I do not see why his practice, economically considered, will not be as good in every respect as that which is possible with a greater number of small furnaces aggregating the same production.

W. Whitwell, Stockton-on-Tees, England: Little has been said during this discussion as to what is being done at English works without much pressure and with a furnace very similar in lines to the one that has been so highly approved by the several speakers (the Edgar Thomson furnace "F," Fig. 8). At Thorneby we are working a furnace 75 feet high and $18\frac{1}{2}$ to 20 feet in diameter, with a 10-foot hearth, a bell of 10 feet 6 inches, and six tuyeres. We do not use the large tuyeres, so successful in America, because up to the present time we have not found their use advantageous. Our tuyeres are about $4\frac{1}{2}$ and occasionally 5 inches, when required. Our pressure of blast does not, as a rule, exceed 5 pounds at the engine-house, or $4\frac{3}{4}$ pounds at the tuyeres. We commenced many years

ago with smaller furnaces, never more than 20 feet in diameter and 60 feet high. We were the first in the Cleveland district to use the rather larger cubic capacity of the furnace, and to some extent we were successful; but the additional height in the Cleveland district was the great element of success. It was arrived at, perhaps, more as an accident than otherwise; more with the desire to obtain additional cubic contents than to put in a new bottom; and in that respect Mr. Vaughn deserves the congratulations of our own district and perhaps of the world, because that was the first occasion when greatly cheapened production was obtained.

But to proceed with this hematite-furnace. Our make is approximately 700 to 800 tons per week. At the present time we are working in the fourth year of the furnace-life, at about 800 tons; and we are improving, and anticipate still further advances. Our ores are very different from the rich ore that we have heard described so vividly here; and I should like to learn from the author whether the general American practice coincides in any degree with what he has described, or whether it is only a particular furnace working this particularly excellent ore from Lake Superior which can show such remarkable results. Our ore does not average over 50 to 50¼ per cent. of iron. We generally use a mixture of seven or eight different kinds of ore—ores from Africa, Elba and Spain, from North and South, and in mixing we have in view the quality of pig that our customers require for the time being. We do not work so low in silicon as is the practice here, to such manifest advantage. We try to keep our sulphur down. Our phosphorus generally runs a little below 0.04 per cent. and we sometimes get it down to 0.0275. Our practice is not as regular as that of works which have their own converters or open-hearth furnaces; but we make what our customers require—iron for shipment, perhaps to the Continent, of one quality, and iron for local consumption of another, and change sometimes from one week to another and yet we manage to produce, as I said, something like 800 tons of pig a week.

Our average consumption of coke for the last two years has been about 19 cwts. per ton of iron running with this 50 per cent. ore, and with an addition of 6¼ to 6⅓ cwts. of limestone. This is much less limestone than is used in America even for the richer ores; and I am much inclined to think that by using more limestone we might work to better advantage, keeping our furnaces more open and getting rid of our sulphur more successfully than we are doing. The average silica in our ores is about 6¾ to 7 per cent.; if we work

with that regularly, we have very uniform results. The richest ores are the most economical ones to smelt.

While speaking of ores I might mention those from the north of the Gulf of Bothnia, which we have used to the extent of 10,000 tons a year. That ore is no longer in the market because the company has stopped shipments. The last few cargoes are arriving at the present time. If the company should continue, I anticipate success for it if they can confine their shipments to purely silicate ore, for there can be nothing better to work with—in fact, it is an ore that would approximate very closely the splendid Lake Superior ore. On the other hand, much apatite occurs in it here and there, but it is as veins of phosphate of lime which run through the ore-lumps and are readily distinguishable, so that the ore can be selected with ease, and the inferior grades set aside, and used to great advantage for ordinary purposes, as it is really an excellent ore.

In regard to the pressure of the blast, I am inclined to think that we must suit our own conditions in our driving, and adopt what is economical with us. If we are willing to have our furnaces run on shorter time; if we agree to lose somewhat in fuel-efficiency, we can drive harder; but if a furnace can be made to drive on a certain burden, I believe that it is a great point that we ought to look to.

We are apt sometimes to put on pressure and drive our furnaces down, which simply means lowering the quality of the product and rising in sulphur—two decidedly objectionable features.

It is well to remember that we change our heats when our furnaces get too hot rather than burn our iron. In America the heats, I believe, are never changed in the best practice.

Again, we work by gauge-pressure. Americans work regularly, I believe, by the revolutions of the blowing-engine, and they always keep up the speed and do not lessen the production of the furnace for the sake of preserving the quality as we do. In these two respects there is something to be learned. How do they manage? In the best practice they have an individual engine or set of engines to each furnace, and not three or four engines working into the same main, and so contributing the common pressure to the several furnaces. They do not check the pressure. If you could give us the same ore, with our beautiful Durham coke, which runs from 5 to at most 7 per cent. ash, I believe we could show you different results; but with the material that we have, under existing circumstances, I think that our growth will have to be gradual. I should like, as I said before, to know whether it is these particular Edgar

Thomson furnaces using three-quarters of a million of tons of the very best Lake Superior ore that are working the best, or whether others which import ores from Spain or Africa would get anything like an approximate result.

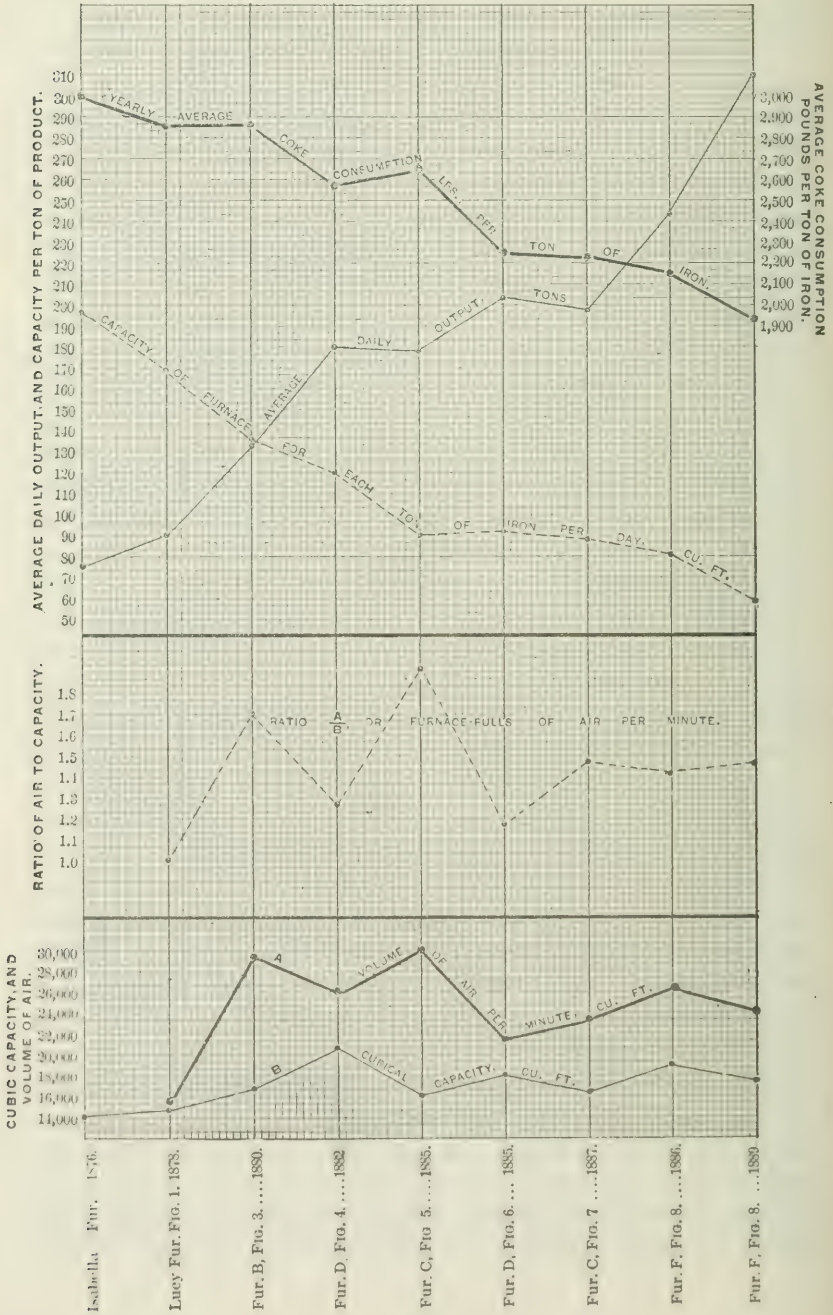
Now what is our English practice? It is to earn the most money, or in some circumstances to suffer the least loss; for it is not always profitable to make iron in England. When we keep a furnace running a long time without checking it for repairs we think it is of advantage. As we are working now, we expect to make pig-iron from one lining for twelve years. If we do that we are perfectly willing to reline. We cannot reline it as it is done in America. Everything in America in the shape of repairs to building seems to be much more rapidly done than we have any experience of. I don't understand it. I hear of relining a furnace (and drying it for fourteen days, too,) in seven weeks. If we reline in three months we consider that we are doing exceedingly well. How you do it, whether your labor is more under command, whether your labor is more energetic, or whether there is competition in your labor, we do not understand. We simply put our money into our furnaces, and perhaps we expect to build our 2 or 3 furnaces for the same amount that you spend on one.

You have great provision in heating-power, yet not so much as is given in our district. You have only three large stoves to one furnace with this large production, and therefore you do not work up to quite the same temperature. In looking back over many years it has been interesting to us to witness the enormous increase in production in America. We congratulate you heartily upon it. We are all striving for the same result, and I am inclined to think some of you are going much more rapidly than we are.

William Kent, New York city: I have attempted to study Mr. Gayley's paper by the method of a plotted diagram, Fig. 9. As I interpret his "abstract of results," there are three columns which represent results or effects—namely, "average daily output" of the furnace; "average coke-consumption" of the furnace; and "capacity in cubic feet of the furnace for one ton of iron production per day." On the diagram I have plotted these results in the three upper lines (omitting the small furnace, Fig. 2, as exceptional). Beginning with the Isabella furnace at 76 tons per day, in 1876, and rising to the Furnace F, Fig. 8, at 310 tons per day, in 1889, the increased product follows a fairly uniform and steady law of rise. The line representing the coke-consumption, begins at 3000

FIG. 9.

DIAGRAM, SHOWING THE CONDITIONS AND RESULTS OF PIG-IRON PRODUCTION AT THE ISABELLA, LUCY & EDGAR THOMSON FURNACES. PLOTTED IN PART FROM MR. JAMES GAYLEY'S "ABSTRACT OF RESULTS."


AVERAGE DAILY OUTPUT AND CAPACITY PER TON OF PRODUCT.
AVERAGE COKE CONSUMPTION POUNDS PER TON OF IRON.
YEARLY AVERAGE COKE CONSUMPTION LBS. PER TON OF IRON.
DAILY OUTPUT TONS
CAPACITY OF FURNACE AVERAGE FOR EACH TON OF IRON PER DAY. CU. FT.
RATIO OF AIR TO CAPACITY.
RATIO A/B OR FURNACE-FULLS OF AIR PER MINUTE.
CUBIC CAPACITY AND VOLUME OF AIR.
VOLUME OF AIR PER MINUTE. CU. FT.
CUBICAL CAPACITY. CU. FT.
Isabella Fur. 1876.
Lucy Fur. Fig. 1. 1878.
Fur. B, Fig. 3. ....1880.
Fur. D. Fig. 4. ....1882.
Fur. C, Fig 5. ....1885.
Fur. D. Fig. 6. ... 1885.
Fur. C. Fig. 7 ....1887.
Fur. F. Fig. 8. ....1888.
Fur. F, Fig. 8. ...1889.

pounds and declines somewhat irregularly to 1920 pounds. The cubic capacity of the furnace for one ton of iron per day, begins at 197 cubic feet and decreases steadily to 59 cubic feet, the present rated capacity.

These are the three results. If we study the paper and the table for the causes of the results, we shall find in the table two columns of figures which may possibly indicate such causes—first, the cubic capacity of the furnace, and second, the volume of air blown into the furnace per minute. The ratio of these two quantities, or the number of furnace-fulls of air blown in per minute may also be considered. This ratio is not given in Mr. Gayley's table, but I have plotted it in the diagram; the figures expressing the ratio, beginning with the Lucy furnace, Fig. 1, being 1.04, 1.68, 1.26, 1.86, 1.16, 1.40, 1.36, and 1.37. The lower line on the diagram represents the cubic capacity. It begins at 15,000 feet, rises to 17,000 feet, and then runs along very irregularly to the present capacity, 18,200 feet. The difference in capacity between the fourth furnace in the list, Fig. 3, and the tenth furnace, Fig. 8, is less than 400 cubic feet, showing that the cubic contents of the furnace is nearly a constant quantity.

The volume of air is very irregular. If the cubic capacity of the furnaces, and the volume of air pumped into them, or the ratios between these factors are supposed to be efficient causes, and the daily output and average coke-consumption are assumed as their effects, I for one fail in the examination of the diagram to discover the existence of any relation between them. If you contend that rapid driving increases production and improves economy, then by taking certain others of the figures I can show you also that the opposite conditions lead to the same results; that rapid driving conduces to economy, and that slow driving does likewise; that large capacity gives economy, and that small capacity gives still greater economy, and vice versa; so that neither the volume of air pumped into the furnace, nor the cubic capacity of the furnace, is the controlling cause for the low consumption of fuel and high production of pig-iron. I do not pretend to define the cause. On the contrary, I think it very obscure, and a subject for further study.

Mr. Kennedy has touched on one possible cause—the lines of the furnace. I think the physical and mechanical, as well as the chemical properties of the ore and the coke, are also very important conditions. Other causes, or influencing conditions, are probably the distribution of the materials in the furnace, the method of charging,

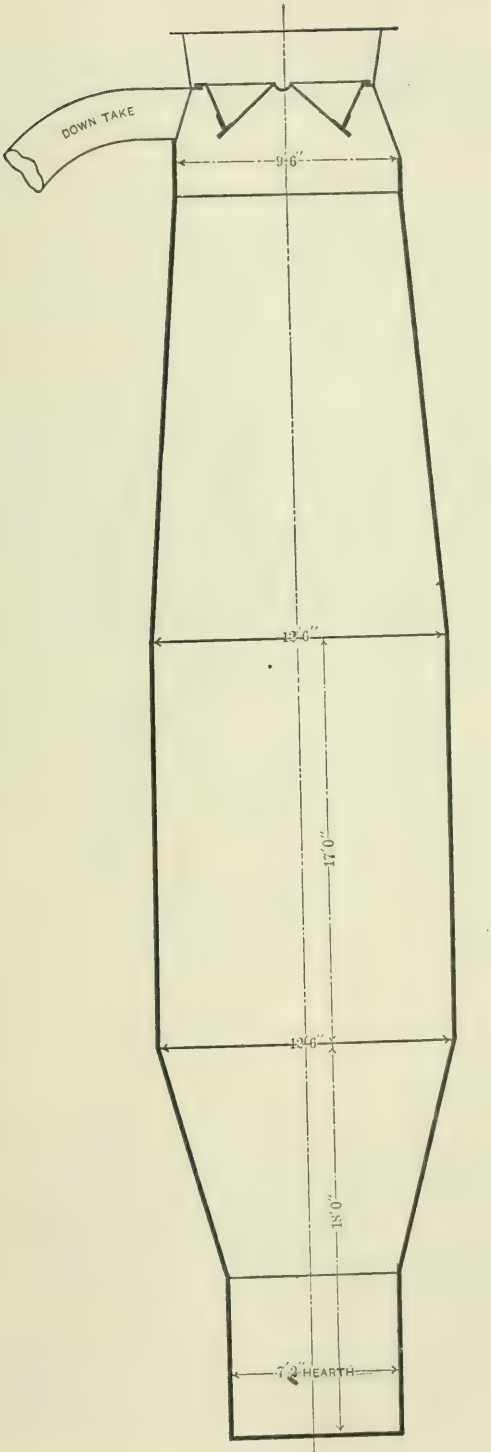
the condition of the hot materials inside of the furnace (about which we can tell nothing to a certainty, because we cannot get inside of the furnace while working), and especially in the zone of fusion; whether the materials at the proper heat for reduction are pulverulent or broken into small pieces by decrepitation, allowing the gases to come up freely through them, or whether they are agglutinated in a pasty mass, preventing the thorough distribution of the gases. All these, I think, exert potent influences on the economy of production. As Mr. Kennedy has said, there are suggestive indications in the author's paper that change of lines has an important bearing on economy. Mr. Gayley has mentioned, that while a furnace is in blast, under uniform conditions, a temporary stoppage or some accident may cause a sudden change in its rate of working and reduce its production; and that, too, with no change in the conditions of blowing. When the furnace is subsequently blown out, it is found that there has been a slip, or scaffold, and thus the cause of the change seems to have been some modification in the furnace-lines, or in the mechanical condition of the mass of material inside the furnace.

On the other hand, as Mr. Gayley also points out, the increase in thickness of the carbon coating inside of the furnace conduces to more regular and economical working; and he intimates that a thorough protection of the bosh-walls with cooling plates, by preserving the interior lines of the furnace, has an important and beneficial effect upon economy. It is not improbable that the better protection afforded to the boshes of recent years by means of these plates may have much to do with the increased regularity of work and improved economy shown by the latest furnaces.

I desire to call particular attention to the fact shown on the diagram from Fig. 6, 1885, to Fig. 8, 1889, that there has been no great increase in the amount of air blown into the furnace, and none whatever in the cubic capacity of the furnace; yet there has been an extraordinary increase in the average daily output and a considerable rise in the economy of coke. This would clearly indicate that the so-called rapid driving of recent American furnaces is not detrimental to economy of production, nor is there the slightest indication that better results would be obtained by slower driving.

J. D. Weeks, Pittsburgh, Pa: In answer to Mr. Whitwell's question as to whether the practice spoken of by Mr. Gayley is confined to Lake Superior ores and Connellsville coke, I would say that there is a little furnace in Virginia, the Ivanhoe, working on the

Fig. 10.



Ivanhoe Furnace, Virginia.

local ores of that region and on Pocahontas Flat Top coke, which has shown as good results, all things considered, as those reported in Mr. Gayley's paper as having been attained at the Edgar Thomson furnaces.

The Ivanhoe furnace, Fig. 10, the property of the New River Mineral Company, is situated at Ivanhoe Furnace, Wythe Co., Va. It was originally a charcoal-furnace, but was altered in 1888 to coke. The dimensions are as follows: Height of stack, 60 feet; diameter of boshes, 12 feet 6 inches; diameter of hearth, 7 feet 2 inches; stock-line, 9 feet 6 inches; number of tuyeres, 6; size of tuyeres, 5 inches; number of Whitwell fire-brick stoves, 2; size of stoves, 60 by 16 feet; number of blowing engines, 1; steam cylinder, 48 by 36 inches; blowing cylinder, 48 by 84 inches; displacement of engines, 308 cubic feet per revolution; capacity of furnace, 4200 cubic feet.

During the time mentioned in the following runs the furnace was burdened for mill-iron, but occasionally ran off on foundry. The ore-mixture for the furnace was $\frac{3}{4}$ Ivanhoe-ore, a limonite from near the furnace, and $\frac{1}{4}$ mountain-ore (Potsdam formation, No. I. of Rogers), of which the following is a fair analysis:

	Ivanhoe, Limonite Ore.	Mountain, Potsdam Ore.
	Per cent.	Per cent.
Metallic iron,	46.65	44.11
Silica,	15.75	15.30
Manganese,	.309	1.20
Phosphorus,	.067	.916

The limestone had the following composition:

	Per cent.
$CaCO_3$,	76.05
$MgCO_3$,	22.13
SiO_2,	1.26
Fe_2O_3 and Al_2O_3,	.65
	100.09

The average analysis of the Pocahontas coke is as follows:

	Per cent.
Water and volatile matter,	1.07
Fixed carbon,	91.77
Ash,	6.60
Sulphur,	.56
	100.00

The first run of the furnace of which we have a record was from July 13 to 27, 1890. The run per day was as follows:

	Tons.		Tons.
July 13,	65	July 21,	65
" 14,	60	" 22,	67
" 15,	68*	" 23,	70
" 16,	68	" 24,	60
" 17,	68	" 25,	64
" 18,	65	" 26,	75
" 19,	72	" 27,	64
" 20,	60		991

This gives a daily average of 66.06 tons. The condition of the furnace was as follows:

Average blast-temperature, 1276 degrees, Fahrenheit; average revolutions, 32, which gives 9824 cubic feet of piston displacement per minute; blast-pressure, 6.9 pounds per square inch; steam-pressure, 78.0 pounds per square inch.

During this period the average consumption of coke per ton of pig-iron was 1.075 short tons, or 2150 pounds of coke to 2260 pounds of iron.† The ore-mixture yielded 46.5 per cent. of iron. The average percentage of limestone used was 32.

The second run of which a record is given was from Aug. 31st to Sept. 6th, both inclusive. This run was as follows:

Record of Ivanhoe (Va.) Furnace for Seven Days in August and September, 1890.

	Tons.		Tons.
Aug. 31,	70	Sept. 4,	68
Sept. 1,	68	" 5,	67
" 2,	74	" 6,	64
" 3,	65		476

This was an average of 68 tons a day. The conditions of the furnace during this run were as follows:

Average blast-temperature, 1155 degrees, Fahrenheit; average revolutions of engine, 31, equivalent to 9517 cubic feet of piston displacement per minute; blast-pressure, 6.3 pounds per square inch; steam-pressure, 76.00 pounds per square inch.

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* Stopped 2 hours.

† An allowance of 20 pounds is made for sandage on each long ton of iron. All products of pig-iron are therefore given in tons of 2260 pounds.

During this period the consumption of fuel was 1.01 short tons, or 2020 pounds to each 2260 pounds of iron produced. The ore-mixture yielded 49 per cent. of iron. The furnace was burdened for mill-iron, but made 25 per cent. of the lower grades of foundry-iron.

The Ivanhoe pig-iron has the following composition:

	No. 1.	No. 2.	No. 3.	Gray Forge.
Silicon, per cent.,	2.026	1.113	1.026	.980
Manganese, " "	.884	.872	.871	.910
Phosphorus, " "	.622	.620	.564	.564
Sulphur, " "	.....	.....	.....	.040

All things considered, it would seem that a fuel-consumption of a little over 2000 pounds of coke to 2260 pounds of iron, made with an ore yielding less than 50 per cent. of iron in the furnace, and that ore a washed ore, carrying with it into the furnace a large amount of water, is fully as good work as that shown by the Edgar Thomson furnaces.

I have noticed in the discussion of this paper that the only comparisons which have been made between different cokes have been as to their chemical constituents. Connellsville coke has been compared with Durham coke simply by the percentage of ash. Now, I think that I am right in saying that while purity of coke is, within certain limits, a good thing, its physical characteristics also, within certain limits, are at least equally, if not more important. There is no question that of two fuels, the one which might be greatly inferior to the other in chemical characteristics might nevertheless be vastly superior to it as a blast-furnace fuel. The Pocahontas Flat Top coke, which shows a much lower percentage of ash than the Connellsville coke, is yet inferior to it in some respects as a blast-furnace fuel. It shows a greater waste in transportation, and does not appear to have the same power to resist the dissolving action of carbon dioxide in the top of the furnace. It has been my observation that cokes which are coated with graphite do not give hot tops in the furnace, and that where the silvery luster or coating of graphite is wanting, there appears to be a greater frequency of hot tops and a greater fuel-consumption. I have been hoping that Sir Lowthian Bell would explain to us how it is that a coke with a high percentage of ash will sometimes give better results in the furnace than a coke with lower ash.

Sir Lowthian Bell, Middlesbrough, England: In answer to Mr. Weeks, regarding the behavior of the coke in the blast-furnace,

I would say that chemical composition affords no solution to the question. Upon what it depends I cannot tell, but this I know, that different kinds of coke are more or less easily acted upon by hot carbonic acid. In the event of that action going on in the upper part of the furnace (which is the only place where it can occur), and the coke there being dissolved by carbonic acid, every unit of coke so consumed causes a direct loss of heat to the extent of 5600 calories, and, moreover, that coke, which ought to have been burned at the tuyeres, never arrives there, so that a still further loss is incurred. I have had many specimens of brilliant and apparently most excellent coke, which, when exposed to a current of carbonic acid in a heated tube, were more rapidly acted upon than other varieties of a less inviting character. I take it for granted that this depends altogether upon the mechanical character of the coke itself, and that the harder it is the more impervious it will be to the action of the hot furnace-gas.

E. P. Martin, Dowlais, South Wales: Some time ago when I took the management at Dowlais, we were making coke in the ordinary ovens, and without a very good system of coal-washing. That coke worked in the furnaces with a very bad yield—in fact it was put anywhere and got rid of as fast as possible. The same coal, thoroughly washed in a Coppée machine and thoroughly coked in Coppée ovens, is now doing work equal to the best South Wales coke made in the district, yet as far as luster and appearance go, it is not at all inviting. I, therefore, do not think that much depends upon whether the coke is lustrous or otherwise, provided it is hard and dense.

Mr. Gayley: The interest that has been manifested in the paper which I have presented is a matter of deep gratification to me, and I am sure that the discussion which has taken place will be of value to us all. On account of the extent of the discussion, I shall make but very few remarks. As to the shape of the furnace, the drawings simply represent types. We had but five furnaces at first to make comparisons with, but now we have nine. The general practice has been to decide on a type of furnace and then to repair, according to that type, throughout the whole line. I understand from Mr. Windsor Richards that in the future advancement in America, changes in the shape of the furnace will probably be in the direction of larger capacity, but our more recent experience at the Edgar Thomson works has been that we get a larger output and a higher fuel-economy by narrowing our furnace-stacks. This will be seen

by a reference to the last furnace that I mentioned, in which the cubic capacity has been reduced from 19,800 to 18,200 cubic feet, resulting in increased output and diminished coke-consumption. The tendency is, I think, in the direction of narrow stacks, and I agree with the statement presented as to the angle of the bosh, since that which I have found in my practice to give the best results is 75 degrees.

With regard to the quality of the ores, I do not know that we have any data showing the reducibility of the Lake Superior ores in comparison with the Cleveland ore, but that is a subject on which I hope to be able to present some results in the near future (see below). There is one point in our practice with regard to ores to which I would draw your special notice; we use at the Edgar Thomson works probably 18 different kinds of ore, and hence have a great variety to select from. In a furnace making regular Bessemer iron, we aim to select from these ore-piles those particular kinds which, by the demonstration of actual practice, give the best results when worked together; and our more recent practice shows that we are getting far better results by discarding the lump-ore and using fine ore exclusively. In fine ore, I include those Lake Superior ores which have a shaly structure.

I would also emphasize the careful selection of coke. We endeavor, as far as possible, to get our coke from mines where the quality is as uniform in appearance as possible, paying little attention to the chemical analysis but selecting the hard and silvery coke, and avoiding, as much as practicable, the softer coke, which, as has been shown by Sir Lowthian Bell, wastes away at the top of the furnace. We find the peculiar circumstance that during wet weather, or on days when there is a great deal of moisture in the atmosphere, the coke is of an inferior quality, and does not give us the same duty in the furnace as coke made in cold, dry weather. There is less density; it is very much softer, and it crumbles to a greater extent. I have found also, that where a great variety of coke is used from many different mines, we get far poorer results than when we make a selection from one or two.

In concluding, I may remark that every detail in the management of our blast-furnaces is most carefully considered. There is no point, however insignificant, that is not treated as if it were a matter of vital importance. Finally, I take pleasure in acknowledging the benefit we have derived from putting in practice the many valuable suggestions resulting from the laborious researches of Sir Lowthian

Bell, whose investigations have lifted the manufacture of pig-iron out of the rut of a rule-of-thumb practice, and placed it on a substantial scientific basis.

* The question of the comparative reducibility of Cleveland ore and the Lake ores of America has often been touched on in comparing English and American blast-furnace practice, and as it has always figured as an uncertain factor in such comparisons, I have had some experiments made which throw light on the subject. The ores selected for the tests are the ones principally used in the mixtures at various works for the production of Bessemer iron. The experiments were conducted by Mr. Grammer, in my laboratory, under precisely the same conditions, and with the same apparatus, etc., as are described in Sir Lowthian Bell's *Chemical Phenomena of Iron Smelting;* and, for the purpose of comparing with some definite experiments, it was decided to follow the conditions governing experiments 36, 37, etc., recorded on page 18 of that work. The quantity of CO passed through the apparatus was 63 liters in 6 hours; the temperature obtained was 780 to 790 degrees Fahr. The results are recorded below:

	Ore Numbers.												
	1	2	3	4	5	6	7	8	9	10	11	12	Average.
Per cent Fe in ore,.....	60.0	58.8	62.4	62.6	68.8	61.0	65.6	67.4	62.0	64.6	61.6	66.2	63.4
Per ct. original O. moved,....	48.0	46.4	41.4	40.2	38.7	31.7	29.5	29.4	23.1	14.7	12.7	10.3	30.5

From the preceding table it will be seen that the average removal of oxygen is 30.5 per cent. Comparing this with the results obtained by Bell, as recorded on page 18, we find:

Exp. 36.—Calcined Cleveland ore, containing 40 per cent. Fe. Original oxygen removed, 37.3 per cent.

Exp. 37.—Lancashire hematite containing 66.6 per cent. Fe. Original oxygen removed, 35.9 per cent.

This shows that the average Lake ore in America is somewhat more refractory than either the Cleveland or Lancashire ore. The ores above given are all used extensively, and are selected as representing an average mixture at many American blast-furnaces.

* Mr. Gayley's remarks from this point on were communicated subsequently to the Secretary.

JOHN BIRKINBINE, Philadelphia, Pa. (Communication to the Secretary): The data presented by Mr. Gayley in his paper and those brought out in the discussion, illustrate the practice of some of the more important American blast-furnace plants, and while the figures have assumed proportions which, as Sir Lowthian Bell says, are surprising, they are not relatively more so than some of the results obtained in the smaller furnaces of the United States. The average work of American blast-furnaces falls far below the record given by Mr. Gayley, but if the figures he offers are compared with results obtained elsewhere in proportion to the cubic capacity of the furnace, and due allowance be made for ores, fuel, temperature of blast, etc., some of the smaller furnaces will be found well up in the race. The demands upon American blast-furnace managers have tended to develop personal characteristics which have done more than anything else to advance them to the head of the profession. In discussing Mr. Gayley's paper I shall confine my remarks to

1st. The remarkable output and low fuel-consumption obtained in some of our smaller blast-furnaces, particularly among those using charcoal as fuel.

2d. The personal factor of management which, after all, is the true secret of the remarkable and unusual records which have been presented in Mr. Gayley's paper and in the discussion.

Few American blast-furnaces, using charcoal as fuel, reach the dimensions of the smaller furnaces using coke. The largest charcoal furnace in the United States has a bosh-diameter of 14 feet and a height of 60 feet; the next in size has a 12-foot bosh and a height of 65 feet. A mean size of the blast-furnaces now active, which use charcoal as fuel, will show a bosh-diameter of only 10 or 11 feet and a height of 50 feet. As a rule, iron-pipe stoves are in use, but fire-brick stoves are connected with four or five plants, while a few furnaces are still operated with cold blast.

Without burdening the discussion with data obtained from many furnaces, the results secured at two charcoal blast-furnaces are presented.

The first of these is the Spring Lake furnace, on the eastern peninsula of Michigan. The stack is 46 feet high, with a bosh $10\frac{1}{2}$ feet in diameter, a 5-foot crucible and a 5-foot bell. It is blown by a vertical engine with air-cylinder of 5 feet diameter and 4 feet stroke. The blast is heated in iron-pipe stoves to 850° F., and enters the furnace through four $3\frac{1}{2}$-inch tuyeres. The average yield of the ore-mix-

ture, which consists principally of soft hematite from the Lake Superior region, is from 58 per cent. to 61 per cent. of iron. This furnace was started on March 4th, 1880, and up to September, 1889, a total period of 3471 days, it had been operated 3205 days or about $8\frac{3}{4}$ years. Complete data subsequent to September, 1889, are not now at hand, but the following table will illustrate the duration of the different campaigns and the results obtained in each.

The iron is weighed from the furnace at 2260 pounds to the ton, being an allowance for sandage of 20 pounds for each 2240 pounds. The charcoal is charged by measure, 2748 cubic inches to the bushel, the weight averaging 20 pounds. To the table, as presented by Mr. J. C. Ford, the manager, have been added the calculated weights of charcoal per ton of pig-iron:

Record of the Spring Lake Furnace, Michigan.

	Total Days.	Stops, Days.	Net Days.	Gross Tons of Pig-iron made.	Tons of Pig-iron per day.	Bushels of Charcoal per Ton of Iron.	Pounds of Charcoal per Ton of Iron. (Calculated.)
1st Blast,	677	7	670	29,040	43	94	1880
2d "	799	7	792	33,124	42	95	1900
3d "	331	11	320	15,279	48	89	1780
4th "	510	14	496	26,867	54	88	1760
5th "	586	10	576	29,237	51	92	1840
6th "	361	10	351	20,451	58	95	1900
Totals and averages,	3264	59	3205	153,998	48	92.2	1844

This table shows that there was an average product of 48 gross tons per day, and a total consumption of 92.2 bushels or 1844 pounds of charcoal per ton of iron made.

In the first year of operation, this furnace made for one week a ton of pig-iron on 1715 pounds of charcoal.

The second example is the Hinkle furnace, at Ashland, Wis., on the shores of Lake Superior, using only the soft red hematite of the Gogebic range. The furnace is 60 feet high, with a bosh-diameter of 12 feet, and is equipped with two fire-brick stoves, 18 by 60 feet, and with a vertical blowing-engine having an air-cylinder of 6 feet diameter and 4 feet stroke. Mr. Morris R. Hunt, the manager, has supplied me with the following information:

The furnace was blown in on April 4, 1888, and blown out on June 30, 1888. During this period it made 3261 gross tons of pig-

iron. The actual number of working-days was 73, and average daily output 44.67 gross tons of pig iron.

It was again blown in on August 21, 1888, and blown out on August 1, 1889. During this period, under Mr. Hunt's management, it made 26,137 gross tons of pig-iron. The actual working-days numbered 316, and average daily make 82.71 gross tons of pig-iron.

The furnace was again blown in on September 9, 1889, and in the interval of 14½ months just passed has made the following remarkable record:

Total length of campaign, 443 days.

Actual running time, 382 days.

Total output for blast, 41,600 gross tons.

An average of 109 tons per day of actual running time.

The yield of ore was 55 per cent. of iron.

The average consumption of charcoal per ton of iron, 1815 pounds.

The average consumption of limestone per ton of iron, 177 pounds.

The cubic capacity of the furnace is 3500 cubic feet.

A volume of 6500 cubic feet of air per minute was blown into the furnace through six 5-inch tuyeres at a pressure of about 5½ pounds and a temperature of about 1300° Fahr. The best week's work was made during the visit of the Institute to the United States, when 1009 tons of pig-iron were produced on an average fuel-consumption of 1806 pounds of charcoal. During that week the ores yielded 56.4 per cent. of iron, and the volume of air blown into the furnace per minute amounted to 9300 cubic feet at a temperature of 1300°. The limestone-burden was 4½ per cent. of the ore-burden. The composition of the cinder at the Ashland furnace is reported to vary from 46 to 51 per cent. of silica and 11 to 15 per cent. of alumina.

In each of the different classes of blast-furnaces—that is, in the furnaces using anthracite coal, coke or charcoal, and in most of the iron-producing districts—a few plants have attained prominence by reason of large outputs and low fuel-consumption or both, and some of the works using lean ores, or ores difficult to smelt, are doing almost as remarkable work as some others whose material or facilities give them advantages. It is not always that the finest equipped plants achieve the best results; on the contrary, quite a number of the blast-furnace managers, who have attained enviable reputations, have furnaces with but moderate equipments, and the credit of

unusual output or low fuel-consumption seldom rests with the plants which have descended as legacies from father to son. A majority of the blast-furnaces which have brought credit to the pig-iron industry of the United States are managed by well-educated men who have graduated from the office, the laboratory, or some other department connected with the industry. A visit to the plants which have been most successful reveals a thoroughness of management and care as to detail, which impresses a stranger at once with the conviction that there is good reason for the results. The factor of personal management as affecting the output and returns of a blast-furnace are too often overlooked, and even when credit is awarded for it, the full importance of such management is not always appreciated.

Constant vigilance, strict attention to details, and a thorough familiarity with all the various features of the plant are necessary; and in addition to these an acquaintance with the chemical and physical characteristics of the materials charged into the furnace, as well as with the composition, structure and suitableness of the product for all accessible markets are essentials of good furnace management. But the phenomenal results which have been obtained are largely traceable to a confidence born of a realization of competence to handle the plant, together with prompt and aggressive action in emergencies.

FUEL-GAS AND SOME OF ITS APPLICATIONS.

BY BURDETT LOOMIS, HARTFORD, CONN.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

THE advantages of gaseous fuel have been too fully set forth during the past few years to need recapitulation. Papers, arguments, and editorials have been issued, in a more or less general way, that would make volumes. Systems have found zealous advocates, and many promises of results that have sounded strangely in the ears of old-fashioned gas-makers, have been born of theory, and have died in the earliest throes of trial; while in some few instances success has demonstrated the value of practical effort. With few exceptions, however, everything attempted or accomplished has tended towards the one end—the realization of more perfect results than we can

Fig. 1.
VERTICAL SECTION ON LINE C.D.

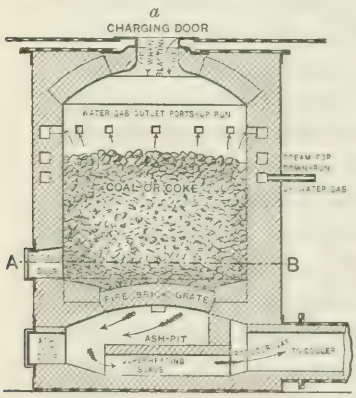


Fig. 2.
ELEVATION

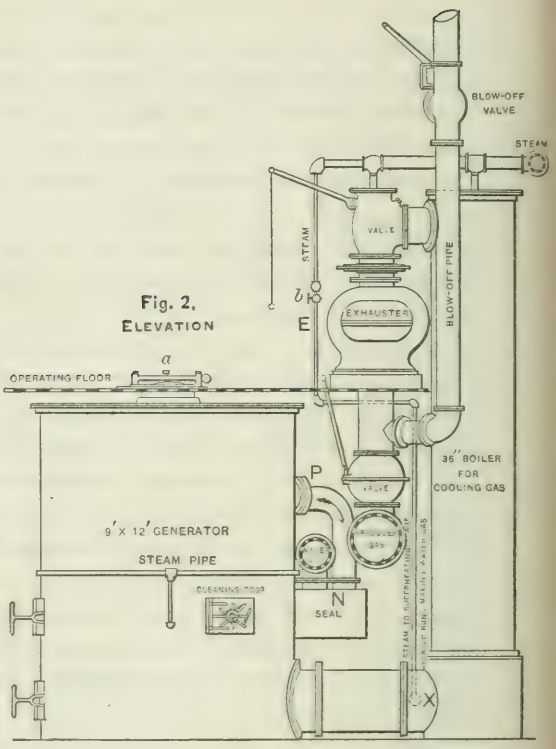


Fig. 3.
HORIZONTAL SECTION ON LINE A.B.

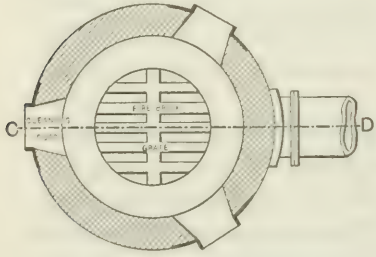
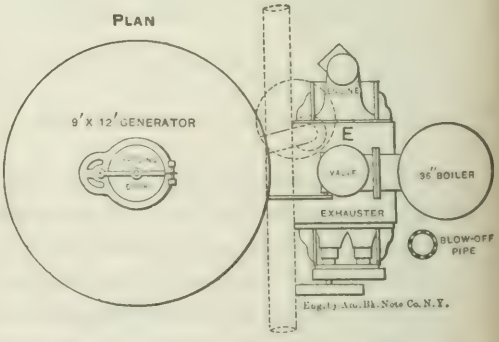


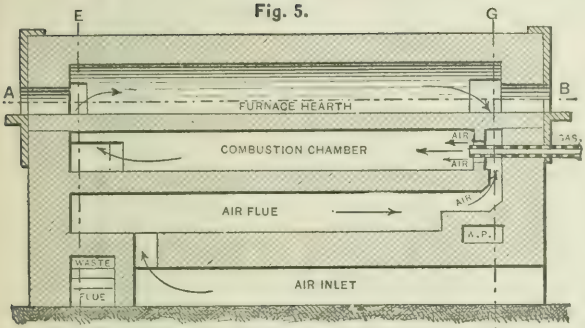
Fig. 4.
PLAN



Scale
Inches 12 6 0 1 2 3 4 5 10 15 20 Feet

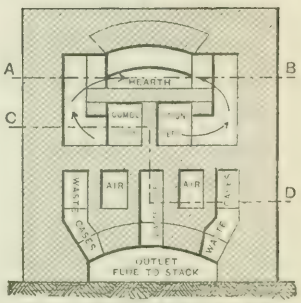
LOOMIS GAS-GENERATING PLANT.

Fig. 5.



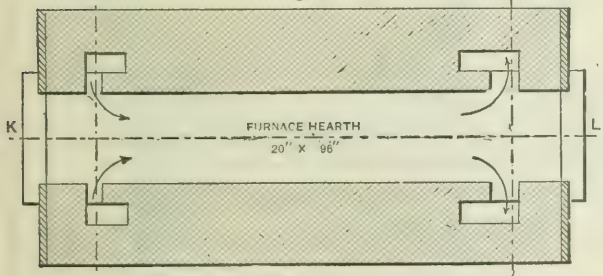
VERTICAL SECTION ON LINE K L.

Fig. 6.



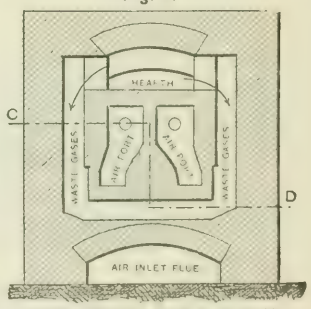
CROSS SECTION ON LINE E F.

Fig. 7.



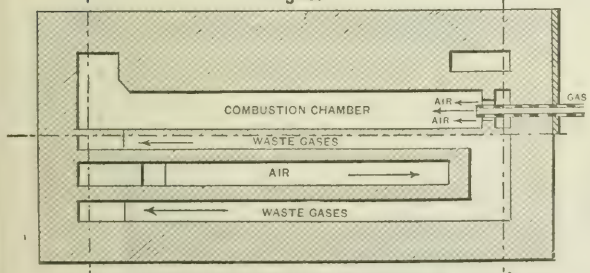
HORIZONTAL SECTION ON LINE A B.

Fig. 8.



CROSS SECTION ON LINE G H.

Fig. 9.



F HORIZONTAL SECTION ON LINE C D. H

SMALL HOT-BLAST ROD-FURNACE, HEATED WITH WATER-GAS

CONSUMPTION 500 CUBIC FEET PER HOUR.

Scale of Feet.

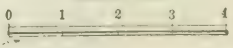
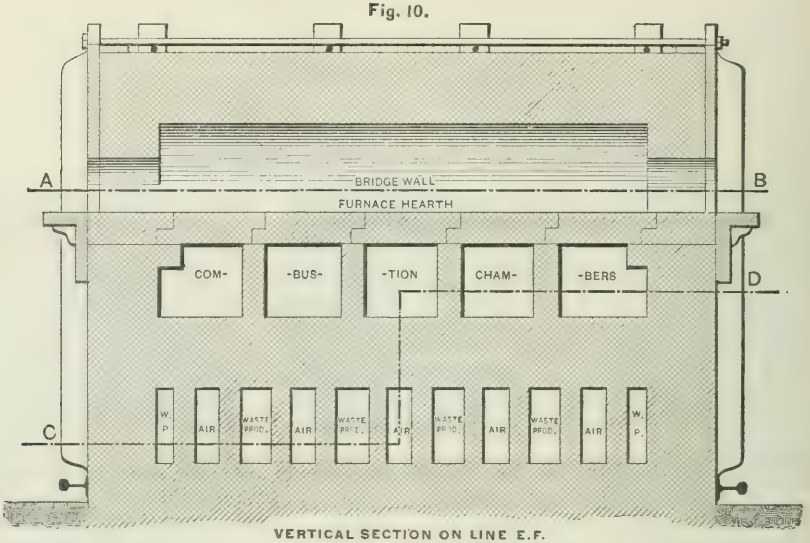
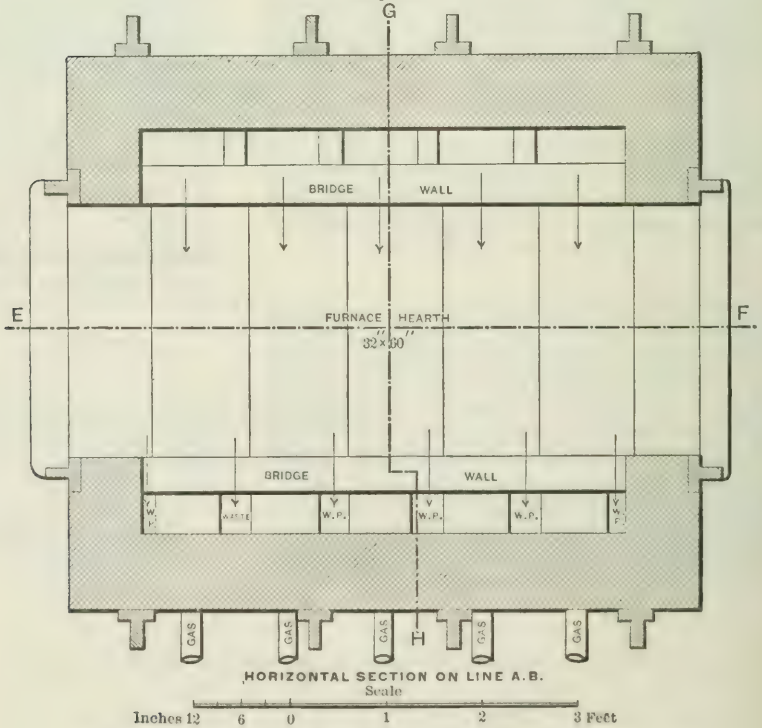


Fig. 10.



VERTICAL SECTION ON LINE E.F.

Fig 11.



HORIZONTAL SECTION ON LINE A.B.

Scale

Inches 12 6 0 1 2 3 Feet

REHEATING HOT-BLAST FURNACE, FIRED WITH WATER-GAS,

Consumption 650 Cubic Feet per Hour.

Figs. 10 to 13.

Engraved by American Bank Note Co., New York.

obtain with the direct use of solid fuels; and there are to-day few to dispute the coming supremacy and general use of gaseous fuel.

It is not assumed that the solid fuels will be wholly superseded; such an assumption would be folly, as there are circumstances in which solid fuels must continue in use, from necessity or economy. These, however, are few, and can almost be named in a breath; while the purposes for which the gaseous fuel is not only cheapest, but best, are so many as almost to defy enumeration. Wherever constant high flame-temperature is necessary, the gaseous form must find precedence, while its easy carriage to long distances and its attainment of maximum temperature almost instantaneously with ignition, give it a leverage that is scarcely covered by that easy term—convenience.

The discovery and application of natural gas has not only borne the weightiest of testimony on behalf of the general necessity for fuel-gas in nearly all fields, but it has done more. Its wide scope of service, from the great establishments wherein thousands use it constantly, to the cottage where it saves countless steps for the housewife, has demonstrated the necessity for a gas of general application, that will fulfil all the demands of the manufacturer, and meet the wants of the domestic user. The fuel gas that is to score a wide success must fill these conditions. For a gas that will not, the field is limited.

In the manufacture of a fuel-gas, then, it is obvious that certain conditions are primarily necessary.

As concerns the apparatus to be used, these are:

1. *Simplicity.*—It must be easily and cheaply worked, and require a mimimum of skilled labor.

2. *Durability.*—It must do its work with but little repair or replacement.

As to the material to be used in the manufacture of the gas, the conditions are:

1. *Richness.*—It must contain the necessary carbon.

2. *Cheapness.*—It must be generally obtainable in large quantities at low prices.

These qualifications are met by the slack or fine bituminous coal, which is the least expensive of all coals, and is obtainable in nearly all markets at low rates.

These points established, it remains to decide upon the kind of gas to be made. This must be calculated to meet all the wants of

those for whose uses it is intended. Here we find it necessary to consider the following:

1. *Quality.*—The gas must contain all the carbon possible, consistent with the other necessary conditions.

2. *Cost.*—It must be low in cost, especially in view of its use in large quantities.

3. *Cleanliness.*—It must be as free as practicable from impurities, and must be clean as regards the more perceptible.

4. *Transmission.*—It must be capable of easy carriage to reasonably long distances, and through small, as well as large, pipes.

5. *Temperature.*—It must have a constant high flame-temperature, and must ignite under all conditions, when desired.

For general adaptability to the wants of all classes of consumers, the requirements named must be met, and the more completely they are met, the greater will be the success of the gas offered.

In many cases, a good producer-gas accomplishes all that is needed, and, used in a well-constructed regenerative furnace, is much cheaper than coal; but its usefulness is confined to places where such furnaces can be constructed, if high flame-temperature is required, or where low heats are sufficient, and obtained without regeneration. Outside of such confines, it is not economical, and is seldom useful.

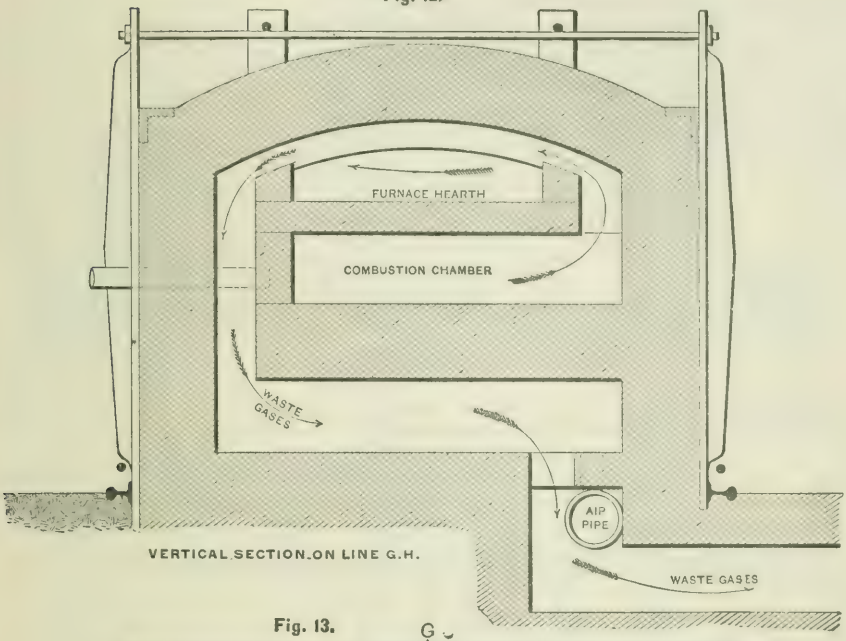
Without discoursing on the relative merits of other gases, I believe that the gas best suited for general fuel-purposes is water-gas. Its flame-temperature is high, it is cleanly, reasonably free from impurities, and it can be transmitted to long distances.

The one possible factor to which objection might be made is its cost. If it meets this requirement, there is no further bar to acknowledgment of its claims. Can it be made from the cheapest of materials offered in the general market for gas-making purposes? In discussing the lions in the path of success in this direction, the *London Journal of Gas Lighting*, some months ago, pronounced the following ultimatum:

"The slack, however wet, dirty, dusty, sulphurous, or stony, should be tipped into the hoppers of the gas-making cupolas, and gasification should go on continuously and completely, so that nothing but gas and ashes should remain of the material put into the apparatus."

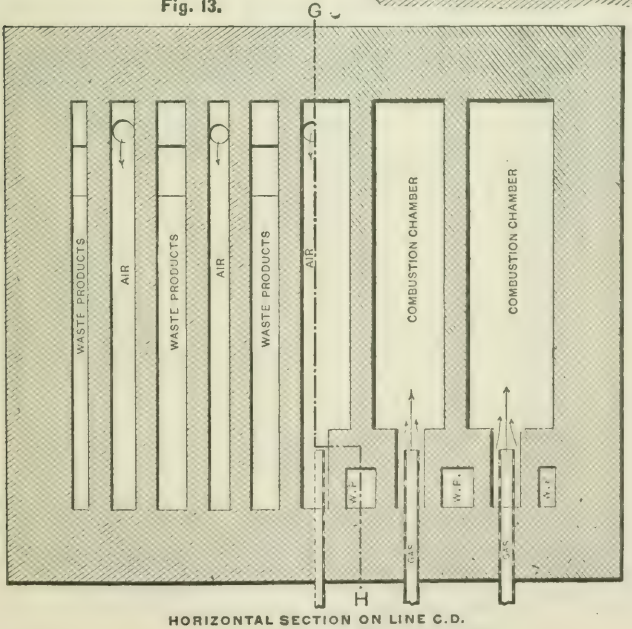
This can be done, and is done, in many places, with the Loomis apparatus, which is using the cheap coal mentioned, as well as the more expensive anthracite coals and coke. This apparatus is the only one that I know of in successful operation making water-gas

Fig. 12.



VERTICAL SECTION ON LINE G.H.

Fig. 13.



HORIZONTAL SECTION ON LINE C.D.

Scale

Inches 12 6 0 1 2 3 Feet

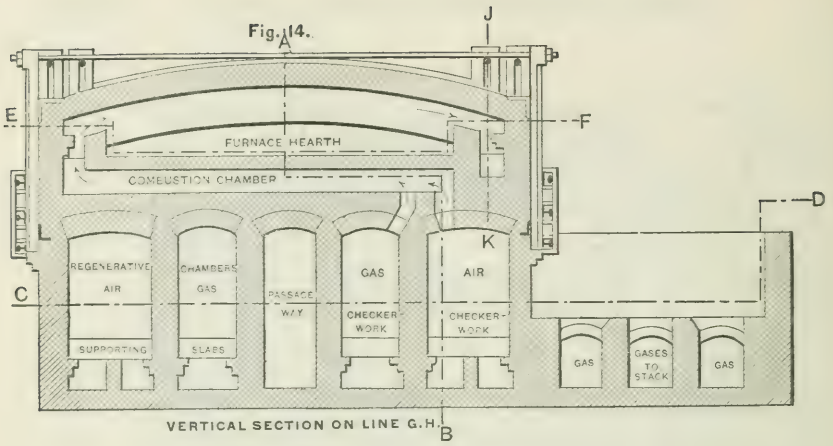
REHEATING HOT-BLAST FURNACE, FIRED WITH WATER-GAS,

Consumption 650 Cubic Feet per Hour.

Figs. 10 to 13.

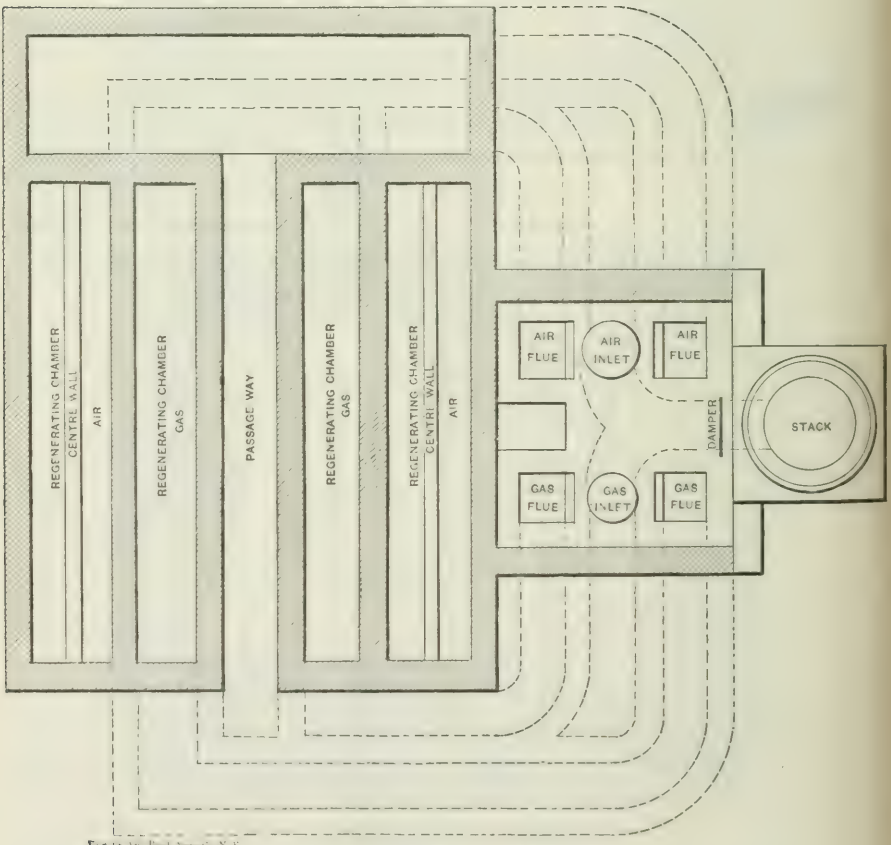
Engraved by American Bank Note Co., New York.

Fig. 14.



VERTICAL SECTION ON LINE G.H.

Fig. 15.



Engd by Am. Bank Note Co. N.Y.

HORIZONTAL SECTION ON LINE C.D.

Scale

Inches 12 6 0 1 2 3 4 5 10 15 20 Feet

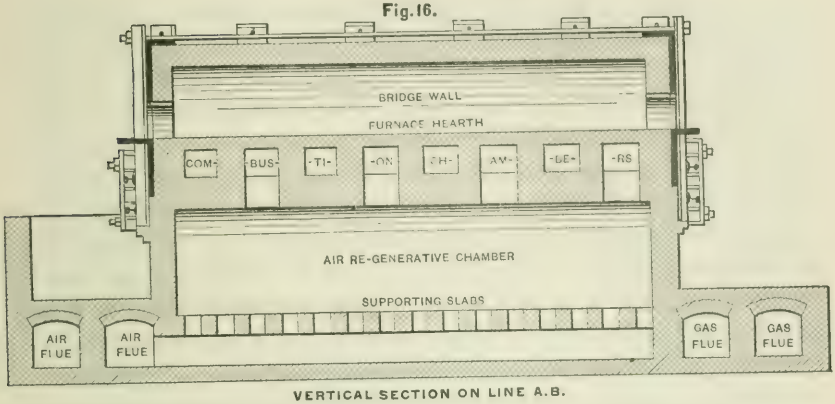
FURNACE FOR REHEATING STEEL PLATES,

FIRED WITH REGENERATED PRODUCER-GAS

at Works of Henry Disston's Sons, Tacony, Pa.

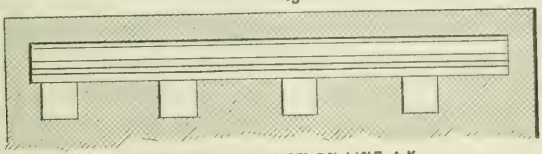
Figs. 14 to 18.

Fig. 16.



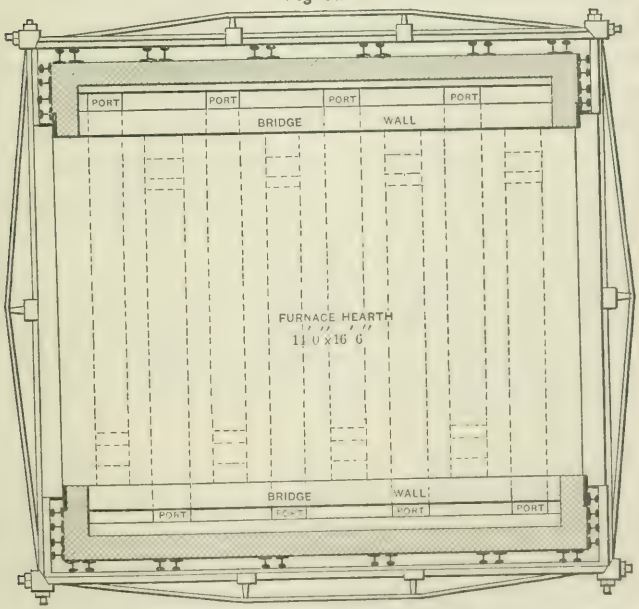
VERTICAL SECTION ON LINE A.B.

Fig. 17.



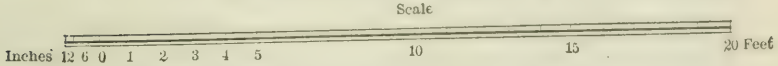
VERTICAL SECTION ON LINE J.K.

Fig. 18.



HORIZONTAL SECTION ON LINE E.F.

Scale



FURNACE FOR REHEATING STEEL PLATES,
FIRED WITH REGENERATED PRODUCER-GAS
at Works of Henry Disston's Sons, Tacony, Pa.
Figs. 14 to 18.

directly from the bituminous slack-coals that are found in abundance in many parts of the world. It is a simple fuel-gas-generating machine, and it will not only make as much water-gas from a ton of this slack-coal as can be produced by any other method from the same quantity of anthracite or coke, but it also utilizes the blast-gases for making the steam necessary to run the engines and exhausters, and for decomposing, leaving a surplus for other work; or the blast producer-gas or generator-gas can be used in regenerative, reheating-, annealing-, melting-furnaces, etc. For these purposes, the producer-gas made by this apparatus answers all requirements, as can be shown by demonstration in the several places where it is so used.

Water-gas generators, to be economical producers of gas, must be so constructed that they will continuously make a first class quality of gas for the full period of a twenty-four hours' run. In all the generators using the up-blast system, after a few hours' run the quality of the gas becomes much poorer than at the beginning, when the fires are fresh, on account of the clinkers and ashes that accumulate.

In the Loomis generator this is obviated, and the gas is of as good quality at the end of a twenty-four hours' run as at the beginning or any part of the run. This is due to the peculiar construction of the generator, and the action of the exhauster in collecting the ashes at the bottom of the fire and in the ash-pit. In the process of exhausting before making a run of water-gas, this is highly heated and useful for superheating the steam before it strikes the incandescent coal, and in this way the carbon is nearly all extracted from the coal, less clinker is made, the gas is freer from carbonic acid, and increased production is obtained.

In the matter of quality, a high grade of product should be aimed at, and the water-gas made as pure as possible, so that it can be used, where practicable, without purification. In works where it must be purified, the better the quality of the gas the less the expense for purifying.

The claim has been made that the water-gas that can be made from a given amount of coal must be less economical than the coal used directly, both on account of the smaller number of units of heat contained in the gas and the labor and expense of making. Where coal can be well fired, and gases economically burned direct from the furnace, this is in many instances true; but the great fields for water-gas lie in the special uses for the many kinds of metallurgical work and kindred work where coal directly used cannot enter as a comparative factor, and for domestic purposes, where the gas is

proving itself as cheap as the coal, if not cheaper, while it accomplishes results that are not obtainable with the solid fuel.

In estimating the economy of its general uses, another factor must be considered. This is application. In this particular, the threshold of investigation has scarcely been crossed, yet much has been accomplished, while the future is fruitful of certain realization. Until lately, the gas has not been obtainable for the purposes of experimenting in large quantities, and as yet, in special work, the methods of application are comparatively crude. Within the past few years, however, much has been done in the face of untoward circumstances, and you can now be shown the gas in use in many ways where it has supplanted coal, and wherein it has been found more economical than to fire coal, without taking into consideration any other features of advantage.

In nearly all branches of metallurgical work wherein the gas can be applied, however, there is found;

1. An increased product.
2. A better grade of product.
3. Direct economy in cost.

Increase in amount of product arises from the saving in direct labor, as can be most easily seen, perhaps, in forge shops and the like, and from the gain in the personal condition of the workmen, owing to the increased healthfulness of their surroundings.

The gain in grade of product arises chiefly from the uniformity in the temperature and the quality of the fuel.

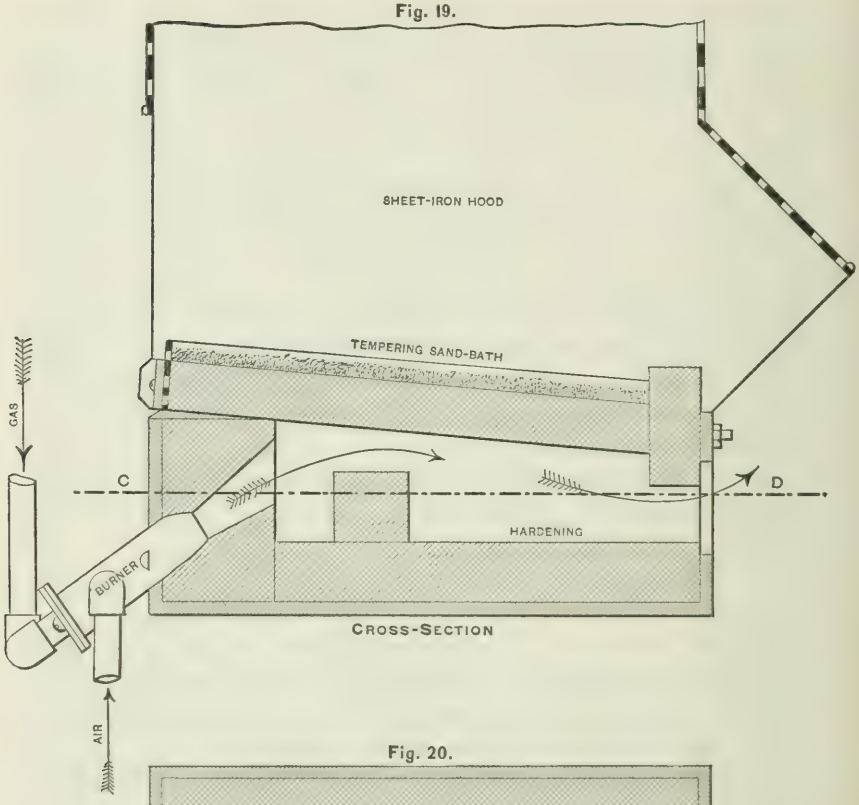
The economy in cost needs no explanation. I may state, however, that in some large works which have changed from coal to the use of fuel gas, the gain in direct economy has been proven to be from $33\frac{1}{3}$ to 50 per cent. in labor, and over 40 per cent. in fuel.

In many instances the gain from having a sharp, quickly heating flame, is such as to overbalance the question of more direct saving in cost of production, and it is not infrequently demonstrated that, by this means, workshop material enough is saved to alone pay all the productive cost of the gas.

Another feature worthy of consideration is economy of space in locating furnaces or other appliances for the use of the gas, especially where it is desirable that these should be grouped together as closely as possible. It also renders possible the location of furnaces, heaters, forges, etc., just where desired, without necessity for considering the questions of convenience that frequently arise where there must be either coal carriage or storage.

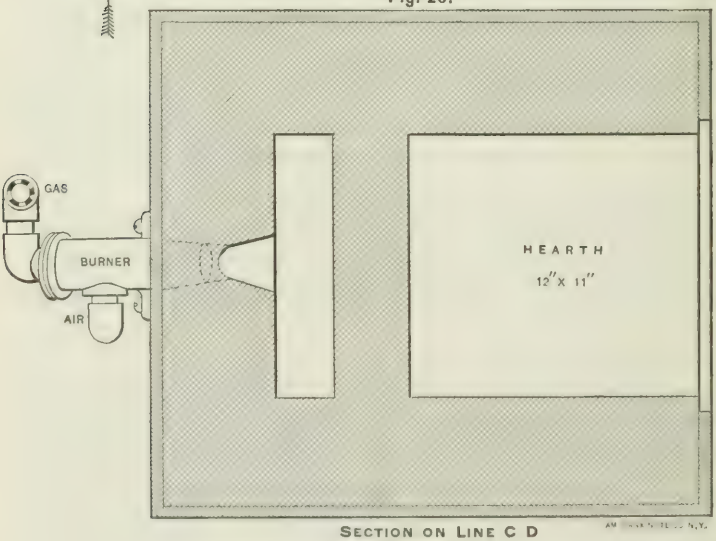
With this, too, may be considered the fact that the heat from the

Fig. 19.



CROSS-SECTION

Fig. 20.



SECTION ON LINE C D

Scale:

Ins. 6 3 0 1 Foot.

TEMPERING-FURNACE, FIRED WITH WATER-GAS,

CONSUMPTION, 150 CUBIC FEET PER HOUR.

Fig. 21.

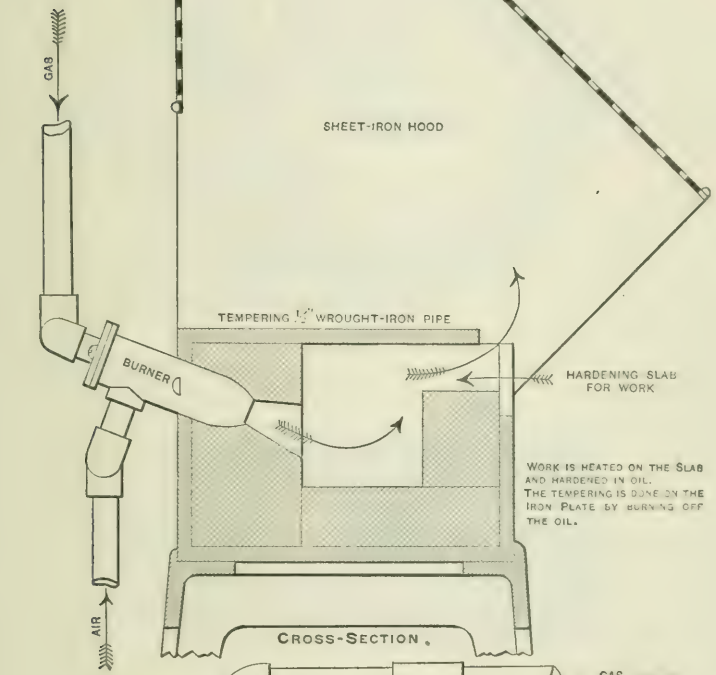
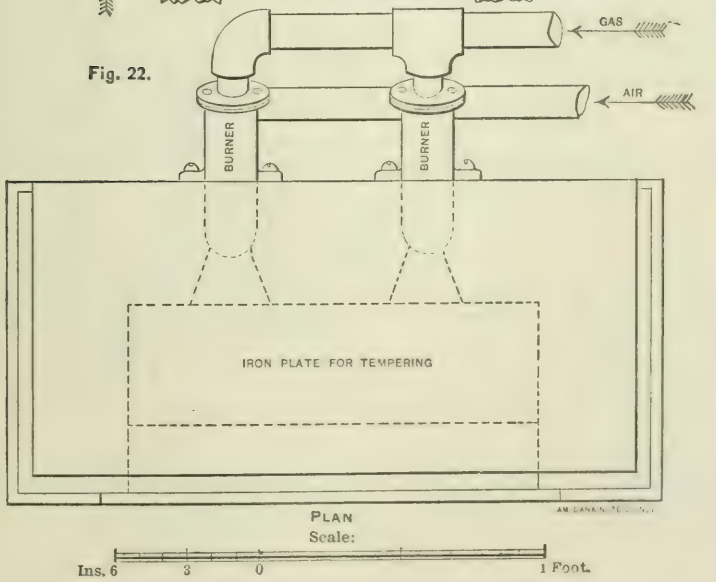

GAS
SHEET-IRON HOOD
TEMPERING ½" WROUGHT-IRON PIPE
BURNER
HARDENING SLAB FOR WORK
WORK IS HEATED ON THE SLAB AND HARDENED IN OIL. THE TEMPERING IS DONE ON THE IRON PLATE BY BURNING OFF THE OIL.
AIR
CROSS-SECTION.

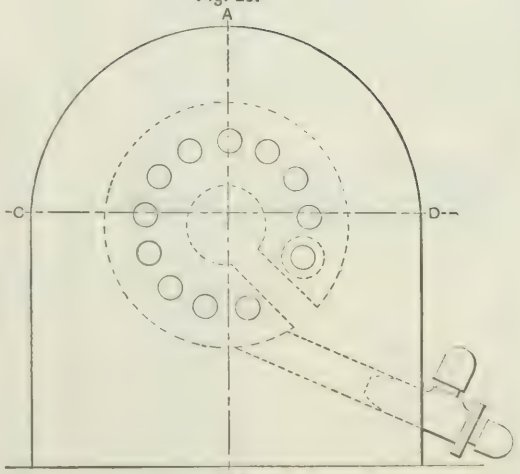
Fig. 22.



TEMPERING-FURNACE, FIRED WITH WATER-GAS,

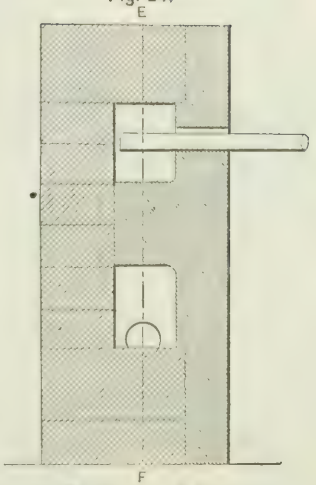
CONSUMPTION 250 CUBIC FEET PER HOUR.

Fig. 23.



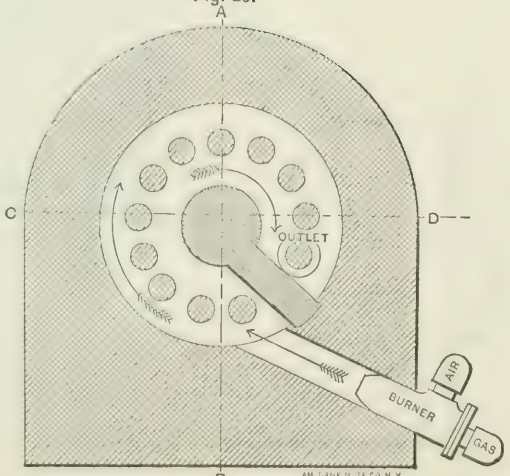
ELEVATION

Fig. 24.



SECTION ON A B

Fig. 25.



SECTION ON E F

Inches 6 3 0 1 Foot.

CIRCULAR-FLAME TIP-HEATING ROD-FORGE, USING 150 FEET OF WATER-GAS PER HOUR.

gas is brought under closer control than that from the coal fires, and there is less heat radiated into the room or space surrounding the point of application. This, perhaps, may seem of minor importance, but in these days of sanitary investigation and of shortening the days of labor, it will be found no little advantage to keep the temperature of the workroom as far as possible under control.

In many parts of the world inventors are devising new applications of this gas, and with marked success. It is so well made and so easily handled, that its application in many new channels amounts to little more than ordinary manipulation and direction by those who understand the work to be done.

Among the several places in this country where the gas made by the process I have named has been applied to such a variety of uses as plainly shows its adaptability as a general fuel, as regards both efficiency and cost, are the saw-works of Henry Disston's Sons, which you will be invited to examine on your journey from New York to Philadelphia. There you will see the generators at work on steady and continued runs, and will see the two gases, *i.e.*, the water-gas, and the producer-gas that is made intermittently with the water-gas, both carried to holders, and delivered under pressure at different parts of the works.

This is the first place of which I have knowledge where producer-gas has been stored in a holder and thence delivered cold to different furnaces at distances from the generators, being fired cold at such distant points. The process of manufacture and the manifold utilization of the different gases, but especially of the water-gas, will be of interest. Full facilities for a thorough examination of the process of manufacture will be given, as well as full liberty to make any tests or experiments desired.

Estimated cost of 1,000,000 *feet of water-gas, based on actual results with coal at* $3.00 *per ton.*

Coal, 25 tons, $3.00 per ton,	$75.00
Coal for steam, 3 tons, $3.00 per ton,	9.00
Labor,	22.00
Extra for supplies and repairs,	4.00
Purifying,	5.00
	$115.00
Received for producer-gas,	40.00
	$75.00
Interest on plant and depreciation,	25.00
	$100.00

Cost per 1000 feet, $0.10.

In the practical using of water-gas, 20,000 feet often accomplishes better results than a ton of coal directly fired; in some cases more is used.

This has been accomplished by new and improved methods of application, and will be improved upon as necessity demands and appliances are perfected.

I may say, in conclusion, that in no place where this gas has been put into use has any thought been given to a return to coal. The solid fuel, once supplanted, will never be sought for again.

Description of the Loomis Generator.

Figures 1 to 4 show the way in which producer-gas and water-gas are made alternately.

In bringing the coal to an incandescent state for making water-gas, the door is open, and the exhauster is started and draws the air down through the coal and ash-pit and up through the cooling boiler, and the producer-gas made by this operation is forced through a pipe to the holder.

When the coal is in a proper state for decomposition of steam the exhauster is stopped, and the door shut, and steam is let on by the valve-cock, *b*, at point X, and it passes through the ash-pit and grates, where it is highly superheated, then through the mass of coal, making water-gas, which passes out at P through the seal box N to the water-gas holder.

After a run of about five minutes the steam is shut off, the door opened, the exhauster started, and the process of making producer-gas is again commenced.

Since the foregoing paper was written, the following actual results have been obtained in melting brass in crucibles with water-gas with improved furnace:

In melting 2000 lbs. of brass in 100-lb. crucibles, 12,000 cubic feet of water-gas was consumed.

In using same size of crucibles in works that are melting from five to ten tons per day with coal, it takes 2000 lbs. of coal for 2000 lbs. of brass.

Units of heat in 12,000 feet water-gas used, 3,600,000.

Units of heat in 2000 lbs. coal used, 27,000,000; or $7\frac{1}{2}$ times the amount that is in the gas used.

One ton of coal will make 40,000 cubic feet of water-gas, which will accomplish as much as $3\frac{1}{3}$ tons of coal in this work.

Fig. 26.

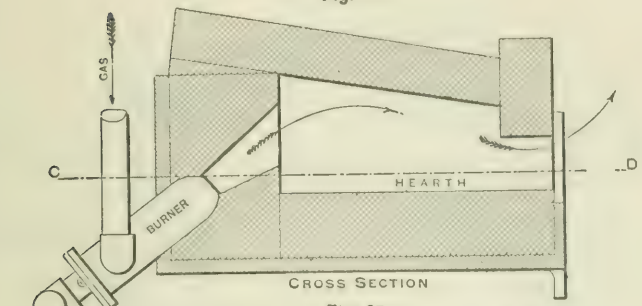


Fig. 27.

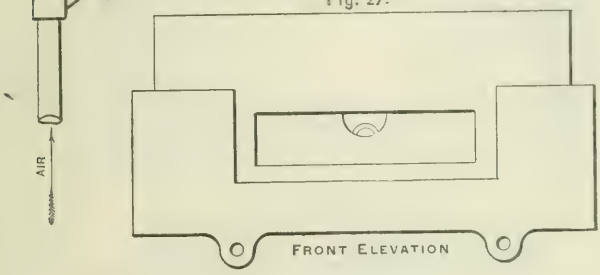
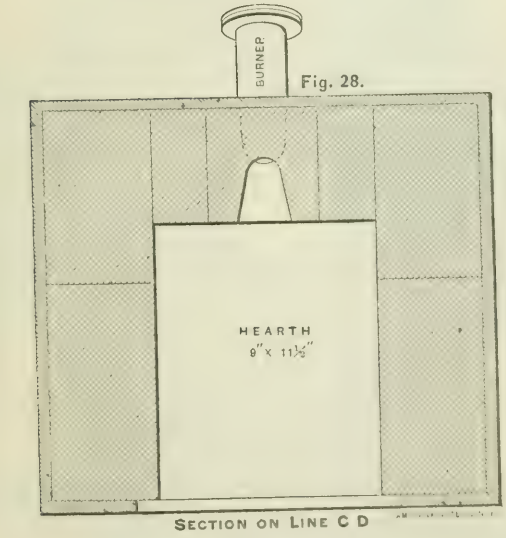


Fig. 28.



Scale

Inches 6 3 0 1 Foot.

FORGE No. 3.

USING 150 FEET OF WATER-GAS PER HOUR.

Fig. 29.

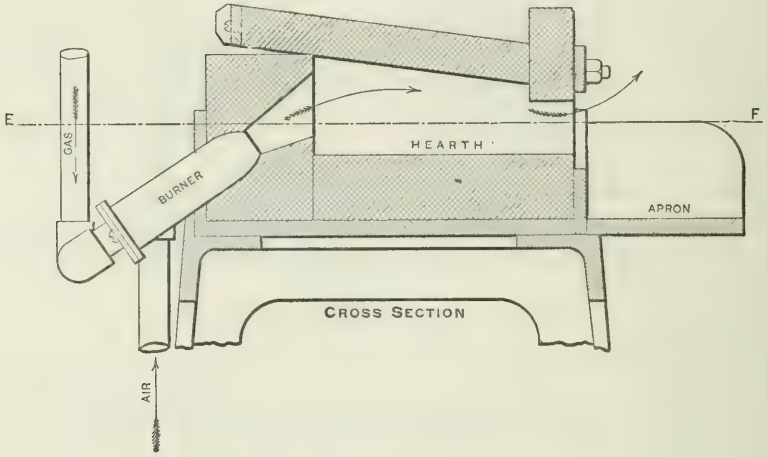
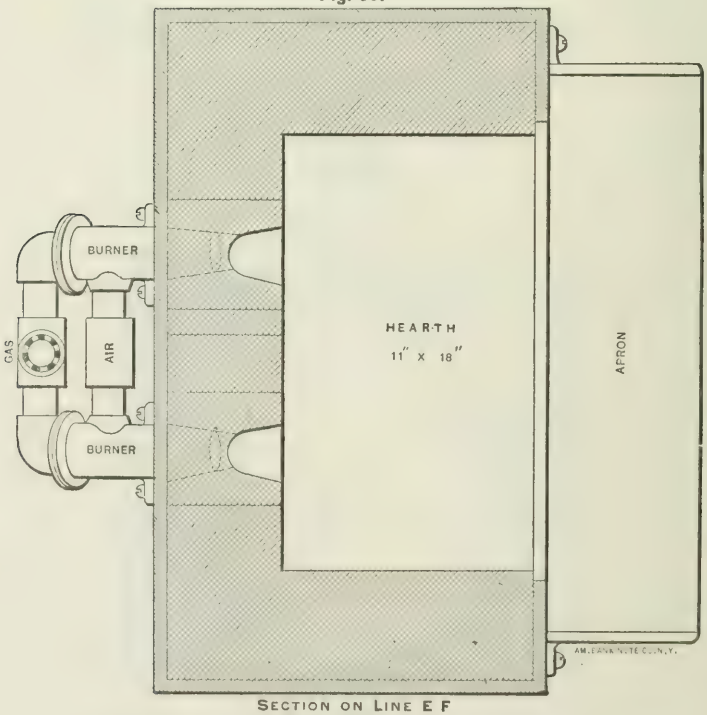
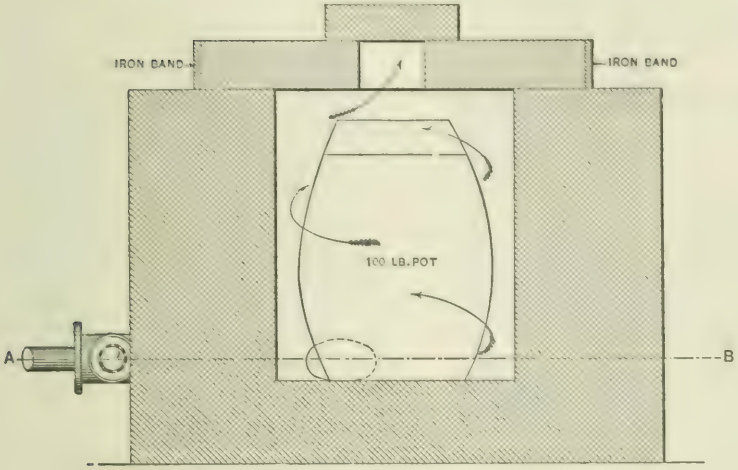


Fig. 30.



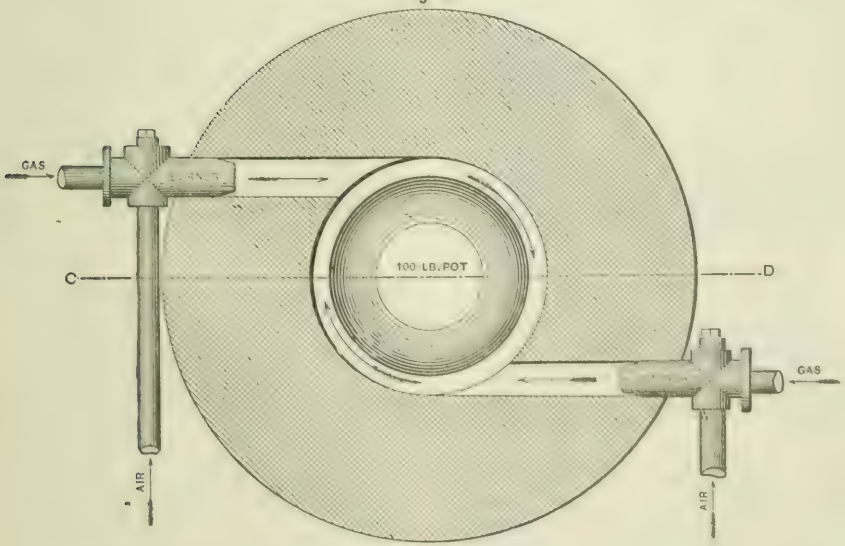
FORGE NO. 4,
USING 250 FEET OF WATER-GAS PER HOUR.

Fig. 31.



SECTION ON LINE C D

Fig. 32.

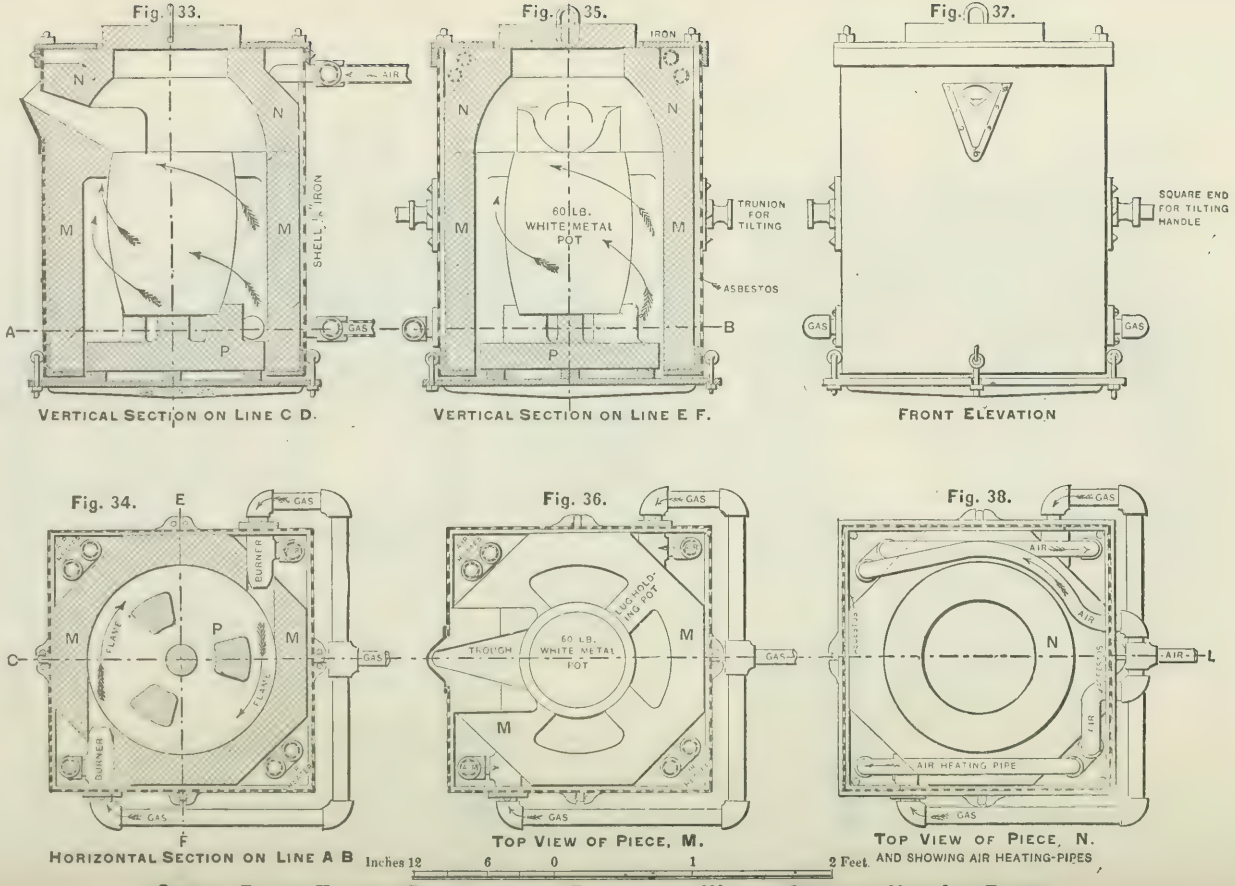


SECTION ON LINE A B

Scale

Inches. 12 6 0 1 2 Feet.

SPIRAL-FLAME POT-FURNACE,
FIRED WITH WATER-GAS.



SPIRAL-FLAME TILTING POT-FURNACE, FIRED WITH WATER-GAS AND HOT AIR-BLAST.

The cost of labor in making water-gas will be less than in firing coal direct. The producer-gas made, in addition to 40,000 feet of water-gas from a ton of coal, will yield all the steam needed and pay interest and repairs on the plant.

The coal used for making the gas, being bituminous slack, only costs $\frac{2}{3}$ as much per ton as that used with direct firing.

One ton of coal to make the gas costs $3.00.

$3\frac{1}{3}$ tons at $4.50 per ton cost $13.50.

Saving on coal in melting $3\frac{1}{3}$ tons, $10.50.

As it only requires $\frac{2}{3}$ the time to melt with gas that it does with coal, here is a great saving, and less space is needed for furnaces, as only $\frac{1}{3}$ the number are needed. Metal is not wasted in the ash, and crucibles will last much longer.

In estimating comparative cost of labor and fuel in making water-gas from bituminous or anthracite coal there is in the Eastern States and England a difference of about $1.50 per ton between the cost of bituminous slack, and coke or anthracite, and the estimate will be made on that basis. Where it is more or less, allowance can be made.

2000 lbs. Cumberland slack will make 40,000 cubic feet of water-gas, with 340 heat-units per foot, or 13,600,000 per ton.

Cost of coal,	$3.00
Cost of labor per ton,	1.00
Producer-gas makes steam,	
Total,	$4.00

Making 1,000,000 heat-units costs about 30 cents. If the balance of producer-gas can be utilized after making steam it will reduce the cost of the water-gas as low as 25 cents.

2000 lbs. anthracite coal or coke:

Cost per ton of coal,	$4.50
Cost of labor per ton of coal,	1.00
Coal for steam,	0.75
Total,	$6.25

If it makes 40,000 cubic feet at 300 heat-units per foot, or 12,000,000 per ton, 1,000,000 heat-units would cost 52 cents, or nearly double that made from bituminous slack. The extra units of heat in gas made from bituminous coal is obtained from the marsh-gas and illuminants, which give the gas a decided odor.

The members of the Institute are invited to visit the works of the Waltham Watch Co., where they will be shown a variety of uses different from what can be seen at Disston's.

Also, Mr. J. Edward Addicks, of the Boston Gas Co., will be pleased to have its gas-works shown to any members of the Institute who may visit Boston, and they will there see the fuel water-gas made and measured with a meter, and the producer-gas used to fire boilers for making steam and to heat retorts for volatilizing the oil to carburet the water-gas for illuminating purposes. The Boston Gas Co. will distribute the fuel-gas through 6 miles of main by November 1st.

The quantity and quality of the water-gas made there is over 40,000 cubic feet, of 340 heat-units per foot, from 2000 lbs. of Cumberland coal.

The Akron Fuel Gas Co., Akron, Ohio, has ten miles of mains, and is distributing the water-gas for household and other uses.

THE IRON-ORES OF VIRGINIA AND THEIR DEVELOPMENT.

BY EDMUND C. PECHIN, ROANOKE, VA.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

THE writer approaches this subject with a great deal of diffidence —first, because it is utterly impossible to treat it satisfactorily within the limits of a paper, and, secondly, because the larger development is of so comparatively a recent date as to make authoritative data on some interesting points unavailable.

For over one hundred years small charcoal-furnaces have been in existence, at widely scattered points, making a few tons of iron daily, and then gradually increasing to a daily output of, say 10 to 12 tons, as the market widened. With the exhaustion of local supplies of wood, inaccessibility to market, and sharp competition, bringing lower prices, many were compelled from time to time to make temporary stops, and finally to cease altogether. During the first half of the century there were at different times in operation between 75 and 90 charcoal furnaces. The last Directory of the Iron and Steel Works of the United States (1890) gives 19 as the number of charcoal-stacks in Virginia. Of these several are cold, and, it is

perfectly safe to say, will never resume. All of them lie within a few miles of the Norfolk and Western and Shenandoah Valley railroads and run on the brown ores of, and adjacent to, the Potsdam sandstones, to be hereafter described. The iron made by these furnaces was almost wholly cold-blast and noted for its excellence for tough castings and chilling car-wheel irons. The change came when the completion of the Chesapeake and Ohio Railway made the fine coking coals of West Virginia available; and an extraordinary impetus was given when, in 1882, the Norfolk and Western Railroad, with wise foresight, built to the Flat Top coking coals and opened up that wonderful field to the outside world.

The census year of 1880 gave the total output of pig-iron in Virginia as 17,906 tons.

The census year of 1890, just completed, gives 302,447 tons. Of this, the charcoal-production was under 18,000 tons, the balance being made by 12 coke-furnaces. The rapid increase in late years is best seen by the figures of the annual production: 1886, 156,250 tons; 1887, 175,751 tons; 1888, 197,306 tons; 1889, 251,356 tons. There are now in course of active construction (and the majority nearing completion) ten large first-class modern coke-furnaces, with a yearly capacity of 250,000 gross tons, and several projected.

The important question now arises: What are the sources of ore-supply for all of these plants? To settle this it is necessary to consider briefly existing geological conditions.

Every known character of ore is to be found, in varying quantities, in the State of Virginia.

1. The magnetic, specular and gossan ores in the older rocks east of the Blue Ridge (Archæan belt).

2. The hydrated brown hematite of the Potsdam sandstones and slates and in the Silurian limestones overlying them (Nos. I. and II. of Professor Rogers).

3. The hydrated brown hematite of the Lower Silurian shales (No. III. of Professor Rogers).

4. The anhydrous red hematite, or fossil-ore, of Professor Rogers' Hudson Epoch No. V., ordinarily known as the Clinton or dyestone-ore (Upper Silurian).

5. The hydrated brown hematite of the Oriskany Epoch of Professor Rogers, No. VII. (Upper Silurian).

6. The hydrated brown ore accompanying the Subcarboniferous limestones of No. XI.

7. The carbonates of the Coal Measures, No. XI.

These will be considered in the inverse order.

The carbonates can be dismissed without further notice. As yet they have only been found as nodules in the slates, and are of no commercial value whatever.

Of the Subcarboniferous limestone brown ores but little is known. They are of unusual occurrence, and apparently only local, and on a limited scale. As far as they have been identified, they occur along the line of the great fault in Tazewell county, Va., on the Clinch Valley extension of the Norfolk and Western Railroad, by which fault No. XI. is brought into direct contact with No. II. The external indications for many miles would lead the casual observer to believe in the existence of very large bodies of ore; but positive developments weaken this impression. The quality of the ore is very fair, a considerable number of analyses showing:

	Per cent.
Iron,	from 38 to 49
Silica,	" 13 to 22
Phosphorus,	" 0.13 to 0.44
Manganese,	" 1 to 2.40

While these ores may serve as a local supply, they can never become, from present knowledge, of commercial value—in Virginia at least.

No. VII.—We now come to one of the great leading ore deposits of Virginia, which has during the last eight years furnished the largest amount of ore, and which was until within the last year the largest annual ore-producer, but which is now exceeded in output by the ores of Nos. I. and II. It is commonly and commercially known as the Oriskany brown ore, and supplies the furnaces at Lowmoor, Longdale, Victoria, and Princess.

At the present time these deposits are only reached by the Chesapeake and Ohio Railway. As there seems to be no reason for doubting the existence of the measures carrying these ores for several hundreds of miles through Virginia, in a northeast and southwest direction, it is quite within the range of probability that at no distant day the large area carrying these measures, now inaccessible, will be penetrated by new railroads or branches from existing roads, and the output of ore largely stimulated thereby. This is particularly probable from the fact that this ore-deposit, in connection with that of No. V., hereafter noted, lies nearer to the great coal-fields than any other. Incidentally it may be remarked that recent developments from Big Stone Gap, Wise county, Va., and Middlesborough,

Ky., are said to show positively the existence of the Oriskany ore at or near those localities. This being the case, it seems to prove the continuity of this ore-deposit over several hundreds of miles. The well known aphorism, "all is not gold that glitters," is thoroughly applicable not only to this great belt of ore, but to all other brown ores, to be hereafter considered. To quote from a recognized authority, the late Prof. J. L. Campbell, of Washington and Lee University, Lexington, Va: "Although the Oriskany is the most important of the ore-bearing formations in this particular region, it must not be inferred that every part of it, or even the greater part of it, is iron-ore. Large portions consist of a coarse, gray sandstone, that is frequently highly colored with iron rust, and that breaks up rapidly when exposed, and weathers into round boulders of a brown color."

In many places gigantic boulders, and even cliffs, 20 to 30 feet high, outcrop on the mountain side, and strike the inexperienced observer as iron-ore, pure and simple; but oftentimes these masses are fatally charged with siliceous matter, and are commercially valueless. Yet these are usually the landmarks which indicate large and valuable deposits of good ore immediately under them. Great variations both in quantity and quality occur in close proximity, even in thoroughly good mining ground, requiring the watchful attention of the practiced eye, and experienced miner. The usual pitch of the ore can be taken at about 45°, conforming to the general stratification, but this is constantly changing, at times flattening out, and at other times becoming nearly vertical.

Originally worked in and near the outcrop, as open cuts, the large mining operations are now chiefly underground. The present mines at Longdale, Alleghany county, Va., give information of much value, and serve as an illustration. Originally supplying a charcoal-furnace, requiring very limited supplies, open cuts were made at the outcrop, about 400 feet above the creek-level, on the hillside. These ran as open cuts about 4000 feet in length, and about 100 feet in depth. At the outset these same workings supplied the coke-furnaces, but at the depth of 100 feet it became impossible to keep up the walls, and underground mining was resorted to, and is now the only source of supply. In locating a tunnel, a shaft was started at the floor-level of the open cut and sunk 200 feet, with a somewhat disappointing result, as the ore when struck was at that point only about 12 feet wide, and not of average quality. Notwithstanding, the tunnel was driven 1100 feet, and struck the ore at the depth of

200 feet, but about 150 feet northeast of the shaft. The ore at this point was 25 feet wide and fully up to the average. From here adits were driven along the ore 1400 feet southwesterly and 1900 feet northeasterly. Still lower down the valley another tunnel was driven, striking the ore 120 feet below the bottom of the shaft, giving thorough drainage to the whole mine. From the surface to this lower level the ground is opened up by adits and winzes, making a splendid piece of mining property. The ore holds throughout, but turns and bends, and stands at every angle, conforming to the interstratified rocks from almost vertical to nearly flat. The mine is worked so that in coming back the pillars are drawn, the roof allowed to fall and the hill completely robbed. No better opportunity for studying the peculiar characteristics of this ore, with its variations, both as to occurrence and quality, can anywhere be found. The changes in its proportions of silica and manganese, frequently within a few feet, can be fairly detected by the eye. At the northeast end of the ore-line, where a ravine occurs, the ore is entirely cut off, or if continuous, it is at a very much greater depth.

These details have been gone into because the conditions found here are likely to be characteristic of this ore-lead throughout the whole length of the State, and in localities now untouched, but before long to be brought on the market.

The total output of this property up to the first of this year has been about 750,000 gross tons, and at the present rate of consumption, say 250 tons a day, if dead work were to stop, there is enough ground opened to give four or five years' supply. The bottom of the lowest level is in solid ore everywhere, and the depth to which the ore may run, is of course, unknown.

At the Lowmoor mine, a few miles away, a shaft has been sunk below water-level 200 feet, with the ore in first-class condition. Although all of the mining operations have to be conducted with explosives, four-fifths of the product must go through the washers to cleanse the ore of the accompanying clays. One double log-washer will wash 100 tons a day, so that a plant of three washers keeps a furnace fully supplied with beautifully cleaned ore. Now, as to the quality of the ore. The glowing accounts going the rounds of high-grade ores to be found in Virginia are misleading and inaccurate. As a rule, picked samples are taken for analysis and the results given as indicative of usual conditions. The following is the average of a number of analyses of picked lump:

	Per cent.
Water,	13.75
Silica,	3.90
Alumina,	2.50
Oxide of iron,	78.94
Oxide of manganese,	0 34
Lime,	0 14
Magnesia,	0.34
Phosphoric acid,	0.35
Sulphur,	trace.
Total,	100.26
Iron,	53.26
Phosphorus,	0 064

These are visitors' and promoters' samples.

The result of analyses of a large number of furnace-samples is:

	Per cent.
Metallic iron,	48.09
Oxygen,	20 48
Water,	10.37
Silica,	15.16
Alumina,	2.81
Sulphur,	none.
Phosphorus,	0.482
Manganese,	0.83
Lime,	0 28
Magnesia,	0.24

The manganese and silica are the main sources of trouble; the manganese, according to most competent furnace-authority, frequently running up to 2 to 3 per cent. There is, however, an authority from which there can be no appeal, and which the writer personally and carefully obtained. The average yield of the ores, worked by themselves, at one furnace in the district, is only 40 per cent. of iron, and the highest reached by any furnace does not exceed 43 per cent.

The massive lump-ore is principally found near the surface, and steadily decreases in depth, over three-quarters of the ore as mined being wash; and it is the wash-ore that carries the high silica. A safe business calculation requires that wherever the great Oriskany ore-beds are found and worked, fancy percentages of iron must be discarded. As these ores are found in immediate proximity to high-grade limestones, and with splendid coke, at moderate cost, available, they form one section, at least, of a broad and stable foundation upon which a magnificent industry may be erected.

The brands of pig-iron made by the furnaces working these ores exclusively are noted far and wide for their excellence, for both foundry and mill purposes.

CLINTON, No. V.—The red hematites, or fossil-ores No. V. of Professor Rogers, as far as Virginia is concerned at the present time, have simply a speculative interest, as they are nowhere mined or used in the State; but they are well known from Northern New York to Alabama. Used for many years at the Franklin and Onondaga furnaces, Oneida and Onondaga counties, N. Y.; one of the principal sources of supply for seventy-five years for a score of charcoal-furnaces; and later on for coke-furnaces chiefly in Bedford and Huntington counties, Pa.; they swing through Maryland, the entire length of Virginia from northeast to southwest; are being opened up at Middlesborough, Ky.; are the principal ores for South Pittsburgh, Dayton and Chattanooga furnaces in Tennessee and Rising Fawn in Georgia, and finally thicken up into the magnificent beds of the Red Mountain at Birmingham, Ala. The maximum thickness of the bed is at or near Birmingham, and it thins as it runs northward to 6, 5, and 3 feet, and down to 18 inches of good ore in northern Virginia, Maryland and Pennsylvania. Necessarily, there are local variations, some localities showing a greater thickness of bed than others. It is, therefore, not safe to predict what one locality is going to do by what has been found at another. There is a very large amount of this ore to be had at a very reasonable cost, with proper mining. There is one point to which too little attention has been given, and which becomes a very serious factor in determining quantity. Of course, the promoter does not want to recognize the fact, but fact it is, beyond any peradventure, and by overwhelming evidence, that a marked change takes place in this ore below water-level, the iron in it running down, and being replaced by carbonate of lime or silica. Over very large areas now worked, a soft fossil-ore above water-level, that shows from 52 per cent. to 55 per cent. of iron, and oftentimes not more than a trace of lime, gives below water-level, 28 per cent. to 34 per cent. iron, with carbonate of lime from 18 per cent. to 30 per cent. At least one large furnace-plant in the South, running principally on this lean, hard ore, uses no limestone whatever in its burden.

One fact must be borne in mind, as far as Virginia is concerned—that it is only recently that railroads have opened up the country carrying this ore, and made a positive development possible. Nota-

bly has this been brought about by the building of the Clinch Valley extension of the Norfolk and Western Railroad to Wise county, Va.; the South Atlantic and Ohio Railroad from Bristol to Big Stone Gap; and the Charleston, Cincinnati and Chicago Railroad from North Carolina to Wise county, Va.

The published prospectus of the Big Stone Gap Company states that about two miles from that place, on the line of the South Atlantic and Ohio Railroad, this ore has been opened up at the base of Wallen's Ridge, and shows a thickness of 39 inches of excellent soft ore. In *The Virginias* (May number, 1880, p. 78) may be found an article by M. P. N. Moore, then connected with the Geological Survey of Kentucky, on the "Iron-ores near Cumberland Gap, Va.," which is a succinct, clear, and forcible article on fossil-ores. He found there two workable seams of ore—one soft, averaging 20 inches, and one hard, of 27 inches—about 75 to 100 feet apart. Careful sampling by him gave:

	Soft ore. Per cent.	Hard ore. Per cent.
Ferric oxide,	73.935	47.965
Alumina,	5.776	2.130
Carbonate of lime,	4.510	1.230
Magnesia,	0.266	0.194
Phosphoric acid,	0.319	0.575
Sulphuric acid,	0.000	trace.
Silica and insoluble silicates,	11.730	43.690
Combined water,	3.850	4.000
Total,	100.386	99.784
Metallic iron,	51.754	33.575
Phosphorus,	0.140	0.251

On the Massanuton Mountain, over 200 miles to the northeast, and fronting on the west, on the Shenandoah Valley Railroad, the same seams exist. Frederick Prime, Jr., is authority for the statement that on this mountain the ore is known as the "hard" or "soft" "fossil-ore," according as the prominent impurities are silica or lime. This differs from the conditions existing in lower Tennessee and Alabama, but seems to verify what Mr. Moore found.

It is worse than folly to attempt to state with any precision what amount of workable ore can be found at any single locality, but we can safely sum up the general situation. This deposit of ore is a stratified bed or seam; it extends, with variations as to quantity, but of

a fairly uniform quality, the length of the State. It is in reasonable proximity to railroads built and building, and, with proper methods, it can be profitably worked. Whether or not, under water-level, it loses its commercial value, there is, in the aggregate, a large amount to be had above water-level. The soft ore will be of great value for admixture with the siliceous brown ores, especially in view of the fact that, as a rule, it carries less manganese and phosphorus; and it is likely to prove a factor of no mean importance in the future iron operations of the State.

Under No. V. comes the Medina sandstone—No. IV., of Rogers. In these measures are to be found large veins of what enthusiastic land-owners want to be iron-ore; but it is simply a ferriferous sandstone, carrying from 18 per cent. to 30 per cent. of iron, and from 30 per cent. to 55 per cent. of silica, and, of course, worthless for furnace-use.

No. III. of Rogers requires especial mention, as it carries valuable brown ores over considerable areas. These, in former days, served to supply a number of small charcoal-furnaces scattered through the mountain valleys, but are not worked at present. They can be made readily available on Purgatory Mountain, near Buchanan, Botetourt county, Va., and the projected Western Virginia Railroad, from Buchanan to the Kentucky line *via* Catawba Creek and Moccasin Gap, will open up many miles of these ores. Recent investigations seem to point to large deposits lying in a fairly regular condition and susceptible of easy and cheap mining. A series of samples taken by E. D. Frazier, a careful and conservative prospector, and analyzed by P. G. Salom, of Philadelphia, gave as follows:

	Per cent.
Metallic iron,	43.61
Alumina,	3.88
Silica,	18.75
Sulphur,	0.04
Phosphorus,	0.48

The manganese, unfortunately, was not determined, but it is safe to assume that it will follow the usual run of mountain brown ores of from 0.6 to 1¼ per cent.

We are now brought to the consideration of what we may fairly assume to be the great ore-bearing measures of Virginia—I. and II. of Rogers.

No. I.—The Valley of Virginia is a part of the great valley ex-

tending from Canada to Alabama, and noted for its rich, fertile limestone lands, its great iron-ore beds, its pure waters, its bracing, health-giving climate, and its varied and picturesque scenery. It is called in New York the Walkill Valley; in eastern Pennsylvania, the Kittatinny; in middle Pennsylvania, the Lebanon or Cumberland; in Virginia, the Shenandoah, James River, Roanoke, and New River; and further south, the East Tennessee Valley. It extends throughout the whole of Virginia, about 330 miles in length and from 12 to 30 miles in width. It is bounded on the southeast by the Blue Ridge Mountains, and on the northwest by broken ridges, locally known as Walker's, North, and Little North Mountains. The Shenandoah Valley and the Norfolk and Western Railroads run through its entire length, for many miles along the base of the Blue Ridge, which holds the Potsdam sandstones throughout, and on the top of which are the ferriferous shales, carrying the great brown ore-deposits of the country. These ores vary widely in quantity, quality, and condition, but in the aggregate seem to promise indefinite quantities. It is impossible, within the limits of this paper, to cover the innumerable matters of interest connected with this great field. Only a generalization can be attempted. The occurrence of the ore is thus described by Prof. W. M. Fontaine, of the University of Virginia: "We cannot, in strictness, call any of the deposits of the ferriferous shales veins or ledges, although ore may be, and usually is, found along certain definite bands. The deposits are lenticular masses or interrupted sheets of ore, that occur sometimes alone, or sometimes overlapping, in echelon, or more rarely radiating from a central point. Lines connecting the several masses in the direction of their greatest dimensions would fall within certain bands, and one mass may lie on the same line with another, so as to produce the appearance of one ledge or vein. The sheets and masses will, however, be found to be entirely enclosed in clay, and they are all formed by the concentrating action of concretionary forces that have collected this once diffused iron into masses that have more or less distinctly a concretionary structure, or that form beds of nodular ore, or crusts lying in an enclosing clay."

Beyond generalization, any theorizing on any special piece of ground is wasted time. The only sure guide is good, hard, common sense, familiarity with existing workings, either by practice or close observation, combined with the pick and shovel. Of course, it is assumed that there is a proper knowledge of the measures carrying the ore, their salient points, and what constitutes the walls of the de-

posit; but how the ore lies, or how it is to be most advantageously worked, must be decided as the work progresses. Horses of clay—sometimes of inches, at other times of many feet—are constantly encountered. The ore at any time may pinch to a trifling lead, and then open into the lenticular masses, of varying and sometimes of enormous size.

While the general direction of the ore is northeast and southwest, at places there are very great deflections, especially at the end of ridges and the foot of hills, where the ore may take a new direction, almost at right angles to the true lead.

Necessarily, under the old charcoal-practice, the quantity of ore required was very small, and no mining, in the proper sense of the word, was done. Where the ore outcropped and looked well, shallow pits would be opened on it, and as soon as one opening reached 10 or 20 feet in depth it would be, as a rule, abandoned, and another surface-point attacked. With the advent of the large coke-furnaces a different course was required. What may be reasonably looked for from these deposits can be best illustrated by an active mine, some five miles from Roanoke, owned and operated by the Crozer Iron Company. It lies well up on the mountain, some 400 feet above the water-drainage of the locality. The ground was opened in 1882 and 1883, and has been worked continuously ever since, and, until very recently, as an open cut over a large area. In some places the ore-ground runs to the surface, in others there is a clay covering of from 2 to 20 feet. This piece of ground becomes to the careful observer a complete and exhaustive object lesson. There is to-day, ahead of a long and wide open cut, a face of at least 60 feet of wash-ore ground, interspersed with boulders of solid ore of varying sizes, that will yield throughout 1 ton of washed ore to 3 tons of dirt handled. On the floor of this cut a shaft was sunk for 60 feet through solid wash-ground. A tunnel to strike this depth was driven for 1500 feet. The ore-ground was struck at 400 feet, and the tunnel was driven for 1100 feet through, leaving off in ore-ground that gives 75 per cent. of ore in the dirt handled. On the floor a well was sunk for 30 feet through the same character of ground. The depth to which the ore may run is unknown, but over a large area the ore has been proved to be 150 feet deep, with no signs whatever of exhaustion.

Eminent geologists have for many years claimed great depth to the Blue Ridge ores, and these developments seem to warrant the correctness of their diagnosis. It must not be inferred that this con-

dition of things exists everywhere along the belt; on the contrary, it is well known that there are many points where the ore is cut or squeezed out entirely, or is too limited in extent to be profitably worked, or the ore is too siliceous and lean for any present attention, and some of these conditions may exist with very fair surface indications; but from what is actually known, and with the knowledge that the same deposits of ore, on their northeastern continuation, sustained for many years the great iron industries of Lehigh and Berks counties, Pa., and largely helped to make that region populous and prosperous, it is safe to infer that for many years to come Virginia possesses, from this source alone, excellent and ample supplies to build up and sustain an iron industry to which all she has done, and is now doing, will be as child's play. Many hundreds of analyses of these Potsdam ores have been made, covering every section of the ground between Maryland and Tennessee, but of which only a summary of the extremes can be given:

Metallic iron from 58 per cent. in wash to 57 per cent. in picked lump.

Silica from 22 per cent. in wash to 3 per cent. in picked lump.

Manganese, from 0.10 per cent. up to proportions constituting manganese- rather than iron-ores.

Phosphorus, from 0.10 per cent. to 3.5 per cent.

Alumina, from 0.5 per cent. to 3 per cent.

Lime, under 0.75 per cent.

Magnesia, under 0.50 per cent.

Sulphur, as an almost invariable rule, in the hundredths of 1 per cent., or inappreciable.

Statements kindly furnished from two furnaces in the district which have largely used these ores for a long period, show the average yield of the prepared ores to have been from 44 to 46 per cent. of iron. In the main, the various deposits lie in such close proximity to the railroads as to be readily reached by short branches of 1 to 6 miles in length.

No. II.—Along the Cripple Creek Extension of the Norfolk and Western Railroad, running through Pulaski and Wythe counties, and extending southwesterly well toward the North Carolina line, is another large ore-field deserving particular notice. It is generally known as the Cripple Creek region, and contains a brown ore that has been famous in the past for its particular excellence in charcoal-irons. It is unquestionably the highest grade of brown ore known in Virginia. It lies on the Lower Silurian limestones, No. II., in

varying quantities, according to the folds of the limestone, but always of excellent quality. It is practically all worked in open cuts. The ore generally lies in a loamy, friable clay, but at times horses of a heavy, unctuous clay (locally called "buckfat") intrude. In some cases the deposits of ore-ground are shallow and in others deep. The richness of the ground varies greatly,—at times 4 yards of dirt yielding 1 yard of ore; at other times, 12 to 1. The average of the district is stated as about 8 to 1. It is dug and handled with extraordinary ease and cheapness where the ground is properly opened up and good washing-facilities are afforded. It shows, by innumerable analyses and by continued furnace-workings, a yield of over 48 per cent. of metallic iron, at times running up to 50 and 52 per cent., and for a brown hematite extraordinarily low phosphorus and manganese. The following analysis by A. S. McCreath is fairly characteristic of the ore:

	Per cent.
Sesquioxide of iron,	76.214
Sesquioxide of manganese,	0.055
Oxides of nickel and cobalt,	0.040
Alumina,	2.360
Lime,	0.820
Magnesia,	0.480
Sulphuric acid,	0.157
Phosphoric acid,	0.171
Water,	12.072
Siliceous matter,	7.480
Total,	99.849
Metallic iron,	53.350
Manganese,	0.036
Sulphur,	0.063
Phosphorus,	0.075

It can readily be seen that this ore plays a very important part as an admixture with the Potsdam ores, with their higher phosphorus and manganese. How long these deposits can stand a persistent, heavy drain upon them it is impossible to say definitely, but that they are available for many years to come is unquestioned.

In leaving the brown ores of Nos. I. and II., it seems not amiss to call attention to the important part they may play in the near future in the manufacture of basic, and especially of open-hearth basic, steel. Low silicon is the rule in all the irons produced in the district, and a great difficulty in the past has been to get high silicon

in foundry-irons. The writer has personally known of a lot of forge-irons, taken at random out of stock in the furnace-yards, and a score of analyses made by thoroughly competent chemists, where the silicon ran from a minimum of 0.30 per cent. to a maximum of 0.70 per cent., and an average of less than 0.50 per cent. By watching the ores as delivered, and properly charging the furnace, any desirable percentage of phosphorus and manganese whatever can be absolutely secured. Virginia can be considered as one of the few natural homes of basic steel.

Incidentally, ore-washing has been spoken of above. With the exception of the lump, which will not exceed if it attains one-quarter of the output of the mines, the brown ores of the whole district, whether No. I., II., III. or VII., have to be washed to free them from the enclosing clays. One well-constructed double log-washer will give an output of from 80 to 100 tons of ore per working-day, according to the nature of the ground, and a picking-belt attached thereto enables the washed ore to be freed from lump and pebbles of free silica. As a rule, jigs are not used, as generally they are not needed. Average ground requires the use of from 250 to 300 gallons of water a minute per washer. In many locations mountain streams and springs afford a supply, but there are many points where large mining operations can only be carried on by the establishment of pumping-stations, and bringing the water a greater or lesser distance. There is one item of expense inseparably connected with every mining operation, viz., the connecting of mines with the main lines of railroad. In the portion of Virginia under consideration, the railroad officials have exhibited a broad, liberal policy in this respect that has largely lightened the labors of the producers, to the advantage of both.

Before considering the last and oldest of the ore-bearing rocks of Virginia (Archæan), a single word may be said in relation to a so-called "ore," lying to the eastward and under the Potsdam sandstones. It has been heralded far and wide as a Bessemer specular ore in unlimited quantities, and as of inestimable value. It is neither. It is simply a ferruginous sandstone, which can be traced along the Blue Ridge from one end to the other. It lies in stratified seams of from 2 to 6 feet in width. Ordinarily, it carries under 30 per cent. of iron, and anywhere from 30 per cent. to 60 per cent. of silica. At one or two points there is a local improvement, which brings the iron up to 40 per cent. and the silica proportionately down. In such cases it is used in very small quantities to mix with the rank,

cold-short brown ores to dilute the phosphorus and manganese, from which it is almost free.

In the older rocks are found two classes of ore, the magnetic and specular, and the gossan. While the quality of the former is high, it has never been worked on a large scale, owing to the fact that no large veins have as yet been developed. Major E. S. Hutter has been working some mines on the Franklin branch of the Virginia Midland Railway, in Pennsylvania county, for several years. He is authority for the statement that they can be traced for many miles, dip nearly vertically, and average from 2 to 5 feet in thickness. Carrying from 56 per cent. to 62 per cent. of iron, and oftentimes within Bessemer limits, these ores are valuable for admixture with the brown ores, and naturally improve the character of the iron.

In the same rocks, in Amherst county, below Lynchburg, Va., considerable money and labor were expended in developing the magnetics and speculars, but without any commercial results.

In another portion of the field, coming under the writer's direct attention, a vein of fair magnetic ore, of about 50 per cent. iron, and 7 to 8 feet wide at the surface, was entirely pinched out at a depth of 75 feet. Moreover, the ore carried from 3 to 5.5 per cent. of sulphur. It is very possible that future research may show better conditions, but at the present time, with its very moderate development, it can scarcely be considered an important factor in Virginia iron-making.

There has been purposely left to the last a source of supply which is unique, and destined, in the writer's judgment, to have a widespread influence upon the iron business of southwestern Virginia,—the gossan ore-fields of Carroll county. For many years in the "long ago" this ore was worked in a small charcoal-furnace in Floyd county, which adjoins Carroll, the iron requiring many miles of transportation over wretched country roads to reach a market. The output of the furnace was put mostly into castings for local use, and acquired wide notoriety by reason of the strength, toughness and durability of the castings. Car-wheels made from this iron proved of extraordinary lasting power and tenacity, the latter undoubtedly owing to the copper contained in the iron. Practically, however, the iron-ore was of no value, the properties being worked for the copper underlying the gossan. In consequence, there is a splendid and unusual opening up of the ground, by many hundred feet of tunnels and shafts, at different points. The lode dips at an

angle varying from 40° to 70°, and also varies very greatly in width. There are at places surface widths of 300 to 500 feet. A conservative estimate would make an average width of say 50 feet throughout the whole lead, and an average depth of say 30 to 40 feet. In many places the ore comes directly to the surface, and generally has only from 2 to 5 feet of stripping. The lead is cut at frequent intervals by ravines, making the working of the ores almost child's play. There is no mining whatever. It will be an open cut or quarry, in which the ore is broken down and delivered to the cars. It is probable that a certain percentage of fine ores, lying near the surface and next to the walls, and somewhat mixed with the mica slates, may require washing, but the bulk of the ore quarried can be loaded directly into the cars. The following is the average of a large number of analyses sampled by experienced furnace-men and determined by competent chemists. It is proper to say that the samples were taken over a large area, from near the outcrop in the tunnels and shafts, and fairly represent the character of the ores:

	Per cent.
Metallic iron,	50.275
Silica,	8.335
Phosphorus,	0.075
Alumina,	from 2.5 to 4.00
Sulphur,	0.486
Manganese,	0.224
Lime,	from 0.20 to 0.400
Magnesia,	0.789
Copper,	0.442
Lead,	0.407

It is safe to assume that, with proper care, the ore can be had on a commercial scale to average 45 per cent. of iron.

One important statement remains to be made. It must be borne in mind that under the gossan occur the oxides of copper, red and black, with the carbonate in seams in the adjacent gossan. Below this follows the lode holding the gray and black sulphurets; beneath these the mundic rock to an unknown depth.

A diamond drill-hole has, within the last few months, been drilled to the depth of 110 feet through, and leaving off in the mundic rock, which is a sulphuret of iron and copper. Two tests were made by roasting this mundic rock, simply for iron and sulphur. A pile of the mundic from near the surface was roasted with wood, and a piece of the drill core, at a depth of 110 feet, was heated in the laboratory, with the following results:

	Surface mundic before roasting. Per cent.	After roasting. Per cent.
Iron,	54.26	63.96
Sulphur,	33.97	0.623

	Drill-hole mundic before roasting. Per cent.	After roasting. Per cent.
Iron,	53.13	59.91
Sulphur,	33.22	0.527

Assuming that modern experience and appliances can successfully handle this material on a commercial basis, so as to expel the sulphur, leaving the iron, the quantity to be obtained for this purpose is simply incalculable. Another "Blue Billy" factory can be started in this country. It is hard to conceive the effect upon the iron industries of Virginia in consequence of the opening up of these ores. As the writer has had occasion to remark elsewhere, it is possible, by varying the mixtures in the furnaces, to produce every grade of iron. As far as known, this is the first red-short ore that has, to this date, been made available in the South. It can be used by itself, or mixed with the brown ores of the district, in close proximity, to produce a rank or mild red-short, neutral or slightly cold-short iron, while certain of the brown ores will give a rank cold-short iron. For castings, especially small ones for shelf-hardware, etc., a low-priced iron can be made as fluid as water, and of very great strength. Its great point of value is, that by admixture it will permit the use of hundreds of thousands of tons of the mountain brown ores, capable of being cheaply mined and handled, that now, owing to their large percentages of phosphorus and manganese, are useless for present purposes. It is proper to state, in this connection, that very large deposits of this gossan are reported in Tennessee, on the southwestern extension of the lode, but at present unavailable by reason of distance from any railroad. The gossans of Carroll are now reached by rail from the Norfolk and Western Railroad at two points,—at the northeastern end by a branch road, 13 miles long, from the Cripple Creek Extension at Big Reed Island Creek, 25 miles from Pulaski, 37 miles from Max Meadows, 117 miles from Bristol, 52 miles from Radford, 89 miles from Salem, 96 miles from Roanoke, and 149 miles from Lynchburg. The lower or southwestern end of the lode is now reached by the Norfolk and Western Extension of the Cripple Creek Railroad, to meet the Cape Fear and Yadkin Valley Railroad, at the North Carolina line. This brings this part of the lead within 17 miles of Ivanhoe, 49 miles of Pulaski, 64

miles of Radford, 108 miles of Roanoke. It follows that this ore is commercially available at every furnace either now blowing, or building, or projected on the line of the Norfolk and Western and Shenandoah Valley railroads.

High-grade limestones and dolomites are to be had everywhere through the great Valley of Virginia, and it is not necessary to encumber this paper with detailed descriptions or analyses.

In considering the possibilities of Virginia as an iron-producing district, it would be manifestly improper not to call attention to the fuels available, as no matter how large the quantity, or how cheaply obtained the ores might be, they could not be utilized in the absence of a proper fuel.

The merits of the Flat Top coals and coke are now too widely known to warrant elaboration. The excellence of the coking coals of the New River district, on the Chesapeake and Ohio Railroad, has been too well tested in furnace-practice for years past to admit of a doubt. The coking coals of the New River (Chesapeake and Ohio Railroad), upon which the furnaces at present working the Oriskany ores (No. VII., as above) depend, are elaborately treated by J. C. White, a recognized authority, in his report to the United States Geological Survey in 1884. He gives three seams of coal: the Nuttall, with its geological horizon on New River, about 400 feet below the top of XII.; the Quinnimont and Fire Creek, from 300 to 400 feet below the Nuttall. These seams run from 3 feet to 4 feet in thickness. The coal literature of the country has been flooded for the last ten to fifteen years with analyses of all these coals and coke. Of course, these vary according to the samples taken. It is believed that the following are fairly representative:

Coals.

	Quinnimont.	Fire Creek.	Nuttall.
Fixed carbon,	75.89	75.02	70.67
Volatile matter,	18.19	22.34	25.35
Ash,	4.68	1.47	2.10
Sulphur,	0.30	0.56	0.57
Water,	0.94	0.61	1.35

(*Transactions* American Institute of Mining Engineers, vol. viii., pp. 261 *et seq.*)

Coke.

	Quinnimont.	Fire Creek.	Nuttall.
Carbon,	93.11	92.18	92.22
Ash,	5.94	6.68	7.53
Sulphur,	0.82	0.61	0.92

(T. Egleston, Dr. Ricketts, C. E. Dwight.)

The percentage of ash in the Fire Creek and Nuttall cokes is entirely too high for the ash shown in the coals, but the chances are that the coal samples were picked, while the coke shows the run of the mines.

The Flat Top seam, with its magnificent thickness of 6 to 11 feet, must necessarily be the source of supply to all of the furnaces located on or near the line of the Norfolk and Western road, and it is in this section that the large present and near future development is taking and must take place.

A. S. McCreath and E. V. d'Invilliers, for many years connected with the Second Geological Survey of Pennsylvania, in a recent report on the New River and Cripple Creek mineral region, give valuable addenda on the limestones and coals and coke, and from these the following are taken. (See also their paper, *Trans.* xv., 752.)

Flat Top coals, averaging fifteen samples:

	Per cent.
Water,	1.011
Volatile,	18.812
Fixed carbon,	74.256
Sulphur,	0.730
Ash,	5.191

From three large works at Pocahontas, in active operation, they selected samples of coke, which gave as follows:

	No. 1.	No. 2.	No. 3.
Water,	0.182	0.196	0.664
Volatile,	0.719	0.494	1.059
Fixed carbon,	92.248	92.585	92.816
Sulphur,	0.565	0.677	0.548
Ash,	6.286	6.048	4.913
Totals,	100.000	100.000	100.000

As one of the possibilities of successfully making iron in Virginia, it does not seem unfair to make the following comparison, as

	Water.	Volatile.	Fixed Carbon.	Sulphur.	Ash.
1. Pocahontas coke.— Average of three samples	0.347	0.757	92.550	0.597	5.749
2. Birmingham, Ala.—Average of four samples	0.157	0.803	87.299	1.195	10.545
3. Chattanooga, Tenn.— Average of four samples	0.447	1.101	80.513	1.595	16.344
4. Connellsville, Pa.—Average of three samples	0.060	0.427	88.962	0.810	9.741

quoted by Messrs. McCreath and d'Invilliers, especially as the analyses of the cokes of other districts have been for a long time public property.

The growth of the product of Flat Top coals from nothing in the census year of 1880 to nearly 2,000,000 tons in 1890, and of coke from nothing to many hundreds of thousands of tons, with an increasing demand which present facilities cannot possibly meet, tells the story of the useful quality of these materials.

The following table of distances to furnaces now in operation, building or projected, may prove interesting:

From Pocahontas to Graham, 10 miles; to Radford, 75 miles; to Pulaski, 90 miles; to Max Meadows, 103 miles; to Bristol, 182 miles; to Ivanhoe, 122 miles; to Salem, 112 miles; to Roanoke, 119 miles; to Lynchburg, 172 miles; to Glasgow, 163 miles; to Buena Vista, 173 miles; to Basic City, 215 miles; and to Shenandoah, 251 miles.

The cokes from the New River district on the Chesapeake and Ohio Railway, 112 miles to Lowmoor, 118 miles to Longdale, 121 miles to Wilton, 133 miles to Goshen, 160 miles to Glasgow, 175 miles to Buena Vista, 198 miles to Lynchburg, 178 miles to Basic City, and 214 miles to Shenandoah.

It is not within the province of this paper to give any figures of cost. The reader must form his own conclusions, based upon his own experience, as to the prices at which materials of the quality as given and lying in proximity as described, can be handled by prudent and practiced business men. The visitor, interested in obtaining information on this point, will find abundant sources to furnish what he desires.

The writer claims the right to say, with a large personal knowledge of different mineral sections of the country, and with no desire to detract in any way from the merits of many other districts in this favored land lavishly blessed by nature, that, taking into consideration the variety and position of the various mineral resources of Virginia, it is the most remarkable and interesting mineral field that has ever come under his examination.

In closing this paper, he desires to express his acknowledgments for the information obtained from the valuable papers and reports of Drs. W. H. Ruffner and Egleston; Profs. Campbell and Fontaine; Messrs. Prime, Moore, McCreath and d'Invilliers; and especially to the invaluable publication by Major Jed Hotchkiss, of Staunton, Va., of *The Virginias*, from 1880 to 1885.

MASSICKS & CROOKE'S AMERICAN PATENT FIRE-BRICK HOT-BLAST STOVES.

BY WALTER CROOKE, MILLOM, CUMBERLAND.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

REGENERATIVE hot-blast stoves are now in general use in all parts of the world, and are so well understood and appreciated, that I need not take up your time with an account of their history and introduction into the United States. My object in writing this paper is to give you a short description of a stove that is now being very favorably received and adopted here (see drawing). The stove was first introduced into this country in 1887 by M'Clure & Shuler (now M'Clure & Amsler), who have the management of the patent right in the United States.

The English design had to be modified to suit the exigencies of American blast-furnace practice, which, in many respects, widely differs from our English experience.

In the first place, the fire-bricks used in America expand to an extent unknown in England, and it became necessary to make special provision for this by increasing the number of cross-walls, and by leaving a space at the top of the stove through which the lining could expand. The expansion laterally was, however, more difficult to provide for, as the combustion-chamber in the center, being the hottest part of the stove, must necessarily expand more than the outer walls, and therefore tend to squeeze up the flue-openings.

On first introducing these stoves to American ironmasters, a feeling seemed to prevail that a "four-pass" stove did not utilize the whole of its surface. They also had an idea that the "two-pass" stove let the burning gas go before all the duty had been extracted from it; so that it was found necessary to design a stove of a special character to suit their wishes. Hence the "three-pass" chimney-top stove now before your notice, the advantage claimed for which is that each stove is independent, requiring no underground chimney-flue or stack, and can be erected at a very light cost. The working of these stoves has proved to be very successful wherever they have been introduced.

The first stoves built in this country were put up for Messrs. Shoenberger, Speer & Co., Pittsburgh, Pa. They were three in number, 16 feet 6 inches in diameter and 57 feet high to the eaves; the furnace was 14 feet bosh and 62 feet high, at that time making 450 to 500 tons per week with pipe-stoves. The difference found on blowing-in the improved plant was at once apparent; the output rose to 800 tons per week, and the fuel-consumption diminished to 1900 and 2000 pounds per ton of iron, instead of from 2700 to 3000 pounds. These stoves have been continuously in blast for a period of nearly three years, and, beyond a few hours' stoppage for cleaning, have not cost a cent for repairs.

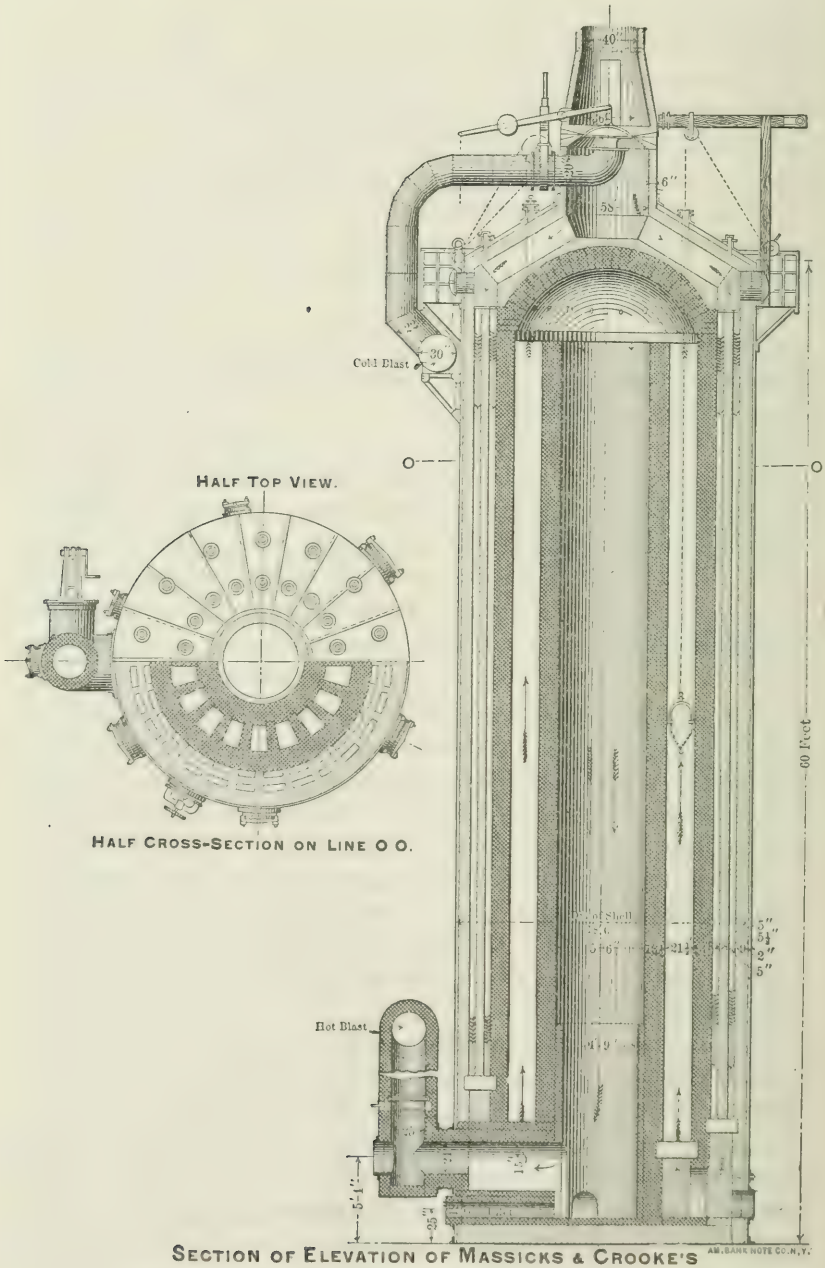
Since that time the Massicks & Crooke's stoves, with three passes, have been used in every iron-making section of the country, and have well proved their efficiency.

The method of cleaning employed in the English stove has substantially been retained, but various improvements have been introduced which add greatly to effectiveness and speed of cleaning. The second pass (that nearest the combustion-chamber) is cleaned with a scraper, which has been specially designed to accommodate itself to any little irregularities in the flue-walls. For the third or outside pass an automatic cleaning-door was devised, which is fitted on to each bottom cleaning-door in turn. The stove is then filled with air at blast-pressure, and the catch on the door is knocked off, when the imprisoned air rushes through the flues towards the door, carrying with it any loose dust that there may be in them. For dislodging such dust as may adhere to the walls a wire brush is used, operating in the same manner as the scraper in the second pass.

Among minor details, the slide gas-valve has been abandoned, and a positive cut-off valve, universally used in this country, has been substituted. The Massicks & Crooke's hot-blast valve has been retained, and has given great satisfaction in respect of its simplicity and durability.

The duty required of hot-blast stoves in America is much heavier than in England, an ordinary-sized furnace, say 18 feet bosh and 75 feet high, being blown at a rate of 20,000 cubic feet of air per minute at a temperature of 1500° Fahr. This of course makes it necessary to heat the stove up to a very high temperature, and so much greater is the duty required than formerly, that devices which had previously given the greatest satisfaction have failed to meet the present requirements.

The stoves at the Tod furnace owned by the Brier Hill Iron



HALF TOP VIEW.

HALF CROSS-SECTION ON LINE O O.

SECTION OF ELEVATION OF MASSICKS & CROOKE'S PATENT 3-PASS FIREBRICK STOVE, 18' 0" DIAM., 60' 0" HIGH.

and Coal Company, Youngstown, Ohio, 16 feet 6 inches diameter, and 57 feet to the eaves, have been doing excellent work. The furnace is 15 feet 6 inches bosh, by 70 feet high, and has been making 185 gross tons per day on a very low consumption of coke, viz., 1700 to 1800 pounds per gross ton of iron; the amount of air passing through the stoves being 17,000 cubic feet per minute, at a temperature of 1300 to 1500° Fahr., with a loss of from 100 to 120 degrees in one hour's blow.

You can very readily see from the above that it was necessary to increase the volume of brickwork, and also to adopt a more thorough system of binding and bracing the circular walls than hitherto, so as to avoid all possibility of the walls moving out of their position, and in order to allow them to expand and contract at will without shearing off the bricks.

In order to give an idea of the approximate cost of a set of three stoves of various sizes that M'Clure & Amsler have been building, it may be interesting here to notice the quantity of material required and the average cost of the same, viz.:

Three stoves, 16 feet 6 inches by 60 feet.

	Pounds.
Wrought-iron,	175,000
Castings,	41,000
Bricks, first quality,	186,000
Bricks, third quality,	203,000

Average cost, $17,521, or £3650, 4*s.* 2*d.**

Three stoves, 18 feet by 60 feet.

	Pounds.
Wrought-iron,	188,500
Castings,	42,000
Bricks, first quality,	235,000
Bricks, third quality,	224,000

Average cost, $19,934, or £4152, 18*s.* 4*d.*

Three stoves, 19 feet 6 inches by 65 feet.

	Pounds.
Wrought-iron,	220,000
Castings,	46,000
Bricks, first quality,	256,000
Bricks, third quality,	342,000

Average cost, $23,839, or £4966, 9*s.* 2*d.*

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* These figures are here reprinted without change from the paper as presented to the Iron and Steel Institute. The author has evidently assumed, for convenient calculation, the value of $4.80 for the pound sterling.—R. W. R.

Three stoves, 21 feet by 70 feet.

	Pounds.
Wrought-iron,	282,000
Castings,	53,000
Bricks, first quality,	373,000
Bricks, third quality,	400,000

Average cost, $30,872, or £6431, 13*s.* 4*d.*

The following is a list of the stoves now in operation and in course of construction:

Stoves in Operation.

	Stoves.	Ft.	In.	Ft.
Shoenberger, Speer & Co., Pittsburgh, Pa.,	6	16	6 by	57
Williamson Iron Co., Birmingham, Ala.,	3	16	6 "	57
Illinois Steel Co., Chicago, Ill. (Bayview),	6	18	0 "	60
" " " (N. Chicago),	3	18	0 "	60
" " " (Joliet),	4	19	6 "	70
Thomas Furnace Co., Niles, Ohio,	3	16	6 "	57
Brier Hill Iron and Coal Co., Youngstown, Ohio (Tod Furnace),	3	16	6 "	57
Brier Hill Iron and Coal Co., Youngstown, Ohio (Grace Furnace),	3	19	6 "	65
Carrie Furnace Co., Pittsburgh, Pa.,	4	19	6 "	65
Chicago Furnace Co., Chicago, Ill.,	1	16	6 "	57
Bellaire Nail Works, Bellaire, Ohio,	3	18	0 "	65
Riverside Iron Works, Wheeling, W. Va.,	3	19	6 "	60
Missouri Furnace Co., St. Louis, Mo.,	1	18	0 "	60
Total,	43			

Stoves in Course of Construction.

	Stoves.	Ft.	In.	Ft.
Illinois Steel Co., Chicago, Ill. (S. Chicago),	16	22	0 by	70
Paducah Iron Co, Paducah, Ky.,	2	16	6 "	57
Roanoke Iron Co., Roanoke, Va.,	4	18	0 "	60
Vanderbilt Iron and Steel Co., Birmingham, Ala.,	3	16	6 "	60
Bellaire Nail Works, Bellaire, Ohio,	1	18	0 "	65
Riverside Iron Works, Wheeling, W. Va.,	3	18	0 "	65
Total,	29			

No doubt many of the members of the Iron and Steel Institute will see these stoves in the course of their visit to the United States.

ALUMINUM-STEEL.

BY R. A. HADFIELD, SHEFFIELD, ENGLAND.

(From the proceedings of the New York Meeting of the Iron and Steel Institute, October, 1890.)

It seems a specially fitting opportunity to present a paper on the alloys of iron and aluminum at the New York meeting of this Institute, owing to the fact that America has, more than any other country, originated and developed the manufacture of cheap aluminum. The Cowles, Castner, and Hall systems have all originated in the ever-busy brain of our cousin across the water.

The problem of cheaply producing this valuable metal has been a difficult one, but to the writer it seems that the more arduous the task the greater the delight with which our American friend attempts to overcome the difficulties surrounding it, and the more *vim* he throws into the work. Another and equally striking proof of this is to be found in the recent appearance of the already well-known remarkable work, *The Metallurgy of Steel*, by Mr. H. M. Howe, of the Institute of Technology, Boston, Mass. It is to be hoped that this Institute will not be slow to recognize by some special mark its appreciation of this signally valuable and original work.

The writer has been led to describe the following experiments, believing that so far no full and exact report has yet been presented to this Institute as to the effect of aluminum alloyed or combined with the metal iron.

It has been a special object to make the material as far as possible a true alloy or combination of aluminum and iron, with as small quantities as possible of other disturbing ingredients, such as silicon, manganese, etc. In the material described in these experiments, the total amounts of other elements present (sulphur and phosphorus being disregarded as present in such small quantities) do not exceed about 0.5 per cent. Such material may therefore be probably taken as fairly illustrative of the effect of the metal aluminum upon iron. Since completing the experiments the writer

understands that Mr. Riley has also made some tests; in his case the other elements present amounting to about one per cent. The samples now under examination may therefore be considered as more nearly approaching what may be termed, for the sake of convenience, aluminum-steel, especially as the manganese, so disturbing an element, is unusually low.

It is not proposed to deal, except briefly, with the actual production of the interesting metal aluminum. A metal the price of which has been reduced in little more than 30 years from 6*s.* per ounce to 6*s.* per pound must have an interesting history. This metal without doubt must play a considerable part in the metallurgy of the future; and whilst there has been much exaggeration* about it—and those who exaggerate its qualities and capabilities are its worst friends—still a metal whose specific gravity is only 2.6; a cubic foot of which weighs only 160 pounds, against iron 440 pounds, or silver, to which it has points of resemblance, 650 pounds; tensile strength about 10 tons per square inch; is inoxidizable in a high degree; and has high electro-conductivity, must indeed have an assured and prominent future.

Whilst the metal itself has proved so difficult of extraction, it is in its oxidized state to be found very abundantly in nature. Native metallic aluminum has not been met with. Somewhat curious, as pointed out by Mr. Richards in his interesting book on the metal, is the fact that the oxide, alumina, unlike silica, which it resembles in many respects, has, notwithstanding its abundance, not been found in animals or plants.

It is not believed that the metal was in any way known to the ancients. Although the term Alum, the *Alumen* of the Romans, occurs in Pliny's and other ancient writings, yet it has not been satisfactorily proved that our alum was the substance referred to. On the contrary, the learned Bechmann has asserted that the "Alum" of the Greeks and Romans was sulphate of iron, and that the invention of our alum was much later, in the twelfth century. It was certainly mentioned by Chaucer in 1356.

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* One scheme propounded and abundantly circularized was to have outcome in the production of 400 tons per month of aluminum and its alloys. The profit was to amount to the modest sum of £20,000 monthly! No wonder such a prospect was entitled "A New Era in Metallurgy."

A well-known technical journal published a statement that "Aluminum can be hardened until the diamond is its only rival!"

Speaking briefly of its early history, Margraff proved in 1754 that the earth alumina was a metallic oxide. Sir Humphrey Davy in 1807 unsuccessfully attempted to extract the metal by means of the electric current, and was followed by Oerstedt, the Dane, in 1827; but the great chemist, Wöhler, in 1827, and more completely in 1845, must be credited as being the true discoverer. Bunsen showed in 1854 how the metal could be obtained by electrolysis. St. Claire Deville, of France, as is well known, was the first to take up and perfect the early manufacture of the metal on a commercial scale.

Our own Dr. Percy in 1855 was, through the outcome of laboratory experiments, the first to develop the improved process of making the metal from the fluoride; and, still later, another member of this Institute, one of our past Presidents, Sir Lowthian Bell, devoted much time and attention to the problem.

Professor Williamson, in his address to the Chemical Section of the British Association, 1863, referred to this, stating "One of the most interesting novelties in metallurgy is the manufacture of aluminum, now carried out for the sake chiefly of its alloy with copper by the distinguished gentleman who holds the office of Mayor of Newcastle (now Sir Lowthian Bell). The mechanical properties of this so-called aluminum-bronze give it great value, and it seems likely to find much favor for its appearance."

At the same meeting Sir Lowthian Bell gave a paper "On the Manufacture of Aluminum," in which he mentioned that it had then (1863) been carried on some $3\frac{1}{2}$ years at his Washington works. His process consisted in the employment of bauxite mixed with the ordinary soda-ash of commerce and heated in a furnace, the remainder of the process being similar to that of M. St. Claire Deville.

Sir Lowthian Bell then stated that the increased activity in the manufacture of the metal was due to the value of the alloy of this metal with copper, and that this "alloy, or aluminum-bronze, as it is termed, was the discovery of Dr. Percy;" and he considered it to be a true chemical compound. He specially pointed out the fact that the union of the two metals, copper and aluminum, was attended with such increase of temperature, that unless the crucible was of good refractory material it would become fused. He also pointed out—and it must be remembered this is nearly 30 years ago—that one of the best bronzes (in its forged state), that is, one of the strongest, most malleable and ductile, was that containing 10

parts of aluminum and 90 of copper, a fact now offered as a recent discovery by some writers on the subject.

Dame Nature keeps most of her secrets very tenaciously locked up, and no doubt in many ways this is a good thing; but none of her problems have proved more difficult than that of finding out how to produce the metal, aluminum, *cheaply*. Too great praise cannot be given for the amount of patient and laborious investigation by the many scientists and practical men who have worked at its solution, and who recognized that if methods could be perfected to this end, there would be a great future for the metal.

During the last 10 years Cowles, Netto, Castner, Grabau, Héroult, Minet, finally culminating in Hall's process, taken up by the Pittsburgh Reduction Company, have all worked successfully. That we are fairly on the road to the desired cheap production may be gathered from the fact that its commercial value has been reduced to 6*s*. per pound; and the writer believes that he is correct in stating that by the Hall and other processes, and notwithstanding these later lower prices, the manufacture of aluminum is now being profitably pursued.

Name.—A recent interesting paper to the American Institute of Mining Engineers by Mr. Oberlin Smith, entitled "Aluminum in Search of a Nickname," refers to the title which should be bestowed on the metal. Whilst hardly agreeing with him that we should come down to the short, but somewhat undignified term "Al," both that of "Alumin*ium*" (adopted in England), or "Alumin*um*" (as in America), are somewhat awkward. It seems to the writer that as "Aluminium" seems in several ways to resemble "Sili*con*," as will be seen from the experiments detailed in this paper, it might be well to name the metal "Alumin*on*." One now seldom hears the old term "Sili*cium*," besides which the affix "ium" belongs to metals with quite distinctive qualities, such as "Sodium," "Potassium," "Lithium," etc., and it would make our now cheaper but respectable friend fall more in line with the ordinary wording at present adopted. The above suggestion would also meet the objection raised by American friends, who rightly contend that if "Alumin*ium*" is adopted, the oxide should be termed "Alumin*ia*," which no one would defend. The title "Aluminon" would, however, meet both cases, as well as proving more euphonic.*

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* *Note by the Secretary of the A. I. M. E.*—For the sake of uniformity, the form aluminum has been adopted throughout in the reprint of this paper.

Processes of Manufacturing Aluminum.—As is well known, the properties of aluminum are essentially modified by even very small proportions of impurities, both as regards resistance and changeableness; but all the users of the following systems claim to supply it in purity from 97 per cent. upwards.

The principal processes now commercially used in the manufacture of the metal are—

The Castner process, carried out by the Aluminum Company at Oldbury, Birmingham, England. This is too well known from the excellent papers and descriptions by Sir Henry Roscoe to need further reference here.

The Netto process, employed by the Alliance Aluminum Company, Newcastle-on-Tyne, which has been described by Professor Williams, of Firth College, Sheffield.

The Cowles electric system, differing from the others in the fact that it preferably produces alloys of aluminum with other metals, and not the metal itself, is also well known.

The Grabau method, based upon the reduction by sodium of fluoride of aluminum, produced from the action of sulphate of aluminum upon fluor-spar and cryolite. As the latter mineral is reproduced in large quantity during the process, in an artificial form, it is of a much higher degree of purity, and therefore it is claimed that a metal of exceptional purity can be obtained. The sodium used in the operation is obtained by electrolyzing melted chloride of sodium in a crucible, one electrode being of carbon and the other an iron wire. As compared with the Deville process, this is said to utilize the sodium more perfectly, viz., from 83 per cent. to 90 per cent. of the reducing effect being realized, as compared with 76 per cent. by the former method.

The cheapness of the product by this method, according to M. Ichon, in *Annales des Mines*, depends on the electrolytic process of manufacturing the sodium, which M. Grabau claims to be able to produce at from 7*d.* to 9*d.* per lb., or about one-tenth the current rate. This inventor, in consequence, claims to produce aluminum at the cost of about 5*s.* 10*d.* per lb., on an output of as little as 50 lbs. of metal (99 per cent. purity) per twenty-four hours. The cost of plant to accomplish this is stated as £7300. The working expenses and the cost of materials are taken at £12 per day of twenty-four hours, to give the above-named output. As it is the most recent estimate of cost, it may be interesting to give this inventor's figures.

Investment.

	Francs.
Buildings,	57,000
Steam-engine, boiler, dynamo,	35,000
Electric apparatus for making sodium,	22,000
Sundry apparatus,	25,000
Working capital,	40,000
	179,000

Working Expenses.

	Francs.
Labor per day,	40.00
Chemists,	40.00
Fluor-spar,	12.00
Sulphuric acid,	2.50
Sulphate of alumina,	60.00
Sea-salt,	36.00
Various chemical products,	16.00
Coal,	50.00
Repairs and sinking fund,	30.00
	286.50

Or about 14.30 francs per kilogram of aluminum produced.

The Héroult * process, in use at Neuhausen, consists in decomposing melted alumina by a powerful electric current between carbon electrodes. The oxygen is disengaged at the anode, burning off in carbonic oxide. The metal is produced in a carbon-lined cast-iron vessel. If it is desired to produce pure aluminum, a quantity of liquid cryolite is poured into the vessel, and also alumina. The negative electrode is a carbon rod, lowered into the bath. Aluminum-bronze and aluminum-iron are manufactured by this process in a similar manner, the only difference being that the melted copper- or iron-ore are poured into the bottom of the bath, and no cryolite is used.

M. Ichon, in the *Annales des Mines*, for 1889, states that MM. Bernard Frères of Creil, who exhibited at Paris last year, claim to be able to produce the metal at as low as 4*s.* per lb. (cost price), but the writer has not seen this substantiated.

Finally the latest system, viz., Hall's, assisted by Messrs. Hunt

* This process has been fully described by Mr. Dewey in a paper to the American Institute of Mining Engineers, Washington, 1890. It is only fair to Messrs. Cowles to state that they consider this modification an infringement on the principles of their patents. To these gentlemen we certainly owe the rapid development, if not practically the introduction, of the electric-arc furnace.

and Langley, carried on by the Pittsburgh Reduction Co., and in England by the Metal Reduction Syndicate, Manchester, has been fully brought before the public in a recent paper,* read before the American Institute of Mining Engineers (Washington, 1890), on "The Properties of Aluminum with some Information relating to the Metal."

According to a recent American paper this company, having met with so much success in its preliminary work, proposes to erect large new factories, which, according to a recent Pittsburgh paper, are to possess 10,000 horse-power. The same source states that they already make over 400 lbs. of aluminum per day (in 1887 the world's production did not probably exceed 10,000 per annum), and that they import at present from Germany ore, or rather clay, from which the metal is extracted. The same paper states that aluminum made by this Company has been supplied in considerable quantities for ornamental work in the cabins of the latest ocean steamers, and that it has been adapted to the manufacture of table cutlery with success.

The English branch of this company established at Manchester state that they believe that as the demand grows and the prices of raw materials used come down, the selling price of the metal can be reduced to even under 6*s.* per lb.

Aluminum-Copper and Bronze.—Having mentioned the principal methods of producing the metal itself, reference may be made to the already large employment of aluminum alloyed with copper, known as aluminum-bronzes, for quite a countless number of purposes. It may be interesting here to state, as one of the first public notices, that in the *Morning Star* of May 21, 1862, considerable reference was made to the probability of aluminum-bronze coming into extensive use. The many good qualities of these alloys will no doubt make the use of aluminum in this direction alone of very wide importance.

The following table gives the range of tensile strength and density of these forged aluminum-bronzes:

The brittleness of alloys above 11 per cent. prevents their use.

Those containing 60 per cent. to 70 per cent. aluminum are very brittle and beautifully crystalline; with 50 per cent. the alloy is quite soft, but under 30 per cent. the hardness returns. The 20 per

* This is the most complete and comprehensive paper yet published on the properties of the metal.

cent. bronze has a whitish-yellow tint, somewhat resembling bismuth, but is very brittle, and can be pulverized in a mortar.

TABLE A.—*Tests of Aluminum-Bronzes.*

By Mr. JOHN H. J. DAGGER, in a paper read before the British Association, 1889.

	Tensile Strength in Tons per Square Inch.	Elongation. Per cent.	Density.
Aluminum-bronze, 11 per cent. Al.....	40 to 45	8	7.23
" " 10 " "	33 " 40	14	7.69
" " $7\frac{1}{2}$ " "	25 " 30	40	8.00
" " $5\text{-}5\frac{1}{2}$ " "	15 " 13	40	8.37
" " $2\frac{1}{2}$ " "	13 " 15	50	8.69
" " $1\frac{1}{4}$ " "	11 " 13	55	.....

One of the most valuable properties of the alloys given in the above table is the fact that they are forgeable, and can be worked at a red heat.

Table B, having been specially prepared by Professor Tetmayer, of the Polytechnic School, Zurich, for the *Aluminum Industrie-Actiengesellschaft*, at Neuhausen, is given for comparison with the foregoing.

TABLE B.—*Tests of Aluminum-Bronzes.*

	Aluminum. Per cent.	Tensile Strength in Tons per Square Inch.	Elongation. Per cent.
Aluminum-bronze.....	11.50	52	0.50
" "	11.00	44	1.00
" "	10.00	42	11.00
" "	9.50	40	19.00
" "	9.00	37	32.00
" "	8.50	33	52.50
" "	5.50	28	64.00
Aluminum-brass	4.00	45	6.50
" "	3.00	39	7.50
" "	2.50	34	20.00
" "	2.00	31	30.00
" "	1.50	29	39.00
" "	1.00	26	50.00

Professor Tetmayer has plotted his results, the curve obtained showing that with increasing aluminum-contents the tensile strength increases slowly at first, but then grows rapidly as the alloy is made richer in the lighter metal.

Alloys of Cast-Iron and Aluminum.—As with other cast-iron, aluminum cast-iron, if it may be so termed, naturally comes under a different classification to that of the malleable compounds of alum-

inum and iron, and it is not intended to deal with it here. We are again indebted to Americans for information in this direction, especially to Mr. Keep, who has done so much to raise and put the system of producing and mixing irons for iron-foundry purposes on a more exact and scientific basis. He has already published a series of valuable investigations in a paper presented to this Institute last year on "Aluminum in Carburetted Iron," also other papers to the American Institute of Mining Engineers. He found that aluminum has, in a considerable degree, a similar influence to that of silicon upon cast-iron, a fact strikingly confirmed in some special experiments of the writer relative to the action of aluminum upon combined carbon in spiegeleisen, and described later on. It is, however, only proposed to deal here with aluminum in its use in the manufacture of steel or steely compounds. There appears to be much misconception as to whether its employment is productive of good, and the writer confesses that he believes that whilst the price remains so high, and except in certain special cases, its application does not seem likely to become large. Aluminum appears to be of most service as an addition to baths of molten iron or steel *unduly* saturated with oxides, and this in properly regulated steel-manufacture should not often occur. Speaking generally, its *rôle* appears to be similar to that of silicon, though acting more powerfully. It must, however, be remembered that these experiments have been made with practically pure aluminum. If pure silicon were also obtainable, its effect would probably be found to be almost the same. So long, therefore, as ferro-silicon, varying in silicon from 8 to 20 per cent., can be obtained at from £3 10*s*. to £10 per ton, as compared with aluminum or ferro-aluminum of like percentages costing £112 to £250 per ton, it will be seen that the probable field of usefulness for the latter must be much circumscribed, cost being so much an element now-a-days. This fact, however, need in no way interfere with an examination as to its influence upon the metal, iron. The writer will as far as practicable endeavor to deal with the subject in a similar manner to that followed in his experiments with alloys of iron and manganese, iron and silicon, and other alloys.

Cast Aluminum-Steel.

Melting.—The material obtained for these experiments was produced by melting in crucible, in the ordinary manner, good wrought bar-iron, and adding the aluminum (about 98 per cent. Al), manufactured by the Pittsburgh Reduction Company's system, shortly

before teeming. The writer takes this opportunity of expressing his thanks to Captain Hunt, one of the directors of the Pittsburgh Company, for a liberal supply of the metal for experimental purposes, and as the company in question was probably the first to sell the metal under 10*s.* per pound, it is to be hoped that the Hall system will meet with increasing success.

Although a difficult one, the object was to obtain an alloy or compound consisting as nearly as possible of aluminum and iron alone. In the material made it will be found that the other elements present do not amount to more than 0.5 per cent., so that this intent may fairly claim to have been accomplished. The ingots, $2\frac{1}{2}$ inches square, were reduced by forging in the ordinary manner to bars $1\frac{1}{8}$ inches in diameter.

The consideration of the qualities of these alloys is divided into two heads: first, the material in the *cast*; second, the material in the *forged* state. The latter will be considered below. In preparing the cast alloys, nothing special was noted during the melting operations, but in all cases upon adding the aluminum "coruscation" was observed. This evolution of heat and light has also been observed when alloying aluminum with copper. Does not this tend to indicate that aluminum added to iron produces true compounds and not merely alloys, as it is well known in laboratory operations that the evolution of heat during the mixture of two substances is often a sign of chemical union? The curious properties of manganese-steel seem especially to prove that such a material is a true compound, and no doubt aluminum-steel may be classed under the same head.

Fluidity.—It is doubtful whether aluminum increases the fluidity of properly made steel, but if it does, the apparent increase or evolution of heat just referred to seem to the writer far more likely to account for any increase of fluidity that may occur rather than the so-called lowering of the melting-point. This is discussed more fully a little farther on.

As the aluminum increases to large amounts, over about 0.5 per cent., the metal becomes quite thick, "creamy," and sets quickly. No doubt this is partly caused by a considerable portion of the aluminum being oxidized to alumina and becoming entangled as slag, in the same way as occurs when silicon is oxidized to silica. Unless suitable flux is present, this excess of oxide cannot be carried off, the molten metal becomes less fluid, and causes the "thickness" noticed. There is a resemblance in this respect to the action of high percentages of silicon added to iron. When exceeding about 0.75 per

cent. Al, the molten material so rapidly "creams over" on the surface, that the production of sand-moulded articles is only effected with difficulty; in fact, it was only by using considerable care that the cast test-bars were obtained.* It must not be overlooked that the writer is here referring to comparatively high percentages. So far as his own experiments have gone, aluminum, as regards lower percentages, also acts in a similar manner to silicon under the same circumstances.

So much has been said as to aluminum lowering the melting-point of iron, that the writer wonders how and why this statement was originated, as it certainly seems to have no foundation in fact. This is confirmed by M. Osmond, who by means of the "Le Chatelier" pyrometer kindly made for the writer a special determination as to the melting-point of an alloy containing 5 per cent. Al. and about 94½ per cent. Fe. The sample in question did not show the slightest signs of fusion until a temperature of 1475° C. was reached. As mild steel by the same pyrometer was shown to fuse at about 1500° C., there could, therefore, have been little or no lowering of the fusion-point; and if this is the case with an alloy containing 5 per cent., still less can it be so with one containing only 0.1 per cent., which is the quantity usually described as being present when the so-called lowering is supposed to occur. It is especially satisfactory to have the testimony not only of M. Osmond but that of Mr. H. M. Howe,† who both agree with the writer on this point.

As this question is so important, it may be well to give a translation of M. Osmond's exact words: "I carried the heating of your sample up to 1465° C. At this temperature slight oscillations of the spot of light on the scale appeared to denote the approach of fusion, and in fact after the cooling-down I noticed slight indications of fusion on that portion of the sample which had been in contact with the tube. We can, therefore, suppose the actual fusion-point would be about 1475° C. As ordinary soft steel by the same pyrometer (Le Chatelier) is shown to melt at about 1500° C., you can see that your 5 per cent. aluminum sample had its fusion-point only lowered about 25°, or scarcely anything. It is, therefore, abso-

* Mr. Keep, even with cast-iron, found a decided decrease in fluidity by the addition of aluminum.

† Mr. Howe publicly questioned more than three years ago, in a leading technical journal, the inaccurate statements then circulated on this point.

lutely untrue that 0.1 per cent. of aluminum can lower the fusion-point many hundreds of degrees, as has been so commonly stated."*

In the writer's opinion, if any increase of heat or fluidity takes place by the addition of small amounts of aluminum, it may be due to either of the following causes, or to both combined. First, there may be evolution of heat, owing to oxidation of the aluminum, as the calorific value of this metal is very high—in fact, higher than silicon. According to Berthollet, the conversion of aluminum to Al_2O_3 equals 7900 cal.; silicon to SiO_2 is stated as 7800.

This oxidation is probable, as in the writer's experiments the addition of aluminum to iron was always accompanied by more or less, often considerable, loss of the former metal. This is also proved by the fact of its appearing in the oxidized state in a white powder upon the side of the ingot-mould, as well as in the slag; also from the fact that when small amounts are added they are rarely found in the product when it is analyzed. In the 5 per cent. specimens previously referred to, the writer found that no less than 2 per cent. was wasted. Against the supposition that by oxidation increase of heat is obtained must, however, be placed the fact that when the aluminum exceeds, say about 0.75 per cent., the less fluid is the product and the more metal is wasted; but at the same time the explanation first offered might still hold good as regards the lower percentages, say under 0.5 per cent.

Secondly, and more probably the correct explanation. As proved in a very clear and able manner by Mr. Galbraith in his interesting paper to the Iron and Steel Institute in May last, the fluidity of molten iron and steel depends not only upon the amount of heat imparted by external means or by oxidation, but often upon the quantity of intermingled slag or oxide of iron. He found that in certain cases in excessively over-oxidized steel, although the heat of the furnace was *more* than usual, the product was still lacking in fluidity. It would be interesting to have the point followed up and thoroughly tested by means of the "Le Chatelier" pyrometer. As supporting this, it may be observed that the well-known German metallurgist, Ledebur, believes the *rôle* of aluminum "to be no other than that of destroying the dissolved oxide of iron present."

That such over-oxidized metal possesses different properties as

* What do Messrs. Hunt, Langley and Hall say in their paper when referring to aluminum-bronze? "The melting-point of 10 per cent. bronze is 1700° F., a little *higher* than that of ordinary brass." No lowering of melting-point here, although h 10 per cent. of aluminum is present.

compared with ordinary steel or iron is very clear; for the writer has had brought under his notice certain cases of over-oxidized iron which continue to rise, give off gases, and prove unsound in the ingots, notwithstanding the addition of even considerable quantities of aluminum. They are, of course, exceptional, and to the writer seem to be explained by the facts pointed out by Mr. Galbraith.

The loss of the metal in melting is variable, but in nearly all cases amounts to a considerable proportion of the percentage added. The writer has seen it stated that there is no loss, although at the same time it has been mentioned that a considerable scum of alumina was noticed. The two facts are inconsistent, as, if the latter was present, it would appear that of necessity more or less of the metal added must have been oxidized, and consequently not present in the final product.

Soundness.—Does the addition of aluminum produce sounder material than silicon, manganese, or silicon combined with manganese? Whilst it has been found to act powerfully in this respect like these metals, still it does not follow, as much of the literature on the subject would have us believe, that blow-holes are quite done away with, and that the steel-founder may come down to his works in the morning without paying the usual visit to the scrap-heap. The art of the steel-founder is not merely the production of sound steel, which by increased knowledge, and choice of alloys for producing soundness, is, whilst still complicated, not so difficult as formerly, but consists, amongst other things, in the careful attention to the preparation of the moulds into which the molten steel is poured. There is no rapid or royal road to the production of sound steel castings; this is only attained by long experience combined with specialized knowledge. Mere attention to the chemical composition of the steel will not produce the desired results; and the writer hopes that these words will be remembered by those who are anxious to embark in the production of steel castings, and who consider from reading certain literature that it is perhaps only necessary "to buy so much aluminum, add 0.1 per cent., lower the melting-point, and the result will be perfection."

One cannot be surprised that strong hopes are raised when, shall we say, ill-advised trade-pamphlets make such statements that by the addition of aluminum to iron—mark the number, ye Muses—*nine* distinct and important qualities are at once conferred. No wonder ferrum has had hitherto to be classed amongst the baser

metals, but is now to be rehabilitated by the nine-mused metal aluminum.

The writer too has seen specimens of so-called ferro-aluminum sold at shillings per pound, but which upon analysis gave aluminium "Nil" per cent., sulphur 0.24 per cent., phosphorus 0.3 per cent.!

Not a little amusing, too, was the ingenious firm who, to meet ideas all round, offered a compound described as "aluminum-ferro-silicon," which, according to the statements of the circular, should not only banish honeycombs from the realm of the steel-founder, but should at the same time "*de-phosphorize* and cleanse the metal of any excess of carbon!" Such an alloy, to quote from the circular, could well claim "to be of value in almost all ordinary steel-practice." Is there any wonder that steel-makers have become skeptical and suspicious as to new processes?

As high as 0.75 per cent. aluminum, or even more, may be admitted without much interference with the ductility or toughness of the cast specimens, as will be seen from Table IV. These experiments prove that more aluminum may be present in the cast material than silicon in cast silicon-steel before brittleness sets in. After reaching the region of brittleness, and by adding still higher percentages of aluminum, there is no return of strength—on the contrary; so that this material in no way resembles the curious property noted in this respect with regard to manganese-steel.

Aluminum seems specially powerful in absorbing gases, and in Mr. Richards' book on the metal he states that M. Dumas found on heating it in a vacuum that towards a red heat considerable quantities of gases were given off, their composition being 88 c.c. H, and 1.50 c.c. CO_2.

In a pamphlet issued by the Alliance Aluminium Co., they state on this point that—

"As regards the affinity of aluminum for oxygen, it ranks next to the alkaline and ferro-alkaline metals, that is, its power of affinity for oxygen is much greater than that of the metals proper, such as manganese for example. Supposing the chemical equivalent of oxygen to be represented by 100, it is found that

Silicon,	278
Manganese,	345
Aluminum,	171

"Comparing the absorption-power of these three bodies, it will be found that

171	parts of	aluminum	absorb	150	parts in	weight of	oxygen.
171	"	silicon	"	123	"	"	"
171	"	manganese	"	49	"	"	"

" Relatively the absorption-powers would be therefore

Aluminum,	5.00
Silicon,	4.10
Manganese,	1.66 "

It is not, however, clear how the above figures are arrived at; and they certainly seem inaccurate as regards silicon.

Silicon (28 At. wt.) converted to silica absorbs 32 of oxygen; or the weight fixed by the Alliance Co., that is, 171 grms., absorbs 195.35 grms. of oxygen. This would quite alter the absorption table as given by them, which would read as follows, silicon being now at the head:

171	parts of	silicon	absorb	195	parts of	oxygen.	
171	"	aluminum	"	150	"	"	
171	"	manganese	"	49	"	"	

The relative powers would then be

Silicon,	6.60	not	4.10
Aluminum,	5.00	(the same)	5.00
Manganese,	1.66	"	1.66

Bending-Tests.—Table IV. gives the bending-tests, from which it will be seen that the annealed samples up to .85 per cent. bent double cold; those unannealed were not so good. The gradual increase of aluminum is very clearly shown to reduce the toughness both of the unannealed and annealed specimens. They were sound with exception of samples E and F, but not more so than cast silicon-steel of corresponding percentages.

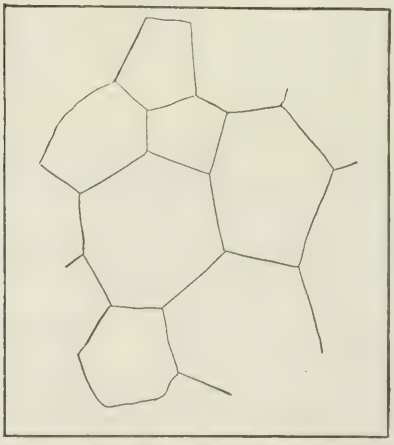
Hardness.—As might be expected from the soft nature of the metal aluminum, its addition to iron does not add materially to the hardness, again in this respect resembling the effect of silicon. Seven to eight per cent. cast aluminum-steel may be readily drilled and filed. This point is dealt with more fully under " hardness " in treating of the forged material.

Neither does its addition, unlike manganese, interfere with the magnetic susceptibility of the iron present. Experiments are in hand to determine what is the actual effect in this direction, and also as regards conductivity.

Appearance of Fracture.—Aluminum in the cast specimens opens the grain, *i.e.*, increases the size and coarseness of the crystals, and as the higher percentages are reached it is somewhat difficult to distinguish the fractures from those of the cast silicon-steel described in the writer's paper at last year's Paris meeting of this Institute. The crystals in the 5 per cent. and upwards specimens become very large and cleave somewhat after the nature of spiegeleisen. This accounts for the increasing brittleness of the cast material, and neither the fracture nor the brittleness accompanying it in such samples can be changed by annealing. A curious fact was noticed, viz., that the freshly broken fractures had a very pronounced odor of sulphuretted hydrogen, although the sulphur present did not exceed more than 0.06 per cent. This remained for several days. Figures 3 and 4 give the appearance of the 5 per cent. aluminum and silicon samples in their cast state.

M. Osmond examined microscopically the fracture of a 5 per cent. specimen (previously heated to 1545° C.) of cast aluminum-steel (see Fig. 1), and says, " It is a mosaic of polyhedrons of striking

FIG. 1.



5% Aluminum-Steel Previously Annealed at 1545° C.
Magnified 45 Linear.
(*by M. Osmond.*)

regularity, the interior of the polyhedrons being of remarkable homogeneity. What I found most interesting was a comparison of the difference between it and the structure of extra soft steel previously heated to a similar temperature. In the aluminum-steel the polygons are nearly regular hexagons or pentagons. The soft

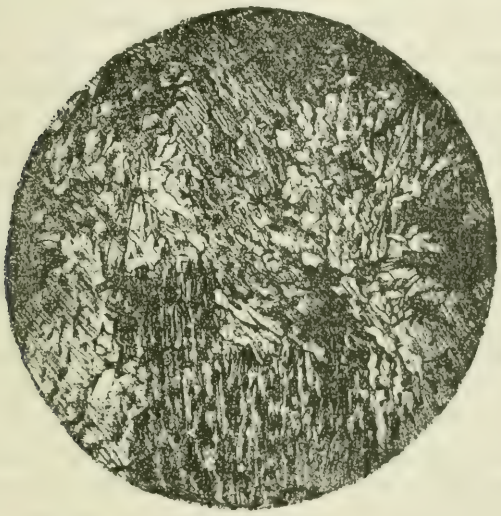
steel sample (see Fig. 2), on the contrary, shows itself to be composed of elements which arrange themselves parallel to each other and in clusters of long narrow section."

Forged Aluminum-Steel.

Malleability.—As high as 5.60 per cent. aluminum may be present before malleability ceases. This is about the same limit as in the silicon-steel described by the writer.

It may be pointed out as regards the specimens under discussion,

Fig. 2.



EXTRA MILD STEEL AFTER ANNEALING AT 1390° C
MAGNIFIED 45 LINEAR
(by M. Osmond)

that they are very low in manganese. Notwithstanding this, the malleability is satisfactory.

Mechanical Properties, etc.—As regards bending-qualities, Table IV. shows the results. Up to 2.24 per cent. aluminum, the annealed samples bend double cold, but upon reaching 5 per cent. a great diminution of strength takes place, and annealing practically produces no effect.

In the testing-machine (Table I.), it will be seen that the elongations and reductions in area, although in some cases irregular, are high, the annealed samples showing, as might be expected, the better results. The samples were all tested on short lengths, 2 inches

(Whitworth standard pattern .7979 inch diameter, 0.5 inch area), and the elongations are very good in those up to about $1\frac{1}{2}$ per cent. of aluminum; afterwards a gradual diminution occurs. This is brought out graphically in Diagram I. Table II. and Diagram II. may also be of interest as showing the effect of similar percentages of silicon and aluminum. Comparing the sample of aluminum-steel containing 1.6 per cent. aluminum and 0.21 per cent. carbon, with silicon-steel of 1.60 per cent. silicon and 0.19 per cent. carbon, the former has an advantage in elongation of 1 per cent. A characteristic of the aluminum-steel is that the reduction in area is greater; in this respect it seems to resemble chrome-steel. * There is, however considerable difference in the tensile strength of the two alloys, the aluminum-steel being several tons less. The same remark applies to the permanent set point, which in the annealed aluminum-steels is fairly low. Numerous comparisons might be drawn between the two steels, especially as the test-bars in both series of experiments were exactly the same in length and section, and were treated on all points similarly. Reference to the diagrams and tables in each case will be found of interest. One difference clearly brought out is, that the tensile strength of iron is not increased so much by an addition of aluminum as of silicon.

In two exceptional samples the writer was able to get the carbon down to as low a point as 0.1 per cent., with aluminum, 0.25 and 0.96 per cent. respectively (these not forming a part of the series, are not included in Table I.). The elastic limits of the two annealed specimens were respectively 13 and 11 tons, breaking-loads 23 and 22.5 tons (per square inch of original area), very clearly showing the slight influence of aluminum both upon the elastic limit and breaking load.

It has been stated that very small percentages of aluminum considerably raise the limit of elasticity. It will be seen from the table of tests, and specially from the two almost carbonless samples, just quoted, that this increase is slight, if any, as compared with the ordinary steel, and also that aluminum does not raise the elastic limit so much as silicon. Seeing that aluminum is an exceptionally soft and malleable metal, possessing a tensile strength of 8 to 10 tons, it is hardly to be expected that this should have been otherwise, and it is a pity that such incorrect statements as above men-

* Messrs. Hunt, Langley, and Hall, in the paper before referred to, speaking of the metal aluminum, say, "The flow of aluminum under tensile tests is very *local*."

tioned are started without foundation in fact. The elastic limit of samples should always be determined on annealed specimens, as by cold forging or rolling, the elastic limit of even mild steel can be readily raised several tons per square inch; and it would hardly be correct to call this the normal permanent set of the sample under examination. Aluminum and silicon, therefore, stiffen iron but little, whilst carbon, chromium, tungsten, manganese and nickel clearly increase rigidity; and this from their nature and hardness might be expected.

Compression-Tests.—Samples A and I (0.15 and 5.60 per cent. Al) were each subjected to a compression-stress of 100 tons per square inch. Table V. gives the full details. It will be noticed that whilst the specimen containing 0.15 per cent. Al shortened 51 per cent. in length under the above-named stress, that containing 5.60 per cent. Al shortened 36 per cent. Thus even the soft metal aluminum has to some, though very slight, extent stiffened the material.

As a comparison may be mentioned samples B and G (Table III. of my last year's paper) of silicon-steel (containing 0.79 and 5.20 per cent. Si), which shortened under the same stress 50 per cent. and 32 per cent. respectively.

Hardness.—As might be expected from the soft nature of the metal aluminum, its addition to iron does not confer appreciable hardness when in either the forged or cast state. In this respect also it resembles silicon steel. The forged material containing 5 per cent. is very soft, and is easily drilled or filed.

The action of aluminum may therefore be classed along with that of silicon, sulphur, phosphorus, arsenic, and copper, as giving no increase of hardness to iron, in contradistinction to carbon, manganese, chromium, tungsten, and nickel. Water-quenching upon either forged or cast aluminum-steel (in which carbon is practically absent) seems to produce no effect (Table VI.), the samples when dipped at even a welding heat being almost unchanged. *

Here again we have another addition to the list of elements which, alloyed with iron, do not produce "water-quenched hardness," if the term may be allowed. In fact, notwithstanding that manganese, chromium, tungsten, and nickel do impart a certain and

* It has been stated that the influence of aluminum on high-carbon or tool-steel, is to destroy its hardening action. This is quite incorrect; the writer has prepared steel containing 1½ per cent. aluminum and 1 per cent. of carbon, which hardens sufficiently to scratch glass. It is a pity such inaccurate statements are circulated.

considerable kind of hardness to iron, either in the cast or forged state, carbon retains its position as being the only element which confers upon iron the property of becoming hard by water-quenching, In steel made with the hard metals just mentioned, if the carbon present be low, no matter at what heat they are water-quenched, the characteristic hardness of carbon-steel, such as scratching glass or forming the edge on a cutting-tool, is wanting.

Fracture.—Aluminum causes the tough percentages, both in the cast and forged material, to break with dark fibrous fracture, similar to wrought-iron of good quality, only much darker in color. This appearance in some of the specimens is very curious, as small patches of bright, fine crystals are to be seen surrounded by a remainder of black and dark fibrous material.

In several of the tested tensile specimens a rough and ribbed surface is noticed, proving looseness of texture.

Weldability.—Samples B (0.61 per cent. Al), D (0.72 per cent. Al) and G (2.20 per cent. Al) were tested for welding, but unsuccessfully. It would be interesting to try electric welding.

The writer has always found that these special steels or alloys, such as silicon, manganese, or chromium steel, have slight, if any, welding-properties.

Applications.

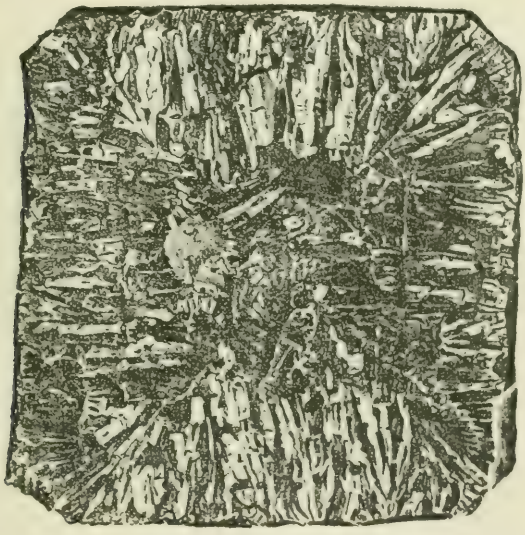
Speaking generally of the application of aluminum to the manufacture of iron and steel, the usual amount stated to be requisite for producing good results is about 0.1 per cent.; but in many cases, as already pointed out, this would be too little. Supposing, however, that an average percentage of 0.1 per cent. or 0.15 per cent. was necessary, and assuming the aluminum to be sold at 8*s.* per lb., the expense of such addition would mean an extra cost of 18*s.* and 27*s.* per ton respectively; whereas if as much as even 0.5 per cent. of silicon is required to do the same work, it does not cost more than 4*s.* 6*d.* per ton. On a large output of steel castings this means a serious difference, and in the above nothing is credited to the latter method by the gain of adding so much ferro-alloy, which afterwards counts as steel.

In making these statements, the writer in no way wishes to disparage the efforts of those who have devoted so much time and attention to the production of aluminum. Let it be rather an inducement to produce the metal more cheaply. If aluminum can eventually be brought to compete in price with existing alloys, there

is a probability of something being done in introducing its use on a larger scale. Aluminum may find a use for certain special purposes, such as in higher carbon-steel, where silicon does not seem to act so powerfully in producing soundness as in the case of milder steel with carbon at 0.5 per cent. and under.

Therefore, whilst for some special purposes aluminum may be employed in the manufacture of iron, at any rate with our present knowledge of its properties, this use cannot be large, especially when taking into consideration the fact of its comparatively high price. Its special advantage seems to the writer to be that it combines in

Fig. 3.



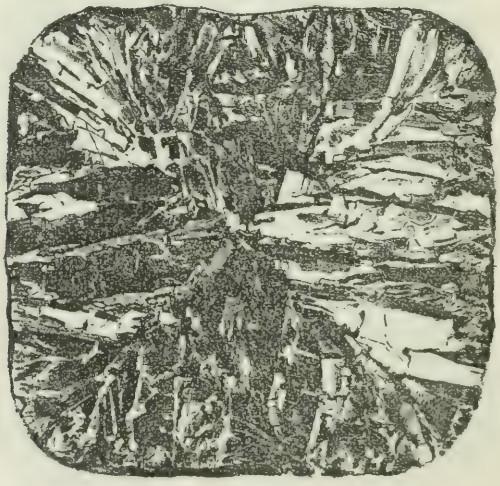
CAST 5% SILICON-STEEL. (ACTUAL SIZE)

itself the advantages of both silicon and manganese, but so long as alloys containing these metals are so cheap and aluminum dear, its extensive use seems hardly probable.

Points of Resemblance between Aluminum- and Silicon-Steel.— Whilst referring to the points of similarity between the mechanical properties of aluminum- and silicon-steel, there may also be mentioned an interesting communication sent some time ago to the writer by M. Osmond. He says: "I believe you were quite correct in stating in your paper on silicon and iron alloys that the action of these two metals was very similar. All that I know on this subject

goes to confirm this. In 1883 I made some aluminum-steel, and found its tensile strength and elongation were modified almost exactly as by the same proportions of silicon. Second, aluminum, like silicon, causes a precipitation of graphite in cast-iron. Third,* the cooling curve of your aluminum alloy is nearly the same as that of your 4.50 per cent. silicon alloy. Fourth, aluminum and silicon have almost the same atomic volume. I will only add that I quite agree with Mr. A. H. Allen, of Sheffield, that ordinary steel, made upon an acid-lined furnace, often contains a little aluminum; at least I found it to be so with Creusot steel, and by analytical methods which I believe to be accurate. I detected as high as

Fig. 4.



CAST 5% ALUMINUM-STEEL. (ACTUAL SIZE)

0.1 per cent. in one sample, yet nobody found that the melting-point was lowered. The detection of small quantities of aluminum is difficult, and requires great care, as reactions are likely to occur upon glass utensils of the laboratory."

Owing to the similarity in behavior of aluminum- and silicon-steel in the testing-machine, Table II. and Diagram II. have been prepared for comparison. The full details of the silicon-steel samples will be found in Table II. of the writer's paper to this Institute last year.

* M. Osmond has kindly furnished a tracing of the curve, see Diagram III.

It will be found from these that the samples selected, which fairly correspond as to their contents of aluminum and silicon, show the following results:

The limit of elasticity does not at first differ materially, but in the higher percentages the silicon-steel gives the higher results. The same remark applies to the tensile strength.

As regards elongation, contraction of area, and bending-tests, the differences are very slight. In fact, as a whole, it may be said that the two metals, aluminum and silicon, produce almost corresponding effects as regards mechanical properties when mixed or alloyed with the metal iron. These facts are well illustrated by Diagram II.

The further points of similarity may be summed up as follows: Producing soundness; increasing the coarseness and size of the iron crystals; annealing, in the higher percentages of both steels, produces very little change in the molecular structure; reducing fluidity in the higher percentages; neither metal appreciably hardens the iron to which it has been added, no matter whether such alloy be either in the cast or forged state; water-quenching produces no appreciable difference; the alloys produced by both metals cease to be malleable with about corresponding percentages; neither alloys weld satisfactorily.

The principal differences are that aluminum-steel in its *cast* state will carry a somewhat larger percentage of its metal before becoming brittle as compared with silicon-steel in its cast state, but after forging this appears to be removed. Larger amounts of silicon seem to be required to produce the same soundness, but, as before pointed out, if it were possible to add pure silicon,* it is probable that this difference would not be found to exist. The highest ferro-silicon yet made does not contain more than 20 per cent. of silicon. Finally, silicon-steel appears to be more red-short.

Effect of Aluminum upon Manganese-Steel and Spiegeleisen.— Believing that aluminum, like silicon, would cause a precipitation of graphite, the writer added between three and four per cent. to ordinary spiegel (12 and 25 per cent. manganese). The result in both cases was a complete change, from the well-known spiegel fracture to that of ordinary close No. 3 gray pig-iron. Although there

* If Messrs Cowles would, with their electric furnace, supply us with say 50, 70 or 90 per cent. ferro-silicon, there would probably be a considerable use for such alloy. Ledebur considers that pure silicon would act more powerfully.

had been this great alteration in the form of the carbon, no change in the non-magnetic properties of the iron was apparent. This experiment, therefore, clearly proves that the neutralization of the special magnetic qualities ordinarily possessed by iron, is in manganese compounds quite independent of the particular state or form in which the carbon present exists. The difference between the molecular structure of gray and white iron, and in the cases under consideration between gray and white spiegel, makes this fact somewhat striking.

The following table gives the analyses of the samples referred to; also of a sample of the same spiegel to which silicon has been added, with even more powerful results—conversion of the greater part of the combined to graphitic carbon:

	C.	Gr.	Si.	S.	P.	Mn.	Al.	Remarks.
Spiegel—								There was considerable loss of aluminum. Non-magnetic susceptibility unaltered.
12 per cent. before addition of Al....	4.80	None	...	...	...	13 65	...	Fracture changed from usual pronounced "spiegel" appearance to that of No. 3 iron.
12 per cent. after addition of Al....	.93	3.45	1.30	.01	.05	11.75	3.19	
25 per cent. before addition of Al....	4.10	...	...	...	...	25.20	...	Ditto, except that the change was not quite so decided.
25 per cent. after addition of Al....	2 30	1.88	2.16	.02	.10	22.16	1.24	
12 per cent. before addition of Si.....	4.10	...	...	...	...	11.00	...	Fracture changed to No. 3 iron. No change in non-magnetic susceptibility.
12 per cent. after addition of Si.....	.43	3.40	3.73	.01	.05	9.50	...	

In conclusion, the writer regrets that the paper is not so complete as he would like, but owing to the press of ordinary work there has not been time to complete further experiments in hand, which might possibly have thrown further light upon this interesting subject.

TABLE I.—*Tensile Tests of Annealed and Unannealed Bars; also Specific Gravity, Forging, Welding and Bending Tests.*

Analyses and specific-gravity determination made on drillings from the ingot; tensile strength, elastic limit and elongation were tested with a Whitworth machine, and will be found plotted on Diagram I. Original diameter of test-bars, 0.7979 inch; original area, 0.5 sq. in.; extension measured on a length of 2 in. Bars marked *u* in first column were tested unannealed. Those marked *a* were annealed before testing.

Series and Mark Referred to in Paper.	Analysis per Cent. C.	Si.	S.	P.	Mn.	Al.	Calculated Percentage of Al.	Loss (Approx.) per ct. of Al.	Specific Gravity.	How Ingot Forged.	Limit of Elasticity in Tons per square inch.	Amt. of Stretch in Inches at Elastic Limit.	Breaking load in Tons per sq. in on Original Area.	Total Extension per ct. measured on 2 in.	Reduct'n of area per cent.	Appearance of Fracture of Test-Bar.	Appearance of Fracture of Annealed Forged Bar from which Test-Bar was Prepared.	Bending-Tests of Annealed Forged Bars, $\frac{1}{8}$ in. wide × $\frac{1}{8}$ in. thick.	Welding-Tests.
u 1167	.22	.09	...	...	.07	.15	.25	.10	...	Very well.	21.0	...	29.0	36.70	62.9	Granular; very dark; a little roaked; smooth on outside.			
a 1167	...	...	...	..	...	...	...	...	...	...	20.0	...	25.0	41.30	63.82	Dark granular; a little roaked; smooth on outside.	Close fine granular.	Bent double cold.	
u 1167 A	.15	.18	.10	.04	.18	.38	.50	.12	...	Fairly well.	23.0	...	30.0	37.85	58.18	Dark granular.			
a 1167 A	...	...	...	...	...	...	...	...	...	...	20.0	...	26.0	40.35	60.74	Dark granular; smooth on outside.	Close fine granular.	Bent double cold.	
u 1167 B	.20	.12	...	...	.11	.61	.75	.14	7.781	Rather shelly.	21.5	...	28.0	38.40	54.50	Dark granular; several specks in body.			Would not weld.
a 1167 B	...	...	...	...	...	...	...	...	...	...	18.0	...	25.5	40.50	61.98	Dark granular; rough on outside.	Fine close granular; much closer before annealing.	Bent double cold.	
u 1167 C	.18	.16	.09	.03	.14	.66	1.00	.34	...	Very well.	20.5	...	29.0	33.35	49.86	Dark granular; several long specks in body.			
a 1167 C	...	...	...	...	...	...	...	...	...	...	18.0	...	27.0	33.0	52.14	Dark granular; rough on outside.	Fine grain.	Bent double cold.	
u 1167 D	.17	.10	...	...	.18	.72	1.25	.53	7.755	Very well.	22.0	...	28.0	40.0	60.74	Dark granular.			Would not weld.
a 1167 D	...	...	...	...	...	...	...	...	...	...	18.0	...	25.0	47.10	64.86	Dark granular; smooth on outside.	Fine close grain; much closer before annealing.	Bent double cold.	
u 1167 E	.26	.15	.08	.04	.11	1.16	1.50	.34	...	Very well.	23.0	...	33.0	32.05	51.46	Dark fine granular; smooth on outside.			
a 1167 E	...	...	...	...	...	...	...	...	..	...	21.0	...	29.0	34.40	53.02	Dark granular; smooth on outside.	Fine grain.	Bent double cold.	
u 1167 F	.21	.18	...	...	.18	1.60	2.00	.40	7.6237	Rather shelly.	20.0	...	31.0	32.70	52.14	Dark granular; rough on outside.			
a 1167 F	...	...	...	...	...	...	...	...	...	...	13.0	...	26.0	36.35	67.06	Dark granular; rough on outside.	Coarser.	Bent double cold.	
u 1167 G	.21	.18	.09	.03	.18	2.20	2.50	.30	...	Fairly sound.	21.0	.0080	31.0	22.75	27.80	Crystalline, rather coarse.			Would not weld.
a 1167 G	...	...	...	...	...	...	...	...	...	...	19.0	.0095	28.0	34.87	47.12	Dark granular; fibrous; very rough on outside.	Coarser.	Bent double cold.	
u 1167 H	.24	.18	...	...	.32	2.24	3.50	1.26	7.554	Fairly well.	21.5	.0085	32.5	20.67	24.64	80 per cent. outer circle, crystalline: 20 per ct. inner circle finer crystalline; rather coarse.			
a 1167 H	...	...	...	...	...	...	...	...	...	...	18.5	.0133	28.5	33.02	48.62	Dark granular; fibrous.	Coarser.	Bent double cold.	
u 1167 I	.22	.20	.08	.03	.22	5.60	6.00	.40	...	Very shelly	None visible	...	38.0	3.67	3.96	Crystalline; lustrous; broke in radius.			
a 1167 I	...	...	...	...	..	...	...	...	...	...	27.0	.0050	36.0	6.45	6.16	Crystalline; lustrous; broke in radius.	Coarse.	Broke at 16°.	
1167 J	.26	.33	.08	.03	.25	9.14	10.00	.86	6.6726	No Test-Bar made; would not forge.									

Diagram I. *Showing the tensile strength, elastic limit, and elongation per cent of annealed and un-annealed forged Aluminum-Steel. Plotted from Table I.*

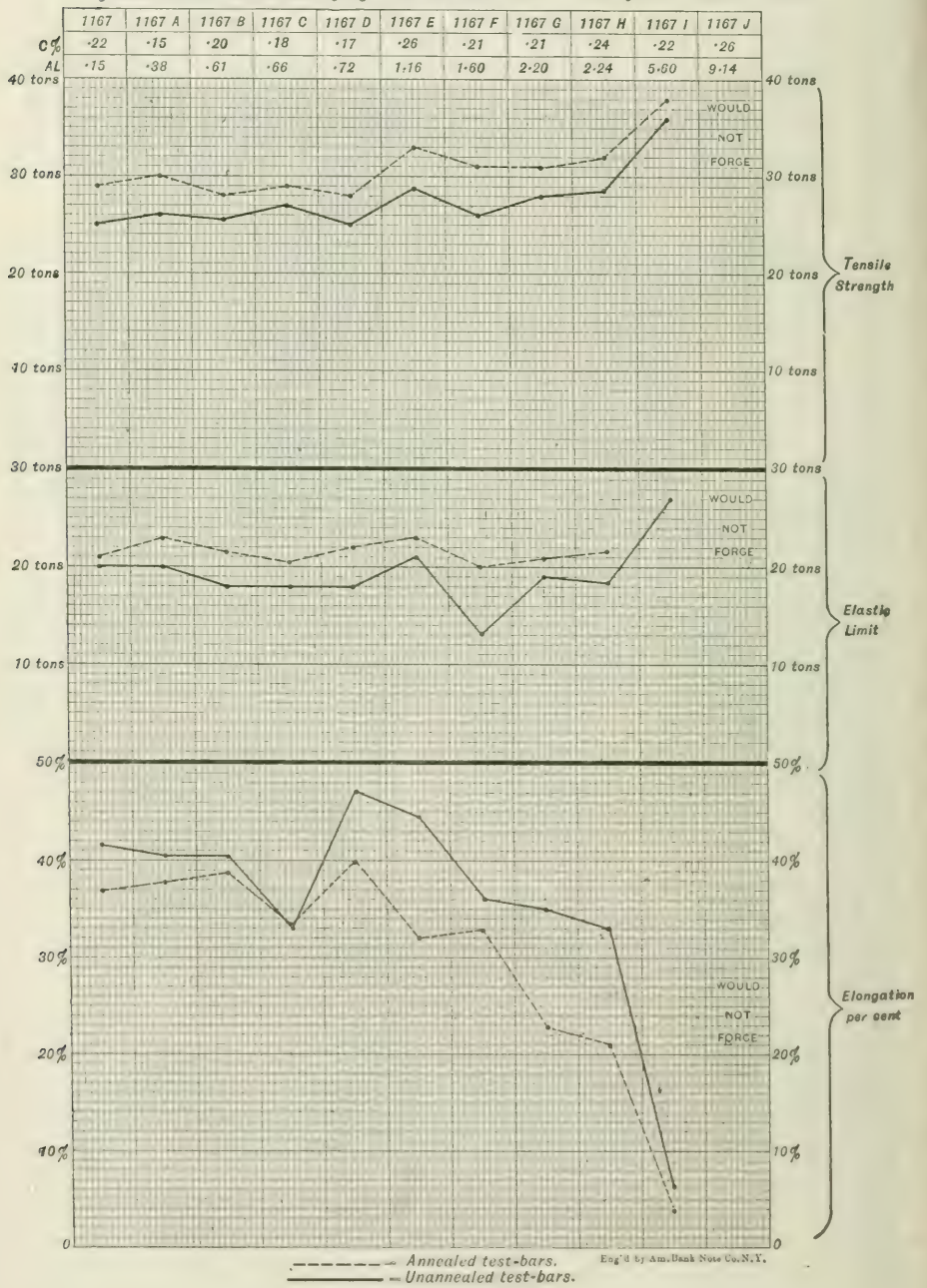

1167 | 1167 A | 1167 B | 1167 C | 1167 D | 1167 E | 1167 F | 1167 G | 1167 H | 1167 I | 1167 J
C% | ·22 | ·15 | ·20 | ·18 | ·17 | ·26 | ·21 | ·21 | ·24 | ·22 | ·26
Al | ·15 | ·38 | ·61 | ·66 | ·72 | 1·16 | 1·60 | 2·20 | 2·24 | 5·60 | 9·14
40 tons
30 tons
20 tons
10 tons
WOULD NOT FORGE
Tensile Strength
50%
40%
30%
20%
10%
0
Elastic Limit
Elongation per cent
Annealed test-bars.
Unannealed test-bars.
Eng'd by Am. Bank Note Co. N.Y.

TABLE II.—*Comparison Table of Forged Silicon and Aluminum Steel, both Materials having been Annealed.*

	Per Cent.			Limit of Elasticity in Tons per Square Inch.	Breaking Load in Tons per Square Inch.	Extension Per Cent. on 2″.	Reduction of Area, Per Cent.	Bending Test of Annealed Forged Bars.	Remarks.
	C.	Si.	Al.						
Si. Steel, **A**	.14	.24	...	15.17	25.0	37.55	60.74	Bent double cold.	The tests of the silicon-steel are fully described in Table I. of the writer's paper, "Alloys of Iron and Silicon," read last year at the Paris Meeting of the Iron and Steel Institute.
Al. Steel, **A**	.15	...	.38	20.0	26.0	40.35	60.74	Bent double cold.	
Si. Steel, **B**	.18	.73	...	19.0	29.50	34.02	52.66	Bent double cold.	
Al. Steel, **C**	.18	...	.66	18.0	27.0	33.0	52.14	Bent double cold.	
Si. Steel, **C**	.19	1.60	...	25.0	33.0	35.0	54.52	Bent double cold.	
Al. Steel, **F**	.21	...	1.60	13.0	26.0	36.35	67.0	Bent double cold.	
Si. Steel, **D**	.20	2.18	...	25.50	34.0	36.50	59.96	Bent double cold.	
Al. Steel, **H**	.24	...	2.24	18.50	28.50	33.0	48.62	Bent double cold.	
Si. Steel, **H**	.26	5.53	...	29.00	39.00	0.70	2.00	Would not bend.	Test-bar in both cases broke in the threads during test.
Al. Steel, **I**	.22	...	5.60	27.00	36.00	6.45	6.16	16° broken.	

Note.—These comparisons will be found plotted on Diagram II.

Diagram. II. *Showing comparison between forged Silicon and Aluminum-Steel. Plotted from Table II, both materials having been annealed.*

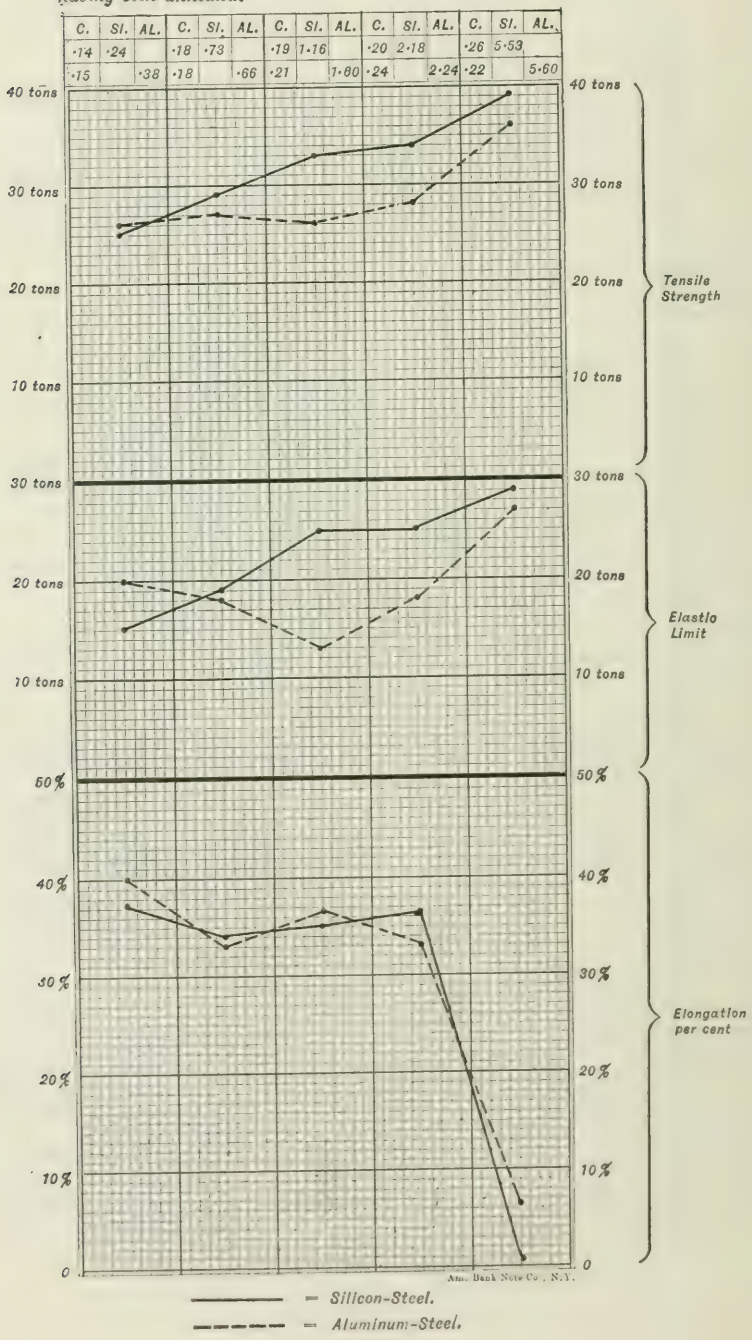


TABLE III.—*Literature Relating to the Metal Aluminum.*

Year.	Date.	Nature of Notice.	Where Given.	Author.	Titles.
				.....	Aluminum-Bronze.
1863.	May.	Press Notice.	*Morning Star.*		
1863.	.....	Paper.	British Association.	Sir Lowthian Bell.	On the Manufacture of Aluminum.
1888.	August.	Do.	American Association for the Advancement of Science.	Mr. W. J. Keep.	Influence of Aluminum on Cast-Iron,
1889.	.....	Do.	American Institute of Mining Engineers.	Do.	Aluminum in Cast Iron.
1890.	May.	Do.	Iron and Steel Institute.	Do.	Aluminum in Carburetted Cast-Iron.
1890.	Feb.	Do.	American Inst. of Min. Engrs., Washington.	Do.	Aluminum in Wrought Iron and Steel Castings.
1890.	Feb.	Do.	Do.	Do.	Aluminum and other Metals Compared.
1889.	April.	Do.	Inst. of the North of England Mining Engrs.	Mr. H. Boby.	Aluminum.
1889.	Sept.	Do.	British Association.	Mr. J. H. Dagger.	The Uses of Aluminum.
1889.	.....	Do.	Royal Institute of Great Britain.	Sir Henry Roscoe.	Aluminum.
1890.	Feb.	Do.	Amer. Inst. of Mining Engrs., Washington.	Messrs. Hunt, Langley, and Hall.	The Properties of Aluminum.
1890.	"	Do.	Do.	Mr. E. H. Cowles.	Physical Properties of some of the Alloys of Manganese, Copper and Aluminum.
1890.	"	Do.	Do.	Mr. W. P. Black.	Note on the Use of Aluminum in the Construction of Instruments of Precision.
1890.	"	Do.	Do.	Mr. E. H. Cowles.	Aluminum Bronze and Brass as Suitable Materials for Propellers.
1890.	"	Do.	Do.	Mr. Oberlin Smith.	Aluminum in the Drawing Press.
1890.	"	Do.	Do.	Mr. F. P. Dewey.	The Héroult Process of Smelting Aluminum Alloys.
1886.	Oct. 16.	Press Notice.	*American Machinist.*	Mr. T. A. West.	Production of Aluminum Bronze.
1886.	Nov. 12.	Do.	*Industries.*	.....	Preparation of Aluminum by the Electric Furnace.
1887.	Aug. 30.	Do.	*Engineering and Mining Journal.*	.....	Aluminum Alloys for Mine and Mill Machinery.
1888.	Aug. 30.	Do.	*Iron Age.*	.....	Influence of Aluminum on Cast-Iron.
1888.	Aug. 31.	Do.	*Engineering.*	.....	Aluminum in Cast-Iron.
1888.	November.	Do.	*Industries.*	.....	Héroult Aluminum Process.
1889.	Dec. 12.	Do.	*Iron Age.*	Mr. Romaine Cole.	Aluminum.
1890.	Jan. 16.	Do.	Do.	Do.	Aluminum-Copper.
1890.	May 2.	Do.	*Industries.*	.....	Aluminum and Its Uses.
1890.	Aug. 16.	Do.	*American Manufacturer.*	Mr. T. A. Wright, Pittsburgh.	Aluminum in the Manufacture of Russia Sheet Iron.
1890.	May.	Do.	*Le Genie Civile*, vol. xvi.	M. C. Talansier.	Héroult's Process for Manufacturing Aluminum.
1889.	.....	Do.	*Annales des Mines*, vol. xvi.	.....	Grabau's Method of Making Aluminum.
1890.	.....	.....	*Proceedings of Institute of Civil Engrs.*, vol. ci.	Mr. Jehon.	" " " "
		Pamphlet.	.....	The Alliance Aluminum Co., Limited.	Aluminum : Its Uses and Alloys.
		Do.	.....	Do.	Aluminum and Its Alloys.
		Do.	.....	The Cowles Syndicate Co., Limited.	Aluminum Alloys.
1887.	.....	Book.	.....	Mr. J. W. Richards.	Aluminum : Its History, etc.

TABLE IV.—*Bending-Tests of Cast and Forged Aluminum-Steel.*

Series and Mark Referred to in Paper.	Al. Per Cent.	CAST. Size of Bar, 6″ × 1″ × ¼″.		FORGED. Size of Bar, 4″ × 1 1/16″ × ⅛″		Remarks.
		Unannealed.	Annealed.	Unannealed.	Annealed.	
1167.	.15	79° broken.	Bent double.	Bent double.	Bent double.	
1167 A.	.38	Bent double.	Bent double.	Bent double.	Bent double.	
1167 B.	.61	174° unbroken.	Bent double.	Bent double.	Bent double.	
1167 C.	.66	88° broken.	Bent double.	Bent double.	Bent double.	
1167 D.	.72	59° broken.	Bent double.	Bent double.	Bent double.	
1167 E.	1.16	31° broken.	54° broken.	Bent double.	Bent double.	
1167 F.	1.60	28° broken.	133° broken.	Bent double.	Bent double.	
1167 G.	2.20	Would not bend.	43° broken.	Bent double.	Bent double.	
1167 H.	2.24	16° broken.	32° broken.	Bent double.	Bent double.	
1167 I.	5.60	Would not bend.	Would not bend.	15° broken.	16° broken.	Very brittle.
1167 J.	9.14	Would not bend.	Would not bend.	Would not forge.	Would not forge.	Very brittle.

TABLE V.—*Compression-Tests of Aluminum-Steel.*

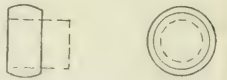
Series and Mark Referred to in Paper.	Al.	Treatment.	Length before being Tested.	Diameter before being Tested.	Total Load Applied per Square Inch.	Reduction in Length by Compression Load of 100 Tons per Square Inch.	Diameter Increased by Compression Load of 100 Tons per Square Inch.	Dotted Lines show Specimens before being Tested. Full Lines show Specimens after being Tested.
			Inch.	Inch.	Tons.	Per Cent.	Inch.	
1167 A.	.15	Unannealed.	.9970	.7970	100	51.00	1.2771	
1167 C.	.66	Unannealed.	1.0025	.7965	100	49.077	1.1735	
1167 E.	1.16	Unannealed.	.9985	.7960	100	47.22	1.1324	
1167 F.	1.60	Unannealed.	1.000	.7972	100	48.10	1.1448	
1167 H.	2.24	Unannealed.	1.0011	.7969	100	45.40	1.1145	
1167 I.	5.60	Unannealed.	.9992	.7966	100	36.57	1.0378	

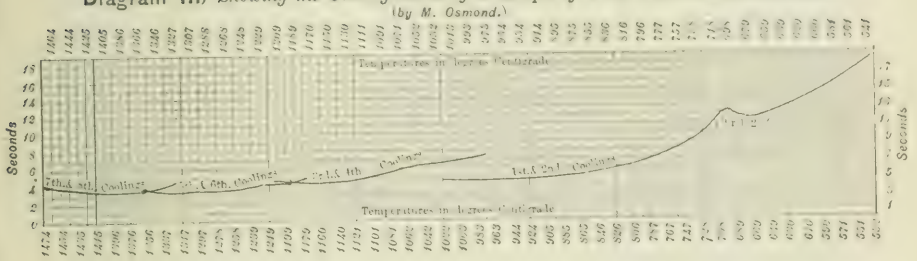
TABLE VI.—*Water-Quenching Tests of Forged Aluminum-Steel.*

Mark Referred to in Paper.	Al., Per Cent.	Treatment No.	How Filed before Treatment.	How Filed after Treatment.	Bending-Tests after Treatment.	Remarks.
1167 B.	.61	1	Soft.	No change.	Double slight crack.	The untreated forged bending-samples of this percentage bent double cold.
"	"	2	"	Shade harder.	Double no crack.	
"	"	3	"	Hardest of the three, but still quite soft.	Double very slight crack.	
1167 D.	.72	1	Soft.	Slightly harder.	Double cracked half way across.	The untreated forged bending-samples of this percentage bent double cold.
"	"	2	"	Slightly harder.	148° broken.	
"	"	3	"	Slightly harder.	133° broken.	
1167 F.	1.60	1	Soft.	Slightly harder.	115° broken.	The untreated forged bending-samples of this percentage bent double cold.
"	"	2	"	Slightly harder.	156° broken.	
"	"	3	"	Slightly harder.	121° broken.	
1167 I.	5.60	1	Soft.	Soft.	23° broken.	The untreated forged bending-samples of this percentage broke at 15°
"	"	2	"	Slightly harder than 1.	12° broken.	
"	"	3	"	About the same as 2.	16° broken.	

Treatment
- 1 = heated to low red heat and plunged into cold water.
- 2 = heated to bright red heat and plunged into cold water.
- 3 = heated to welding-heat and plunged into cold water.

Note.—These experiments clearly prove that aluminum, like silicon, and unlike carbon, does not confer upon iron that is carbonless the property of becoming hardened when quenched in water.

Diagram III. *Showing the Cooling-Curve of a Sample of 5 Per Cent Aluminum-Steel.*
(by M. Osmond.)



Discussion.

J. E. Stead, Middlesbrough, England: Within the past two years I have made many experiments to ascertain the effect of aluminum upon steel, and also the nature of the chemical reaction which takes place when aluminum is added to steel. My experiments clearly confirm Mr. Hadfield's statement that on adding fluid aluminum to fluid iron there is an increase of temperature. By adding it to cast-iron, free from any oxide of iron whatever, the temperature is very materially increased. By adding it in large excess to white-iron —that is, to iron containing no silicon—a considerable quantity of combined carbon is forced out of its combination with the iron, thus showing that the combination of iron and aluminum cannot be a mere mixture, but must be a true chemical compound. As corroborative evidence, it is noted that when aluminum and iron are added together the specific gravity of the resulting mixture is greater than the mean of the constituents: 77 cubic inches of iron and 23 of aluminum, making a total of 100 cubic inches before melting, combine to form a compound of only 97.3 cubic inches; and the heat which is evolved during the process is a still further indication of true chemical reaction.

The first question with regard to this compound is, how will it behave toward oxidizing agents when mixed with steel. A ferro-aluminum containing about 8 per cent. of aluminum, 3.25 per cent. of silicon, and about 3 per cent. of carbon, was melted in a crucible under an ordinary silicate of iron slag for an hour. It was found, upon analysis, that it had lost 1.8 of the silicon, and 6.41 of the aluminum, showing clearly that aluminum is very much more readily oxidized and removed from its combination with iron than is silicon. It is a powerful reducing agent. In this respect it resembles silicon, but is more powerful. That aluminum should be

found in any steel which has been made by the removal of silicon and carbon from pig-iron by oxidizing agents, *i.e.*, in any Bessemer or open-hearth steel, is therefore utterly impossible, since aluminum is more readily oxidized than silicon.

Another point of very great interest, which has been shown by my experience is, that aluminum in combination with iron or steel has power to reduce carbonic oxide. In this respect, again, it is similar to silicon, but much more powerful. I have heated a 17 per cent. ferro-aluminum for several hours in an atmosphere of carbonic oxide, at a high temperature but not sufficient to cause fusion, and then observed a dark-gray deposit on the outside of the metal. This was found to contain 16 per cent. carbon, 14 per cent. metallic iron, and 69.66 per cent. of free alumina. The carbon could only have come from the carbonic oxide circulating around the piece, and the oxygen of the alumina could only have been derived from the same source. I therefore consider it a safe and logical inference that, if carbonic oxide, and not hydrogen, is the cause of blow-holes in steel, the addition of aluminum would produce sound castings and sound steel. The facts bear this out; every one of our trials has indicated that aluminum can kill the most fiery steel, providing, of course, that it is added in sufficient quantity to combine with all the oxygen which the steel contains. The metal will then be absolutely dead, and will pour like the dead-melted silicon-steel. My experiments prove, that if the aluminum is added as metallic aluminum, and not as a compound, and if the addition is made just before the steel is cast, one-tenth per cent is ample to obtain perfect solidity in the steel. Five-hundredths per cent., in fact, is often sufficient for the purpose. Some investigators regard this proportion as inadequate, but I am inclined to think that, in their experiments, they may have underrated the extreme rapidity with which aluminum is oxidized, and may, therefore, have allowed the aluminum-steel to remain too long in the furnace or crucible before tapping it.

Now, if aluminum will remove such detrimental elements as oxygen and carbonic oxide from steel, it ought to improve the quality of the metal. In my trials with crucible-steel, I think this improvement was clearly demonstrated; a large crucible of steel was melted in the ordinary way, and then one-half of it was teemed out into an ingot-mould while aluminum was added to the other half, and that also was cast. The steel produced was tested in every way I could imagine—for springing quality, for edge or cutting-properties, for power to withstand sudden shock, for resistance to

corrosion, and, in fact, for all the useful properties I could think of. I found that in tool-steel, containing about 1 per cent. of carbon, the elongation or ductility was most materially increased by the aluminum; the tensile strength was reduced; the edge or cutting-properties remained substantially the same. In order to ascertain the relative value of the steels for springing-purposes, they were very carefully hardened and tested, side by side, in the form of wire. The aluminum appeared to make the steel more elastic when hardened. The power to resist sudden shock was also found to be increased by the addition of aluminum, and the same was true of its effect on the torsional resistance; a given length of aluminum-steel wire stood very many more twists than the same steel without aluminum. As for corrodibility, my results have proved very clearly that aluminum, although itself not readily corroded in the air, tends to increase the corrosion of steel when alloyed with it in small quantities. With regard to electric conductivity, I have found that, although aluminum itself is a good conductor, minute quantities of it in steel increase the electrical resistance; for example, of two steels having exactly the same composition excepting that one-tenth per cent. of aluminum was added to one, the one with the addition showed in successive tests an increased resistance of 12 per cent., 6.5 per cent., and 6.8 per cent.

A question that occurred to me in reading Mr. Hadfield's admirable paper was whether the comparatively poor results he obtained from using aluminum in steel-castings could not be explained by the very superior quality of castings which he almost invariably produces in the ordinary way. There are many founders however, who do not make such good castings as those of Mr. Hadfield, and for them the use of aluminum would appear more beneficial.

The important point is: Does the aluminum make steel more fluid? To answer this question a number of experiments were made, of which the following is a type. A piece of flat-iron $\frac{1}{4}$-inch by 2 inches was rolled up into a spiral form, and used as a pattern from which two moulds were made. These were placed side by side, and two steels from the same source, but one containing aluminum and the other not, were poured at the same time, one into the center of each mould. The soft steel which had the addition of aluminum usually flowed to a considerably greater distance in the mould than the non-aluminous steel; but on the other hand, whenever the metal was naturally sound and free from blow-holes (free from oxide of iron or carbonic oxide), such as dead-melted high-silicon steel, that which

contained no aluminum flowed the greater distance. Again, taking two pieces of steel, one containing aluminum and the other not, and gradually heating them in a crucible, it was found by careful observation that the steel containing no aluminum began to melt first. How is it that the non-aluminous steel (if naturally sound) flows the longer distance and melts first, but that a naturally fiery steel flows farther with than without the addition of aluminum, even though the latter does not lower its melting-point? In my opinion it is because the fiery steel, without the aluminum, begins to froth almost as soon as it enters the mould, and as there is no body of metal and accompanying heat present, it sets rapidly, and in the case of most of the non-aluminous steels tested in the spiral moulds, it prevented the flow of the still fluid metal which was behind it. On the other hand aluminum-steel, being perfectly quiet, and as I have said, usually showing the longer flow, was in every such instance found to be sound to the very end or point where it had stopped. I therefore account for the apparently greater fluidity of aluminum-steel by its greater continuity or body, which allows the mass of metal to pour steadily into the mould and flow to the very end of it and fill it completely.

The same thing takes place in the Bessemer ladle; it is my experience that the addition of aluminum to a ladle of steel tends, if anything, to raise the melting point, but the chemical action of the aluminum in eliminating the gases results in a dead quiet metal which pours readily and lies solid up to the top of the moulds.

My friends in Sheffield have been very kind in making several confirmatory experiments for me in this direction. Two crucibles were placed in a fire side by side, each with a charge of boiler-scrap containing only about $\frac{1}{10}$ per cent. of carbon. Into one aluminum was introduced, but not into the other. If anything, the crucible containing the aluminum was the last to melt—rather differing from the trials which have been described by other experimenters. The aluminum was added as a cold alloy, a short time before the steel was completely melted. This naturally checked the melting just as any piece of cold metal added to a steel crucible would do; in both cases however the steel became perfectly fluid and both were poured into ingot-moulds; one was honey-combed and the other was perfectly sound. The trials were all made with aluminum made by the Aluminum Company of London and Birmingham.

E. P. Martin, Dowlais, South Wales: Mr. Hadfield's paper mentions "that these special steels or alloys, such as silicon, manga-

nese, or chromium steel, have slight, if any, welding-properties." I should be glad if Mr. Hadfield would add any information which would enable us to increase the welding-properties of steel. I have always thought—but perhaps mistakenly—that the presence of manganese assisted considerably in making the ordinary Bessemer or Siemens steel weldable.

P. C. Gilchrist, London, England: Two or three years ago I made some experiments with alloys of iron and aluminum, manganese and carbon. To commence with the purest iron I could get, I took as the material to which the aluminum was to be added the iron got in blowing basic metal before the ferro was added. I was indebted to Mr. Seaman, of Sheffield, for melting that pure iron and pure aluminum to make the various alloys of iron and aluminum alone; of iron, aluminum, and of manganese; and of iron, aluminum, and carbon. I started out to see if aluminum would replace manganese entirely, that is to say, whether it would take away the red-shortness of the iron at the end of the Bessemer operation or at the end of the Siemens operation, and I found that it would not. Iron and aluminum alone would not work; it was necessary to have a small amount of manganese. The presence of aluminum would reduce the amount of manganese that was required, but it would not replace it entirely. That is all I am prepared to say from memory. Other points I can bring out in notes that I will send to the Secretary for publication.*

Mr. Hadfield: I am glad to know that Mr. Stead agrees with me as to the material described in the paper being a compound, and not an alloy. Mr. Stead considers aluminum to act more powerfully than silicon. According to a pamphlet published by the Alliance Aluminum Company, its statement seems to prove the contrary, viz., that silicon is really the more energetic in absorbing gases, which is the principal point to which a steel-founder looks.

I cannot help thinking that if it were possible to obtain pure sili-

* Mr. Gilchrist's notes, subsequently sent in for publication, are appended:

A considerable number of experiments were made in order to test whether and to what extent manganese could be replaced in ingot-iron by aluminum without fear of red-shortness.

The aluminum was added to blown metal from the basic converter.

As a result of the experiments, it was shown clearly that aluminum has no influence in removing or preventing red-shortness. Its action on the mechanical properties seems to be to increase slightly the ultimate tensile strength, at the same time increasing the elongation.

Samples prepared with .2 per cent., .5 per cent., and .7 per cent. of aluminum (the

con, or say an alloy of 98 per cent. purity, we would find it quite as efficacious as aluminum, but, of course, such an alloy is practically unobtainable at present. If some of the makers of aluminum would try also to make pure silicon, seeing that silica, the base with which they would start in the manufacture of such a product, is so much cheaper than alumina, it might be worth while to investigate the point.

Mr. Stead also considers as small an amount as 0.1 per cent. of aluminum quite sufficient to produce soundness. Mr. Keep, of Detroit, Mich., however, who has made many experiments on the effect of aluminum upon both cast-iron and steel, considers such a small proportion quite inadequate, and I feel rather inclined to agree with him. Mr. Keep says that as much as 0.75 per cent. would be

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manganese being .10 per cent., .19 per cent., and .18 per cent. respectively), all refused to roll. A sample prepared with 1.12 per cent. of aluminum and .19 per cent. of manganese rolled well; but the blown metal to which no aluminum was added rolled equally well, so that the freedom from red-shortness cannot in this case be attributed to the presence of aluminum.

To show the influence of a comparatively high percentage of aluminum on soft steel, the following series of tests may be cited:

Sample.	Analysis.						Mechanical Tests.			
	Si.	P.	Mn.	C.	S.	Al.	Ultimate Stress per Sq. In.	Elongation in 8 inches.	Reduction in Area.	Remarks.
A. ...	...	.075	.198	.05	.04	1.12	Tons. 31.0 28.7	Per cent. 24.3 24.8	Per cent. 67.4 69.3	As rolled, annealed and quenched.
B. Blown metal to which Al was added in experiment.	...	.040	.140	.05	.05	nil.	26.3 27.4	20.0 18.6	69.9 65.1	As rolled, annealed and quenched.
C. ...	...	.061	.196	.05	.033	2.75	35.0 32.1	12.9 19.8	30.7 37.7	As rolled, annealed and quenched.
D. Blown metal to which Al was added in experiment.	...	.061	.190	.06	.033	nil.	27.1 32.1	19.9 16.4	70.2 52.5	As rolled, annealed and quenched.

requisite to secure uniform soundness, and if so, even with aluminum at the price of 6*s.* ($1.50) per pound, this would mean an increase in the cost of £4 (nearly $20) per ton, not a satisfactory item in a steel-founder's material-bill.

In regard to Mr. Stead's observation that fluidity is increased by the addition of aluminum, I think his own explanation should especially be borne in mind, viz., that his experiments were with reference to *frothy* steel, that is, steel which, to start with, is unsound. A good steel-founder would not use such steel to put into his castings at all, apart from the question of using or not using aluminum; so that Mr. Stead is evidently looking at it from quite a different point of view to that of a steel-founder.

As to Mr. Martin's inquiry, it is certainly my opinion that the greater the percentage of foreign elements in steel, the more difficult it becomes to weld. My own observations indicate that both silicon and manganese hinder good welding-results, that is, when the manganese is in excess of 1 per cent. This is especially proved in the manganese-steel manufactured by our firm (Hadfield's Steel Foundry Company, Sheffield), containing about 9 to 14 per cent. of manganese; it is practically unweldable. Even the electric-welding system has failed to make a satisfactory weld, though Professor Thomson informs me that he hopes to have better success with his later machines; but in any case much difficulty will be experienced.

Speaking of electric welding, there must be a great future for it, because it heats the material in quite a different manner from the present mode, and for this reason, if any system can give successful results in welding these special alloys or compounds of iron with manganese, silicon, chromium, nickel, etc., Professor Thomson's is the most likely. But yet, I think that, as regards steel, the more nearly the material approaches pure iron, the better the weld, and the more foreign elements that are present, the more difficult will it be to secure good results. My own attempts to weld aluminum-steel have been unsuccessful, although the samples were given to a clever blacksmith, who did his utmost by several trials to attain the desired result.

Discussion Continued by Correspondence.

Mr. Thomas Andrews, F.R.S.: I have seen with great interest the admirable paper by Mr. R. A. Hadfield on aluminum-steel. There can be no doubt that aluminum will play a very important part in the metallurgy of steel in the near future. I am at present

engaged in studying the influence of aluminum on the passivity of steel, in course of my research on the passivity of iron and steel for the Royal Society, and I anticipate some interesting results, though at the present juncture it would be premature to give them. The high percentage aluminum-steel on which I am working has so far shown somewhat brittle properties, and in course of rolling the ingots into small bars many of the latter have been broken. It has so far also been found very difficult to draw the small rolled rods cold through a wortle. Although my research is yet incomplete, I may perhaps say that the aluminum-steel, as regards its passivity, has hitherto behaved somewhat differently in comparison with the behavior of ordinary cast-steel, Siemens, Bessemer, manganese-, nickel-, tungsten- and other steels; but on this I hope to have more to say when my investigation is complete. I congratulate Mr. Hadfield on the excellent and valuable results he has already obtained in connection with the study of aluminum-steel, and heartily wish him every success in his future investigations.

Mr. F. Osmond, Paris, France: Mr. R. A. Hadfield has studied the action of aluminum in the metallurgy of iron in the admirable manner that characterizes all his work. His researches enable industrial questions to be satisfactorily solved by the aid of the exact processes of the physical sciences. There consequently remains but little to say after him; and, as the author has shown in his paper, I am pleased to be completely in accord with him on all points concerning which I have been able to form an opinion of my own.

From an industrial point of view, it appears to be well proved that aluminum is, like silicon, a reducer of oxide of iron. Anhydrous silica, obtained from amorphous silicon, is formed with a liberation of 219.24 calories, according to Troost and Hautefeuille, and aluminum hydrate, $Al_2O_3 + 3H_2O$, is formed with a liberation of 391.6 calories, according to Thomsen. Hence it is deduced that if the heat of hydration of the alumina (not known, and probably very insignificant) is neglected, 16 units of oxygen liberate 109.6 calories on combination with silicon, and 130.5 calories on combination with aluminum. Thus, according to the indications of thermal chemistry, aluminum is a still more energetic reducer than silicon, not for equal weights, but in the presence of the same quantity of oxygen. In other words, aluminum would be burnt before silicon in intermolecular combustions, so far as it is allowable to state this without more complete data. In such case there would be a slight advantage in favor of aluminum, an advantage, however, that the

present price of aluminum, in spite of its enormous fall during recent years, would render very expensive.

It can merely be asserted that aluminum, when met with in steel in small quantities, furnished without expense by the hearth, appears to be indicative of good quality, not on account of the negligible influence that it exercises on the properties of iron, but as evidence of a well-conducted charge on the open-hearth. If, in fact, an element so easily oxidizable has been able to retain the metallic state, it is probable that the bath itself has experienced the minimum amount of oxidation, and has yielded sound ingots.

From a scientific point of view, the samples obtained by Mr. Hadfield might be, and have already been, the object of interesting researches. If the curve of cooling of the alloy containing 5 per cent. of aluminum is examined, and compared with that of an extra soft steel containing nearly the same percentage of carbon,* it will be seen that the point d_{13} is wanting. The aluminum thus maintains the iron, for the most part, in the α-state from the ordinary temperature to the melting-point. This explains why annealing effects no notable change in the structure, as Mr. Hadfield has shown, and also why the structure, at the same temperature of annealing, differs completely from that of ordinary mild steel.

For the same reason, if a sufficient quantity of aluminum is added to a bath of molten steel which is in the β-state, the β-iron will be transformed into the α-modification—a change which liberates a considerable quantity of heat. In this way I should explain the phenomenon of coruscation observed by Mr. Hadfield.

If my interpretation of the phenomena is accurate in all points, steel with 5 per cent. of aluminum should present another curious peculiarity—that of remaining magnetic at all temperatures. This is, however, a theoretical deduction, which consequently requires to be proved by experiment. I should be pleased if Professor Barrett or Dr. Hopkinson, whose former researches have so important a bearing upon scientific metallurgy, would direct their attention to this question.

Mr. Hadfield's paper also brings into notice the question of the determination of aluminum in iron. This presents, at least in so far as it deals with very small quantities of aluminum, a difficult problem in analytical chemistry; and I have good reasons for believing that in many cases the aluminum that has been found in certain steels has been derived from the glass vessels used, or from

* *Journal of The Iron and Steel Institute*, 1890, No. I. p. 43, Plate III.

impurities in the reagents. A general paper on this subject would be of great interest. Personally, I have had occasion only to occupy myself with the matter incidentally. The method that I adopt is as follows: The steel cuttings are first heated to redness in a current of hydrochloric acid gas, as in the case of the classic method proposed by Boussingault for the determination of silicon. When the steel contains aluminum, this element remains with the silicon in the white residue, in which a certain quantity of phosphoric acid also remains. The weight of the residue gives the total of the three substances. The silicon alone is then determined by Drown's method, and the phosphoric acid is determined in the residue. The methods of Boussingault and Drown are sufficiently accurate in the hands of a good chemist to enable the alumina to be determined by difference. Further, the presence of phosphoric acid guarantees that of aluminum, and it is always possible as a check to separate by the usual methods the silica and alumina in the residue obtained by Boussingault's method. All the iron having been eliminated, this separation demands only a very small quantity of reagents, and can be made with precision, provided that care is taken to employ exclusively platinum vessels and well-tested reagents.

Mr. John H. J. Daggar: In taking part in this discussion, let me first thank Mr. Hadfield for his very interesting sketch of the early history and properties of aluminum and the present-day processes for its production, and then let me enter a protest against any attempt to alter its name. All metals discovered in modern times have been given Latinized names terminating in *-ium*, the non-metals or metalloids, in *-ion*, and in accordance with this accepted nomenclature, sili*cium* has been changed to silicon, selen*ium* to selen*ion*, both these elements being distinctly non-metallic. The objection to "alumina" will hardly stand unless one is prepared also to accept also sod*ia*, potass*ia*. To give the distinctly metalloidal termination *-on* to an element with the most marked characteristics of the metal, totally differentiated from the metalloid silicon, would only introduce needless confusion into chemical nomenclature.

According to Gautier, ferro-silicon has been used for some years past in French foundries for the prevention of blow-holes in steel castings, and much has been done to extend its use in England by the work and experience in this direction of Mr. Turner, Mr. Hadfield, and others; and there is no doubt at all that very great benefit has been derived from the judicious use of aluminum, though of course steel manufacturers are naturally very reticent as to the re-

sults actually obtained; but the elaborate and careful researches of Keep and others have thrown very much light on the exact influence of aluminum on cast-iron and the limits of its use, and repeated experiments have shown that it is equally valuable in steel-manufacture. In an experiment brought under my own notice, with basic Siemens-Martin steel, the tensile strength was increased from 98,560 pounds to 112,000 pounds by the addition of 0.2 per cent. aluminum.

Chemical Action.—From my own experience with alloys of aluminum and iron, I have scarcely any doubt but that aluminum combines chemically with the iron; the addition of aluminum to molten iron in the crucible being accompanied by phenomena closely resembling those occurring in the formation of the copper alloy.

Fluidity.—The extra heat evolved during the union of the two metals is, however, in my opinion, insufficient to account for the increase of fluidity followed by rapid setting. The experiment by Mr. Osmond, quoted by Mr. Hadfield, disproves the theory as to the lowering of the melting-point. The great affinity of aluminum for oxygen renders it exceedingly probable that Ledebur's theory is correct; and in this connection aluminum would be far superior to silicon. One word as to the affinity of silicon and aluminum for oxygen. The point bearing upon this question is not the weight of oxygen which will combine with these two elements respectively, but the readiness with which oxidation will occur.

Silicon which has been strongly ignited in a covered crucible does not oxidize in the oxyhydrogen flame, or on fusion with nitre, though amorphous silicon readily does so. Aluminum, on the other hand, absorbs oxygen readily when heated above its melting-point in contact with bodies containing oxygen, and will reduce SiO_2, forming Al_2O_3, and liberating Si. $4Al + 3Sio_2 = 2 Al_2O_3 + 3Si$.

The effect of aluminum in diminishing hardness is doubtful. An ingot made from pure aluminum and hematite-iron, containing 15 per cent. to 16 per cent. aluminum, cast in the chill, was only marked by the file with very great difficulty, great force being required to chip it; the fracture was close, fine, and crystalline. Another sample of aluminum-steel, containing 7 per cent. aluminum with 1 to 2 per cent. manganese, will just scratch glass.

The soft nature of aluminum itself does not affect this question; for its addition to copper gives alloys which are glass-hard. Its influence on iron in this direction, used as it is in small propor-

tions, will depend entirely upon the change it produces in the state of the carbon present.

Magnetism.—Above 5 per cent. of aluminum rapidly destroys the magnetic properties of iron, and some time ago experiments were carried out to determine whether this phenomenon would furnish a clue to the percentage of aluminum present; but the results then obtained were not precise enough for quantitative indications.

The action of aluminum resembles that of silicon, and is probably as follows: (*a*) Its action on the combined carbon. (*b*) Its action on dissolved oxides of iron. (*c*) Decomposition of gaseous and solid compounds of carbon and hydrogen entangled in the molten metal, and possibly also of brittle combinations of carbon and iron.

The action of aluminum is more certain to produce sound ingots than the use of silicon, as, indeed, one would expect, aluminum forming a true alloy with the iron, an alloy being taken to mean the union of two metals. Silicon *alloy* is a misnomer, though often used for convenience; silicon is not a metal.

To conclude, Mr. Hadfield's paper has undoubtedly proved the great value of aluminum to the steel-founder, and contributed much to the somewhat meager knowledge of aluminum-alloys. In his comparison of this element with its rival silicon, his only objection to its use has been the additional cost. From my own knowledge of the use of aluminum, I can only repeat that the more certain results obtained with smaller amounts of aluminum will more than compensate for the extra cost in the finer quality of castings produced.

M. Alexandre Pourcel, Saltburn, England: Although Mr. Hadfield's interesting paper on aluminum-steel does not solve all the obscure points relating to this subject, it goes a long way towards explaining the action of aluminum in steel. Professor Ledebur, of Freiberg, was the first to contest that aluminum lowered the melting-point of steel. The fluidity that is imparted to steel by the addition of only $\frac{1}{1000}$ of aluminum is due, according to Professor Ledebur, to the disappearance of every trace of ferric oxide and slag from the intermolecular spaces, under the energetic influence of aluminum as a reducing element. In my opinion, however, the cleanliness of the bath is not the only cause of its fluidity, provided that manganese silicide, possibly also clean without being fluid, has been added to the bath. The researches of Mr. Osmond prove that the melting-point of a steel with 5.60 per cent. of aluminum is at most 25° C. lower than that of an extra mild steel without aluminum. Professor Lede-

bur's opinion is thus confirmed by experiment—aluminum does not lower the melting-point of steel.

Besides, however, the action as a reducing-agent, which experiment proves aluminum to have, there must also be recognized its action as a calorific agent. In this connection Mr. Hadfield does not deal with the question in its true light. According to Berthollet, he points out that the change of the unit of aluminum into Al_2O_3 liberates 7900 calories, and that of the unit of silicon into SiO_2 liberates only 7800 calories. The difference is not very considerable, and as $\frac{2}{1000}$ or more of silicon is usually added, the question arises, why a more fluid metal is not obtained with silico-spiegeleisen than with $\frac{1}{1000}$ of aluminum? The confusion made by Mr. Hadfield, and by many others with him, is to attribute the same calorific power to silicon in the state of silico-spiegeleisen or manganese silicide—in fact, in an alloyed state, as in the state of purity. This is an error, a grave one, which I pointed out ten years ago, and which I shall continue to emphasize until the honor is done me to discuss seriously, after reflection, the opinions I set forth in my paper on the applications of thermo-chemistry to metallurgical reactions. 1. The calorific power of silicon in the state of alloy cannot attain half of that of pure silicon according to the richness of the alloy. 2. As the alloys of silicon are usually employed in the solid state, account must be kept of the heat absorbed by the bath to melt them, or, in other words, of its calorific capacity. 3. Lastly, the oxidation of the alloy with silicon gives rise to a slag, the calorific capacity of which, determined by the researches of Professor Gruner, Sir Lowthian Bell, and other investigators, greatly exceeds that of the metal. Indeed, it may be asked whether the total of these intermolecular reactions has not contributed to lower rather than to raise the temperature of the bath of metal.

With aluminum, on the other hand, and especially with commercially pure aluminum, containing 98 per cent. of the metal, which is employed in most foundries, and which melts between 600° and 700° C., it may be admitted that all the heat liberated by its change into alumina, or, in most cases, the greater part of this heat, is added to that of the bath of steel. There is a re-heating of the steel. By utilizing the heat added to a bath of mild steel by the addition of $\frac{1}{1000}$ of pure aluminum, I have been enabled to produce recarburized steel containing chromium, nickel, or tungsten, in which, thanks to the fluidity of the metal, perfect homogeneity is preserved.

Mr. J. H. Darby, Brymbo Iron Works, North Wales: As far

as my experience goes, aluminum is useful to those who make castings for themselves, as by its use metal of almost any temper can be used, and the castings made therefrom in the majority of cases do not require annealing, and with very little care in teeming are quite sound. No doubt excellent castings can be made by firms who work day after day on metal suitable for castings, but where a charge is only occasionally required, a small addition of the metal aluminum, about 1 per cent., enables you to get perfectly sound castings. The metal, however, "pipes" badly, and requires a large head to feed the castings from. The great disposition to "pipe" in metal to which aluminum has been added will, in my opinion, always prevent its use in the manufacture of solid ingots. I have seen them with a considerable hole caused by contraction, which continued for three-fourths of the length of the ingot from the top.

Aluminum is useful by its absence, *i.e.*, its avidity for oxygen contained in the steel at once converts it into alumina, which I have frequently picked out of any small irregularities that may exist on the surface of ingots when cool where aluminum has been added to the steel. The ingots are so marked with these thin patches of alumina that it is quite distinctive, and it is not to be confounded with wash off the mould. In fact, there is just as much on ingots from perfectly clean moulds.

My experience goes to prove that there is practically only one kind of gas absorbed or occluded by molten steel, and that is carbonic oxide. Remove the oxygen completely, as is possible with aluminum, and the metal, Bessemer or open-hearth, acid or basic, is perfectly quiet and absolutely solid, whether hard or dead-soft.

A peculiarity I have noticed, and am not quite able to account for, is that although molten steel may be completely deprived of oxide by aluminum, still it will fall to pieces, and will not "work," unless it contains a small percentage of manganese. Possibly in the latter case the small percentage of sulphur always present causes red-shortness.

Mr. W. S. Sample, of the Metal Reduction Syndicate: Since our works have begun the manufacture of aluminum, we have supplied a great many steel-makers, and by their continuing to use the metal, we are led to believe that they are getting satisfactory results by its use in steel castings. They have not been merely trial-orders for experiment, as orders have been duplicated and increased. The benefit of using aluminum is not confined to steel for castings alone. A careful series of experiments as to the effect of aluminum in mild

open-hearth steel has shown that the addition of about half a pound of pure aluminum to a ton of steel has a markedly quieting effect on the steel, produces sound tops to the ingots, and consequently less percentage of scrap in rolling. If too much aluminum is added, it will cause piping. Professor John W. Langley, of the Crescent Steel Works, Pittsburgh, Pa., allowed me to say on his behalf that the saving by use of aluminum is such that no steel-works can afford to do without it, at the low price at which it is now sold. Of course, experience is a great factor in determining how the aluminum shall be used. The aluminum should be added to the ladle as the metal is being tapped into it. Should the steel be very "gusy" or "wild" in the ladle, up to two or three pounds should be added per ton of steel in order to quiet the metal, and for such "wild" heats, aluminum should be in hand at every furnace. The two extremes are too much and too little, and the best results have shown, as above stated, that half a pound of aluminum per ton of steel will save far more than the cost of the aluminum by the decrease in scrap in rolling. We do not want to make steel-makers believe that aluminum is a panacea for all the ills, but when so many users are continuing and report benefits, it is very good testimony.

Prof. T. Turner, of Mason College, Birmingham, England: I have read Mr. Hadfield's paper with great interest, and feel that this last contribution greatly increases the obligation which those who are interested in the connection between the chemical and mechanical properties of iron and steel already owe to the author. Probably, however, few chemists would at present approve of the suggested alteration in the name of the metal aluminum, even if it were practicable, for though the name now in use is not euphonious, the suggested change would class aluminum with carbon, boron and silicon, all of which are decidedly non-metallic in character. The American termination of *-um* is perhaps, on the whole, the best, as the characteristic metallic properties of aluminum more resemble those of ferrum, cuprum, aurum and argentum.

The close resemblance which Mr. Hadfield has shown to exist between the effect of silicon and aluminum on the properties of iron and steel is perhaps the most important outcome of his researches, and should do much to moderate the sanguine anticipations of some inventors who claim so much benefit from the use of aluminum. Mr. Hadfield sent me some samples of steel, containing various proportions of aluminum, to be tested for relative hardness and the

results of my tests fully corroborate the statements made in this paper, that little or no real hardness is conferred by aluminum.

A short time before Mr. Hadfield's paper was written, I was consulted in connection with a recently-introduced process for producing aluminum-steel, and a number of samples were submitted to me for examination. Some of these were beautifully polished and exhibited a most brilliant luster, while most of them were unusually hard. It was believed that the aluminum, which it was stated these samples contained, was the cause of the unusual luster and hardness. I ventured at the time to cast very serious doubts upon this matter, and to suggest that the properties above mentioned were such as could be accounted for by the known composition and method of production of the steel quite apart from the presence of aluminum. I am now indebted to Mr. Hadfield for the more complete and exact knowledge furnished in his paper, which confirms and amplifies the views on this matter which I previously expressed. So far as our knowledge goes at present, it appears to me that the greatest opening for the application of aluminum in the iron-trade is for special purposes in the foundry for producing gray iron, and occasionally in steel for obtaining soundness. But it is evident that before aluminum can be of much use to the iron manufacturer, the cost of production must be even further considerably reduced.

Professor J. O. Arnold, Sheffield, England: Mr. Hadfield's research must be regarded as a most valuable addition to the literature of what may be termed the pure metallurgy of iron. In it he has shown to perfection the mechanical effects produced by the addition of varying percentages of aluminum to wrought-iron or very mild steel. The results of his experiments, however interesting from an academical point of view, have little bearing upon the commercial production of crucible-steel and steel castings, because the difficulty of melting and teeming so mild a material, and the consequent wear and tear of refractory material, place it out of court as an everyday product, capable of being manufactured in large quantities.

The present writer, assisted by the demonstrators and students of the Sheffield Technical School, has been engaged for some time upon a careful investigation of the effects of aluminum applied to the manufacture of tool-steel and steel castings as produced in ordinary practice. The results obtained are extremely remarkable and interesting, and will, he believes inaugurate a new departure in the manufacture of such steels. The detailed results of the research will form the subject of the inaugural lecture of the Sheffield Technical School

Metallurgical Society, but, by permission of the Council of that body, the following preliminary and abridged statement is made to the Iron and Steel Institute. The results of repeated experiments have:

(1.) Fully confirmed Mr. Hadfield's observations, that no general rise of temperature accompanies the addition of aluminum to molten steel, but that, if anything, the metal is rendered less fluid.

(2.) Induced the writer to dissent altogether from Mr. Hadfield's view, that the action of aluminum is an analogue of that of silicon. The writer's experiments show clearly that the effect of even small quantities of aluminum in producing steel free from blow-holes is perhaps the most remarkable phenomenon in the metallurgy of steel. Its action is about twenty times as powerful as that of silicon, and the resultant steel is far superior in ductility and toughness to steel rendered sound by the addition of silicon; when due precautions are taken to avoid the danger attending its use mentioned under heading 4. (*Vide infra*).

(3.) Proved that the action of aluminum is almost certainly chemical. The writer has not yet completed this branch of the research, but the results so far obtained are as follows:

That blow-holes proper are due to the non-oxidizing occluded gases, hydrogen, carbonic oxide, and nitrogen, separating (under a pressure of many atmospheres) from the steel just prior to solidification. It has been proved by repeated experiments, made in a specially-constructed tube-furnace, that *metallic aluminum readily decomposes carbonic oxide below steel-melting heat with the formation of alumina and free carbon;* in other words, the most powerful reducing gas known to metallurgists acts, in the case of aluminum, as an oxidizing agent. To confirm this reaction, under the exact conditions of steel-manufacture, the writer blew 40 gallons of pure carbonic oxide through a crucible of molten steel containing aluminum, and obtained an increase of 35 per cent. on the carbon present before passing the gas.

(4.) Indicated that the addition of 0.05 to 0.1 per cent. of aluminum causes mild crucible steel (carbon, 0.5 per cent.) to "pipe" freely, the cubic capacity of such pipe representing the contraction of the liquid metal occurring between the molten and pasty conditions. In the case of "blown" steel no such piping is observed, because the separation of gas-bubbles more than counterbalances the contraction. In both ingots and castings, piping has to be guarded

against by special precautions, which will be dealt with in the main communication.

The writer has here to record the important result of the first practical application of Sorby's method of microscopically examining steel. It will be obvious that steel from which the occluded gases are removed, or do not separate, will be very susceptible to linear contraction. The disastrous effect of this, in the case of castings of certain shapes, firmly imbedded in rigid "compo," is well known to steel-founders from the fact that such castings are sometimes "pulled" in two.

A careful microscopical examination of the phenomenon of "pulling" has revealed the alarming fact that incipient pulling may exist which is quite invisible to the eye even when aided by a powerful hand-lens.

Polished test-pieces were turned from such castings which appeared absolutely free from flaws, but on testing they stood less than half the maximum stress usually associated with their chemical compositions; and the fractures showed dark patches, some of them a quarter of an inch in diameter, where the air had penetrated and oxidized the crystals through microscopic fissures $\frac{1}{5000}$ to $\frac{1}{10000}$ of an inch in breadth.

All steel which has been unduly strained in tension whilst hot is permeated with these fissures, which may or may not be in communication with the atmosphere, but which in any case weaken the steel to a great extent. How far these conditions may obtain in *forgings* containing parts of highly unequal mass is a matter for further research. Another interesting point is whether or not the remarkable increase in toughness and capability of resistance to shocks obtained in annealed over unannealed castings is due in some measure to the closing up and welding of these contraction-fissures.

(5.) Shown that the addition of 0.05 of aluminum to high-class crucible-steel ingots, cast in such a manner as to avoid the evil of "piping," will bring about the following important results:

(*a.*) Dispense with the addition of manganese, and so produce the ideal iron- and carbon-steel.

(*b.*) Do away with the necessity of "wash-welding."

(*c.*) Save 25 per cent. of the time and fuel respectively occupied and used during the operation of "drilling," together with the resultant saving of wear and tear to crucible and furnace. A pure steel, lower in silicon and manganese than that at present produced, will thus be obtained, with a saving of some shillings per cwt.

Explanation of Photographs, Showing Complete Elimination of Blow-Holes.

Fig. 5.—*Transverse Fracture of Ingots.*

A. Melted from LSS converted bar, melted with 0.6 per cent. and 80 per cent. ferro-manganese. Ingot supposed to be "killed." No piping precautions. Teemed "fiery." Composition: Cc. 0.99, Si 0.14, Mn 0.39, S 0.02, P 0.02.

B. Same as A, with the addition of 0.1 per cent. aluminum five minutes before

Fig. 5.

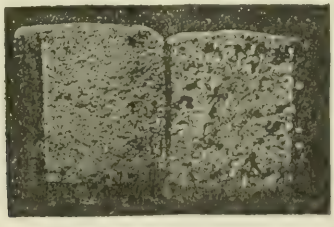
A B

pulling out. No attempt made to kill the steel, and no precautions taken to prevent "piping." Teemed quite "still." Composition: Cc. 0.95, Si 0.10, Mn 0.42, S 0·02, P 0·02. Aluminum not estimated.

Fig. 6.—*Transverse Fracture of Ingots.*

A. Melted from LSS converted bar without addition of manganese, without killing, but with addition of 0.1 per cent. aluminum just before teeming, and precautions taken to prevent piping into body of ingot by means of a core or

Fig. 6.



A B

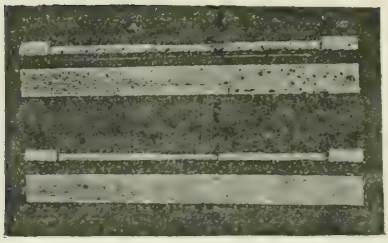
"dozzler." Teemed quite "still." Composition: Cc. 0.97, Si 0.06, Mn 0.05, S 0.02, P 0.02. Aluminum not estimated.

B. Same as A, but no aluminum added, and no piping precautions. Teemed "fiery." Composition: Cc. 1.04, Si 0.08, Mn 0.06, S 0.03, P 0.02.

Fig. 7.—*Tensile and Bending Test-Pieces From Dry "Compo" Castings.*

A. Melted from Bessemer spring-scrap only. Composition: Cc. 0.62, Si 0 27, Mn. 0.46, S 0.11, P 0.08.

FIG. 7.



B. Exactly same as A, but 0.1 per cent. aluminum added five minutes before casting. Composition: Cc. 0.64, Si 0.29, Mn 0.62, S 0.10, P 0.08, Al 0.04.

The writer hopes, upon some other occasion, to lay before the Institute the results of the application of aluminum to open-hearth steel, for which he anticipates a great future.

MR. LE CHATELIER: Without having made any experiments with the alloys of iron and aluminum, I am of the opinion that in respect to lowering the melting-point of iron by the addition of aluminum, some interesting deductions may be made from the general laws of chemistry. Metallic alloys behave exactly like mixtures of salts *per se*, or mixtures of salts with water, that is to say, solutions. By adding to a salt a small quantity of another salt or of water—by adding, for example, a small quantity of salt to ice—the melting-point of the ice or of the salt that is in excess, is *always* lowered. For mixtures of two bodies in variable proportion, for example, potassium nitrate and sodium nitrate, the curve of the melting-points starting from the pure salts (about 350°), gradually descends and for a certain intermediate value reaches a minimum (about 200°). In the case where the two bodies can form a stable compound of definite proportions in the molten mixture, the form of the curve may be more complicated, a cusp in the curve being formed at the point where the combination occurs. In all cases, however, the melting-point of each of the two bodies is lowered by the addition of a very small quantity of the other, whatever may be its melting-point. Thus, potassium sulphate, which melts at 1045°, lowers the melting point of ice.

It is exactly the same in the case of metals. The addition of a small quantity of one metal to another always lowers the melting-point of the latter, the mixture of lowest melting-point being the alloy without liquation. The curve of fusion of the alloys of lead and tin is analogous to that of the mixtures of alkaline nitrates.

The alloys of platinum and tin give, on the other hand, a minimum very close to the melting-point of tin, but always lower than it. It is thus certain, without experimental evidence, that the addition of small quantities of aluminum to steel lowers its melting-point. It is merely necessary to know the degree to which it is lowered. This may be calculated in the following way:

The lowering of the melting-point $\triangle t$ is given by the formula

$$\triangle t = \mathrm{KT}n,$$

where T is the absolute temperature of fusion;

$$\mathrm{T} = t + 273,$$

or 1773 in the case of iron; n the number of molecules of the metal in small quantity that is added to 100 molecules of the metal present in greater quantity. The molecular weight of aluminum is 27.5, and that of iron is 28. For the alloy with 5 per cent. of aluminum, the value of n is therefore equal to 5. Lastly, the value of K deduced from the experiments of Messrs Heycock and Neville* would be $\frac{5}{1000}$. Whence it may be deduced that the lowering of the melting-point,

$$\triangle t = \tfrac{5}{1000} \times 1773 \times 5 = 43^\circ.$$

This calculated result does not differ from that obtained experimentally by Mr. Osmond, more than to a degree compatible with experimental error.

Mr. SPENCER, of Newburn Steelworks, Newcastle-on-Tyne, England: I beg to enclose results of experiments on the use of aluminum in the manufacture of steel castings, with the chemical and mechanical tests of the same.

In conducting the experiment, I had chiefly in view the effect of aluminum on the prevention of blow-holes, the mechanical test being subordinate though important. I therefore used in the melting-mixtures of steel such as come to hand, as would be done in a usual way,.and in the production of ordinary steel, as distinguished from "steel castings," and the methods used to infuse soundness into them.

* *Journal of the Chemical Society*, vol. lvii., pp. 376, 656.

The result has been satisfactory in every instance, so far as soundness and the usual attributes of a good casting are concerned, running fluid and without ebullition into sharp clean castings; the milder mixtures, when tried by the hammer-test, breaking very strong, though unannealed.

The chemical reaction of the aluminum in the crucible in presence of other combinations may be various, but reducing the carbonic oxide and preventing blow-holes is the chief desideratum. In clay pots the aluminum may rest upon the silica reducing it,—the silicon going into the steel, especially in presence of high carbon, which of itself generally increases the silicon contents in the steel made in ordinary crucibles.

The present experiments, however, have not been conducted with the nicety needed to enable a scientific analysis to be made of the various reactions taking place in the different mixtures; but it will be observed, in looking over the analyses, that there is a marked increase of silicon in presence of the higher percentage of aluminum and carbon.

All the low carbons, A 1, 4, 10, 13, were obtained by using Swedish iron alone, as also A 7, 14, 15. Though A 7 with carbon = .28 per cent, has a tensile strength of 25.58 tons per square inch, as against A 15 with carbon = .29 per cent and 23.62 tons, it may be said that aluminum has increased the elastic limit and tenacity by 2 tons; but it will be noticed that silicon is .22 per cent. in the former, and only .074 per cent. in the latter, which may partly account for the difference.

Unfortunately several of the specimens for mechanical test have had slight flaws on the outside or circumference of fracture, vitiating the result for strict comparison, yet the elastic limit remains, to enable an estimate to be formed. The general conclusion, as regards the mechanical tests, is, that though possibly aluminum may increase the tensile strength and elastic limit slightly, yet this is done at the expense of ductility; and in comparison with our usual steel castings, a copy of tests whereof is annexed, I do not find that there is any advantage in its use for increasing the strength and ductility of steel castings, but, on the contrary, I think that, in the presence of high carbon, it is disadvantageous in this respect. It will be found that neither the elastic limit nor the tenacity are in any case more than can be accounted for by the carbon and silicon present; in fact, I have been astonished that they have come out so low.

The carbons of A 6 and 9 are higher than I intended, owing to the melter not having allowed for carbon in the alloy; but as all the analyses as well as the mechanical tests are given, a general comparison can be made.

I am having another mixture made in place of A 15, also caused by error of the melter in mixing, and also one of the same material, but without any aluminum.

The first mixtures, A 1, 2, 3, were cast into castings only, without any mechanical test-pieces being taken.

In A 4 to 9 the steel castings had all been annealed, according to usual practice, before it was thought that unannealed test-pieces would have been desirable as well as annealed.

A 10 to 15 had tests made from both unannealed "as cast" and also from annealed; the effect of the annealing may be interesting to you.

Below are tables of analyses and mechanical tests, together with analysis of the alloy used.

Explanation.

Chemical analysis: Results are given in per cent.
Mechanical test: *Elastic limit* in tons per square inch.
" *Maximum strain* in tons per square inch.
" *Contraction of area* in per cent. of original area of a diameter of .757 inch on the average.
" *Elongation* in per cent. measured on the length of 3 inches.
" *Flaw* in per cent. of area of fracture with indication of position:
" *o.* as on outside or circumference of fracture of test-piece.
" *i.* in inside of fracture.

Chemical Analysis of Aluminum Alloy.

(*June* 14, 1887).

		Per cent.
Iron,		86.69
Carbon combined,	1.01	
" graphite,	1.91	
Total,		2.92
Silicon,		2.40
Manganese,		.31
Aluminum,		6.50
Copper,		1.05
Sulphur,		none
Phosphorus,		.13

Materials Used.				Chemical Composition of Metal Cast.						
No. of Mixture.	Swedish Iron.	Spring Scrap.	Alloy.							
A.	Lbs.	Lbs.	Lbs.	C.	Mn.	Si.	Al.	S.	P.	Cu.
1	56	...	1	.12	traces	.074	.11	.013	.051	...
10	56	...	1	.10	traces	.06	.12	.015	.04	...
13	56	...	1	.15	traces	.093	.12	.013	.037	...
14	56	...	1	.25	.045	.093	.12	.013	.037	...
15	56	...	1	.29	traces	.074	.11	.013	.037	...
4	56	...	2	.15	traces	.084	.19	.013	.038	...
7	56	...	4	.28	traces	.22	.33	.012	.053	...
2	28	28	1	.21	.16	.14	.11	.035	.051	.041
11	28	28	1	.21	.09	.093	.12	.026	.04	...
5	28	28	2	.33	.09	.14	.16	.032	.053	...
8	28	28	4	.49	.16	.23	.31	.023	.049	...
3	...	56	1	.53	.25	.28	.11	.05	.062	...
12	...	56	1	.48	.30	.33	.11	.037	.045	...
6	...	56	2	.65	.25	.25	.18	.036	.054	...
9	...	56	4	.85	.30	.40	.29	.028	.055	...

A 1.

56 lbs. Swedish iron.
1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.12	traces.	.074	.11	.013	.051

Mechanical test not made.
Casting: Fluid; ran sound into small castings.
Remarks: Tough and malleable.

A 2.

28 lbs. Swedish iron.
28 lbs. spring scrap.
1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.	Cu.
Analysis:	.21	.16	.14	.11	.035	.051	.041

Mechanical test not made.
Casting: Fluid; ran into sound small castings.
Remarks: Tough and malleable.

A 3.

56 lbs. spring scrap.
1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.53	.25	.28	.11	.05	.062

Mechanical test not made.
Casting: Fluid; casts sound.
Remarks: Hard and comparatively brittle.

A 4.

56 lbs. Swedish iron.
2 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.15	traces	.084	.19	.013	.038

Tensile:	Mark. Elas. L.	Max. S.	Contr. per ct.	Elong. per ct.
Mechanical test: Annealed.	1159 10.9	21.8	37.17	27
	1160 9.5	21.16	41.97	27

Fractures: 1159 Gray, fibrous.
1160 Gray, crystalline, 12 per cent. of area of fracture shows slag mixed with the steel.
Casting: Fluid.
Remarks: Castings are sound and tough.

A 5.

28 lbs. Swedish iron.
28 lbs. spring scrap.
2 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.33	.09	.14	.16	.032	.053

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.
Mechanical test: Annealed.	1161	10.41	26.28	43.3	32
	1162	10.82	26.08	43.7	29

Fractures: 1161 Dull gray, fibrous.
1162 Do. do.
Casting: Fluid; runs sound.
Remarks: Much more brittle than A 2.

A 6.

56 lbs. spring scrap.
2 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.65	.25	.28	.18	.036	.054

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test: Annealed.	1163	14.38	24.3	4.4	3.5	8–*o*
	1164	14.38	26.28	9.5	9.0	1–*o*

Fractures: Both are granular, dull gray, with slight radiating crystalline flaws on outside.
Casting: Fluid; runs sound.
Remarks: Unannealed castings are brittle.

A 7.

56 lbs. Swedish iron.
4 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.28	traces.	.22	33	.012	.053

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test: Annealed.	1165	12.20	25.56	.19.95	19.0	10–*o*
	1166	11.31	25.58	19.95	185.	5–*o*

Fractures: Both are gray, granular; some flaw on outside, owing to imperfect cohesion, caused by film of gas or slag.
Casting: Fluid; sound.
Remarks: Very brittle and rotten in unannealed casting.

A 8.

28 lbs. Swedish iron.
28 lbs. spring scrap.
4 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.49	.16	.23	.31	.023	.049

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test: Annealed.	1167	13.59	32.7	16.9	15	2–*o*
	1168	13.18	32.47	22.28	15.5	2–*o*

Fractures: Both are gray, granular, and have small flaw on outside.
Casting: Fluid; runs sound.
Remarks: Unannealed casting broke easily, brittle; close-grained fracture.

A 9.

56 lbs. spring scrap.

4 lbs. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.85	.30	.40	.29	.028	.055

Tensile:	Mark.	Elast L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test: Annealed.	1169	15.87	40.67	4.4	4.5	...
	1170	15.74	21.87	2.6	2.5	10–*o*

Fractures: 1169 Dull gray, coarse, granular; broke at neck of test-piece; no apparent reason.
1170 gray, granular; small flaw on outside of fracture.

Casting: Ran fluid into sound castings.

Remarks: Annealed castings hard and brittle.

A 10.

56 lbs. Swedish iron.

1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.10	traces.	.06	.12	.015	.04

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test: As cast	1189	11.2	22.81	37.1	25	...
...	1190	11.6	22.81	37.1	29.5	...
Annealed	1201	9.72	20.48	60.1	38	...
...	1202	9.97	21.16	30.35	27	...

Fractures: 1189 Coarse crystalline.
1190 Crystalline.
1201 Gray fibrous.
1202 Crystalline.

Casting: Fluid; ran into sound small castings.

Remarks: Tough and malleable.

A 11.

28 lbs. Swedish iron.

28 lbs. spring scrap.

1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.21	.09	.093	.12	.026	.04

Tensile:	Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.
Mechanical test: As cast	1191	11.9	26.28	14.48	13
	1192	13.39	26.78	14.48	12
Annealed	1203	10.02	24.06	50.39	33.5
...	1204	10.47	23.43	52.14	34

Fractures: 1191 and 1192 Crystalline; parts show lack of cohesion, as if from gas-film.
1203 and 1204 Gray, finely fibrous.

Casting: Fluid; ran into sound small castings.

Remarks: Tough and malleable.

A 12.

56 lbs. spring scrap.

1 lb. alloy.

		C.	Mn.	Si.	Al.	S.	P.
Analysis:		.48	.30	.33	.11	.037	.045

		Tensile: Mark.	Elast. L.	Max. S.	Contr. per ct.	Elong. per ct.	Flaw. per ct.
Mechanical test:	As cast	1193	...	17.68	1.3	2	...
	...	1194	...	18.19	...	1	40-*o*
	Annealed	1205	14.3	29.6	10.0	9	2-*o*
	...	1206	14.38	35.7	9.5	8.5	2-*o*

Fractures: 1193 Broke at neck of test-piece owing to a flaw; vesicular, crystalline.
1194 Vesicular, crystalline; small flaw on outside.
1205 Granular; small flaw on outside.
1206 Granular; small flaw on outside.

Casting: Fluid; ran into sound castings.

Remarks: Unannealed, comparatively brittle and hard.

A 13.

56 lbs. Swedish iron.*

1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.15	traces.	.093	.12	.013	.037

		Tensile: Mark.	Elas. L.	Max. S.	Contr. pr. ct.	Elon. pr. ct.	Flaw. pr. ct.
Mechanical test:	As cast	1195	12.4	22.8	30.73	26.5	
	...	1196	11.9	22.8	37.17	25	
	Annealed.	1207	9.97	20.44	61.26	38	
	...	1208	9.92	20.33	64.66	38	

Fractures: 1195 and 1196 Gray, crystalline.
1207 and 1208 Gray, fibrous.

Casting: Fluid; ran into sound small castings.

Remarks: Tough and malleable before annealing.

A 14.

56 lbs, Swedish iron.*

1 lb. alloy.

	C.	Mn.	Si.	Al.	S.	P.
Analysis:	.25	.045	.093	.12	.013	.037

		Tensile: Mark.	Elas. L.	Max. S.	Contr. pr. ct.	Elon. pr. ct.	Flaw. pr. ct.
Mechanical test:	As cast	1197	12.12	22.23	7.84	6.5	
	...	1198	12.96	21.94	9.04	6.5	
	Annealed.	1209	9.39	22.75	52.94	31.0	
	...	1210	9.47	22.93	56.14	32.5	2-*i*.

Fractures: 1197 and 1198, 60 per cent. of area are granular, 40 per cent. coarse granular, but crystalline near outside and of a duller color, appearing to have a different composition to rest of fracture, and show that a film of gas may have prevented cohesion.

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* Each of these entirely of Swedish iron; carbon content altered by addition of more or less "hard cemented bar."

1209 and 1210. Fine fibrous, gray.
1210 has a small flaw on inside of fracture.
Casting: Fluid; ran into sound small castings.

A 15.

36 lbs. Swedish iron.*

1 lb. alloy.

		C.	Mn.	Si.	Al.	S.	P.	
Analysis:		.29	traces.	.074	.11	.013	.037	
		Tensile:	Mark.	Elas. L.	Max. S.	Contr. pr. ct.	Elon. pr. ct.	Flaw. pr. ct.
Mechanical test:	As cast		1199	12.3	24.11	10.25	9.	
	...		1200	9.9	22.81	12.02	7.	15–o.
	Annealed		1211	9.55	23.62	39.9	34.5	
	...		1212	9.35	23.62	47.8	33.0	

Fractures: 1199, 70 per cent. of area is granular, 30 per cent. coarse; crystalline of dull color.
1200, 70 per cent. of area is granular, 15 per cent. coarse; crystalline of dull color, with large flaw on outside. The large crystals appear of different composition to rest of fracture, probably because a film of gas prevented cohesion when crystallization was taking place.
1211, 1212, fine, fibrous, gray.
Casting: Fluid; ran sound into small castings.
Remarks: Hard and brittle when unannealed.

The mechanical tests of the samples A 4 to A 15 inclusive are tabulated on the next page, under the headings "As Cast" and "Annealed," for more convenient study.

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* This is entirely of Swedish iron; carbon-content altered by addition of more or less "hard cemented bar."

As Cast.

Mixture.	Chemical Analysis.			Mechanical Tests.					
A.	Al.	C.	Si.	Mark.	Elastic Limit.	Maximum Strength.	Contraction, Per cent.	Elongation, Per cent.	Flaw, Per cent.
10	.12	.10	.06	1189	11.2	22.81	37.1	25.0	...
"	"	"	"	1190	11.6	22.81	37.1	29.5	...
13	.12	.15	.093	1195	12.4	22.8	30.73	26.5	...
"	"	"	"	1196	11.9	22.8	37.17	25.0	...
11	.12	.21	.093	1191	11.9	26.28	14.48	13.0	...
"	"	"	"	1192	13.39	26.78	14 48	12.0	...
14	.12	.25	.093	1197	12.12	22.23	7.84	6.5	see note
"	"	"	"	1198	12.96	21.94	9.04	6.5	...
15	.11	.29	.074	1199	12.3	24.11	10.25	9.0	...
"	"	"	"	1200	9.9	22.81	12.02	7.0	15-*o.*
12	.11	.48	.33	1193	...	17.68	...	...	see note
"	"	"	"	1194	...	18.19	...	...	40-*o.*
Annealed.									
10	.12	.10	.06	1201	9.72	20.48	60.1	38.0	...
"	"	"	"	1202	9.97	21.16	30.35	27.0	...
13	.12	.15	.093	1207	9.97	20.44	61.26	38.0	...
"	"	"	"	1208	9 92	20.33	64.66	38.0	...
11	.12	.21	.093	1203	10.02	24.06	50.39	33.5	...
"	"	"	"	1204	10.47	23.43	52.14	34.0	...
14	.12	.25	.093	1209	9.39	22.75	52.94	31.0	...
"	"	"	"	1210	9.47	22.93	56.14	32.5	2-*i.*
15	.11	.29	.074	1211	9.55	23.62	39.9	34.5	...
"	"	"	"	1212	9.35	23.62	47.8	33.0	...
12	.11	.48	.33	1205	14.3	29.6	10.0	9.0	2-*o.*
"	"	"	"	1206	14.38	35.7	9.5	8.5	2-*o.*
4	.19	.15	.084	1159	10.9	21.8	37.17	27.0	...
"	"	"	"	1160	9.5	21.16	41.97	27.0	...
5	.16	.33	.14	1161	10.41	26.28	43.3	32.0	...
"	"	"	"	1162	10.82	26.08	43.7	29.0	...
6	.18	.65	.28	1163	14.38	24.3	4.4	3.5	8-*o.*
"	"	"	"	1164	14.38	26.28	9.5	9.0	1-*o.*
7	.33	.28	.22	1165	12.2	25.56	19.95	19.0	10-*o.*
"	"	"	"	1166	11.81	25.58	19.95	18.5	5-*o.*
8	.31	.49	.23	1167	13.59	32.7	16.9	16.0	2-*o.*
"	"	"	"	1168	13.18	32.47	22.28	18.5	2-*o.*
9	.29	.85	.40	1168	15.87	40.67	4.4	4.5	see note
"	"	"	"	1170	15.74	21.87	2.6	2.5	10-*o.*

o. indicates position of flaw as on *outside* or circumference of test-piece.

i. indicates position of flaw in *inside* of fracture.

The following records of tests of castings of Newburn mild steel, passed under Lloyds' and Board of Trade survey, are here given for comparison.

Mark of test	O705	O706	O707	O708	O709	O710
Size of specimen, inches	.753	.757	.759	.760	.755	.755
Original area, sq. in.	.4453	.4500	.4524	.4536	.4476	.4476
Fractured area, sq. in.	.2551	.3318	.2827	.1809	.1734	.1590
Permanent set induced, tons per sq. in.	17.54	17.36	16.77	16.23	11.47	14.47
Maximum strain, tons per sq. in.	27.5	27.2	26.6	26.06	24.9	26.9
Contraction of area, per cent.	42.7	26.2	37.5	60.1	61.2	64.4
Elongation in length of 6″, per cent.	...	...	23.5	...	...	31
Elongation in length of 4″, per cent.	25	22	26	30	33	37
Appearance of fracture, silky, per cent.	100	...	100	100	100	100
Appearance of fracture, fibrous, per cent.	...	...	...	...	...	...
Appearance of fracture, fine granular, per ct.	...	100	...	...	...	...
Appearance of fracture, crystalline, per cent.	...	...	...	...	...	...

Mark of test	O781	O782	O783	O784	O785	O786
Size of specimen, inches	.763	.762	.763	.764	.765	.764
Original area, sq. in.	.4572	.4560	.4572	.4584	.4596	.4584
Fractured area, sq. in.	.2083	.2231	.3247	.3247	.3216	.3444
Permanent set induced, tons per sq. in.	12.21	12.35	15.02	16.36	17.00	16.72
Maximum strain, tons per sq. in.	23.81	25.84	28.10	28.34	28.40	28.15
Contraction of area, per cent.	54.44	51.07	28.98	29.16	30.02	24.92
Elongation in length of 2″, per cent.	38	85	22.5	25	21	21
Elongation in length of 1″, per cent.	55	50	28	36	26	29
Appearance of fracture, silky, per cent.	100	100	100	100	100	25
Appearance of fracture, fibrous, per cent.	...	...	...	...	...	...
Appearance of fracture, fine granular, per ct.	...	...	...	...	...	75
Appearance of fracture, crystalline, per cent.	...	...	...	...	...	...

Tests of Castings of Newburn Mild Steel—Continued.

Mark of test	O2770	O2782	O2773	O2775	O2776	O2780
Size of specimen, inches	.539	.2529	.532	.537	.575	.537
Original area, sq. in.	.2281	.2197	.2264	.2264	.2596	.2222
Fractured area, sq. in.	.1963	.1385	.1320	.1194	.1661	.1520
Permanent set induced, tons per sq. in.	15.65	23.36	21.69	21.69	17.15	20.09
Maximum strain, tons per sq. in.	42.07	37.59	37.49	40.42	40.41	52.27
Contraction of area, per cent.	14.02	36.9	41.6	47.2	36.01	31.5
Elongation in length of 2″, per cent.	24	20	25	24	19	19
Appearance of fracture, silky, per cent.	...	...	...	...	...	...
Appearance of fracture, fibrous, per cent.	...	...	...	...	...	...
Appearance of fracture, granular, per cent.	...	...	...	...	...	...
Appearance of fracture, crystalline, per cent.	...	...	...	...	...	...

Mark of test	O2875	O3151	O3152	O3232	O3258	O3256
Size of specimen, inches	.76	.55	.554	.543	.546	.558
Original area, sq. in.	.4536	.2375	.2410	.2315	.2341	.2448
Fractured area, sq. in.	.3421	.1963	.1885	.1520	.1885	.1963
Permanent set induced, tons per sq. in.	19.68	14.84	15.37	23.14	17.16	20.45
Maximum strain, tons per sq. in.	50.68	44.17	37.06	42.42	43.56	41.5
Contraction of area, per cent.	24.5	17.3	21.7	34.3	19.4	19.8
Elongation in length of 2″, per cent.	14	10	22	21.5	16	16
Appearance of fracture, silky, per cent.	...	40	50	100	...	...
Appearance of fracture, fibrous, per cent.	...	...	...	...	...	...
Appearance of fracture, finely granular, per c.	...	60	50	...	...	...
Appearance of fracture, crystalline, per cent.	...	...	...	...	100	100

[Note by the Secretary of the American Institute of Mining Engineers.—In the discussion of Mr. Hadfield's paper, as originally printed and as furnished to me for publication in this volume, and also in the version published in the *Journal of the Iron and Steel Institute*, and received by me while these pages were in press, there are, at this place, two additional tables of tests of castings of Newbern mild steel, which appear, upon examination, to be merely carelessly-made duplicates of those given on page 1102. The marks of the test-pieces are the same, and the figures correspond so

nearly as to make it certain that the same tests are meant to be recorded, the few discrepancies being evidently clerical or typographical errors. There are also the following differences in the first column of the table: Instead of "*Maximum strain*, tons per square inch," as on page 1102, the term "Ultimate strength" is employed; instead of "*Permanent set induced*," etc., the term "Elastic limit;" and the appearance of fracture is somewhat differently (and less fully) stated.

Under the circumstances, I have felt justified in assuming the tables on page 1102, which correspond in form with those on page 1103, to be the authentic ones, and I have therefore omitted the others, as likely only to perplex the reader.

It may be permissible to say here that great care has been exercised in editing all the papers published in this volume, and that with regard to those of American authors, the version here given is the correct one, while in some cases the same papers or discussions, as presented in the *Journal of the Iron and Steel Institute*, vol. ii., 1890, contain errors, the correction of which did not reach London in time to be made before the printing of the sheets of that volume.

With regard to the papers of English authors which are published in this volume by the courtesy of the Council of the Iron and Steel Institute, I have had in most cases the advantage of the copy in the *Journal of the Iron and Steel Institute*, besides corresponding directly with the authors and with every one who took part in the discussions. By the aid of the co-operation thus secured, I have been able to rectify a number of inaccuracies of statement, and moreover, I have caused the text to be carefully revised and all obvious typographical errors to be corrected.—R. W. R.]

Mr. Hadfield: It is with great pleasure that I note Professor Arnold's able contribution. As his results may be termed the first fruits of the working of our metallurgical branch of the Sheffield Technical School, those who have watched its development with much interest, feel their view strengthened and confirmed, that a systematic study of metallurgy, just as such study in other directions, is absolutely necessary to keep us abreast of the rapid growth of our competitors abroad.

There are several points upon which I must differ from Professor Arnold; but it is difficult, in a brief reply like this, to cover the ground he has touched upon, which comprises so many important questions.

It seems to me, that the experiments quoted principally deal with crucible-steel; whatever may be the application of the crucible process to tool-steel, it is for many reasons much less suitable for the manufacture of steel castings on a large scale. For other processes, whilst in some instances aluminum may be employed advantageously, still, notwithstanding the results quoted by Professor Arnold, I am inclined to believe that its use in the large majority of cases is superfluous. Great stress is laid upon the fact of producing sound steel; now I believe I shall have with me the majority of steel-founders in saying that this is not so much the difficulty nowadays; it is the manipulation of the steel after its manufacture that is so much more important. For instance, the unsound sample as shown by Professor Arnold, would naturally be quite useless for castings, and apart from and without the use of aluminum, no steel-founder would think himself competent, or if he did he would not find customers, unless able to produce sound steel for his moulds.

Then, apart from the crucible-process, all the saving shown by Professor Arnold would not apply. For example, in Siemens steel the addition of aluminum would not offer the incidental gains pointed out, such as the saving in coke, wear and tear of the melting-plant, etc.; therefore, the question is much more narrowed down.

Then, as to feeding the piping with "settling- or shrink-heads," that is, of necessity, a common practice with every steel-founder. There is no reason why the same system should not be applied to the manufacture of ingots, whether large or small.

As to the bearing of my paper upon practical results, I can only say that at our own works the knowledge gained by the series of experiments which I have had the privilege of laying before this Institute, from time to time, comprising the effects of manganese, silicon, and aluminum upon iron, has been of great value to my firm, and I venture to hope it has been so to others. That such alloys have been, as far as practicable, carbonless, was one of the express aims in all the experiments. To introduce high carbon, such as Professor Arnold has experimented with, would have been no service from a steel-founder's point of view. As every steel-founder knows, the difficulty in getting hard steels sound is not great, even without the use of aluminum; in fact, the specimens shown by the photographs seem to be below the average of even crucible tool-steel. That good steel has been made in Sheffield for the past fifty years, and with the crucible-ingot trade, goes without saying.

Referring now to Professor Arnold's different summaries—first,

he says he agrees with me, "that no general rise of temperature accompanies the addition of aluminum." What I said was that no *lowering* of the fusion-point takes place. I believe this is what he means to convey, and it is to be hoped that now, with the general opinion of practical steel-makers confirmed by such well-known scientists as Professors Osmond, Ledebur, Roberts-Austin, and others, the last has been heard of the misleading statements circulated so largely that aluminum lowers the melting-point of iron several hundred degrees. I would specially say that, from some recent experiments with the Le Chatelier pyrometer on molten steel, it is quite possible that the difference between fluid and pasty steel may not mean any great rise or fall in the temperature; therefore, the calorific effect of the oxidation of aluminum to Al_2O_3 when added to molten steel, even if only a few degrees, might render the metal more fluid for a short time; but, in any case, this must certainly not be ascribed to the lowering of the melting-point.

In reply to summary No. 2, and the point upon which, to some extent, we seem to differ, viz., as to the action of aluminum being analogous to that of silicon, as pointed out in my paper, I do not for one moment say that they are identical; in fact, it is clearly mentioned that aluminum was found to be the more powerful; but I do say that, in many instances they produce very similar effects. Let us see where we differ.

Professor Arnold says that aluminum is twenty times as powerful as silicon, or, in other words, that 0.1 per cent. of aluminum (the amount he used for making the experimental products) is equal to 2 per cent. of silicon. If he had made the same experiment with silicon as my paper described at the last Paris meeting of this Institute, I am convinced that he would withdraw such a statement. It is quite beyond the facts known to steel-founders. Whatever may be the best and most suitable amount of silicon to be employed, I have seen plenty of sound castings with 0.20 per cent. of silicon. Only a short time ago, I was told by a large maker of steel-castings that, though he had, two or three years ago, extensively used metallic aluminum and ferro-aluminum, its use was now entirely abandoned; and it should be remembered that whatever method he employed to effect sound castings, required him to obtain the same results as to ductility, tensile strength, etc. Facts like these are difficult to be explained away. Mr. James Riley, of the Steel Company of Scotland, in an appendix to a recent paper read before this Institute, in summing up his remarks upon aluminum-steel states: "Whilst

there are some advantages in using aluminum, the disadvantages are, that it is not desirable to use it in large castings, the steel being rendered more crystalline; that contraction-cracks are more liable to occur with this steel, and also that the results are not so much better as to make it desirable to use the metal in steel intended for rolling, etc."

Whatever may be the advantages of employing aluminum in certain instances, it only again proves the old truth that there may be several ways of accomplishing the same end. This is all I contend for.

I feel convinced that if it were possible to employ the element silicon in its pure state, its effect would still more confirm these views. In my own experiments I obtained the nearest approximation I could to pure silicon, viz., an alloy containing 20 per cent. silicon and only about 2 per cent. carbon. Such an alloy, say sufficient to give 0.2 per cent. silicon, would, I believe, if added to 1 per cent. carbonaceous steel, such as described by the professor, produce soundness and piping of the ingot. Whether the low manganese, such as mentioned (under 0.1 per cent.) would then be sufficient, could be determined only by experiment. One of the samples mentioned in my own paper, although containing over 1 per cent. silicon and under 0.2 per cent. of manganese, was forged satisfactorily.

Thirty years ago Sir Henry Bessemer discovered that silicon gave sound fluid steel. He patented the process, the additions being made in small quantities, viz., 1 pound of silicon (by means of ferro-silicon) to 2000 pounds of decarbonized iron, and the whole question was fully thrashed out in the admirable paper of Mons. F. Gautier, "Solid Steel Castings," and the discussion that ensued at the London meeting of this institute in 1877.

Professor T. Turner, Mason College, Birmingham, in his report to the British Association Committee on the "Influence of Silicon," gave the following results, his experiments being carried out on the most difficult kind of material that can be employed, viz., blown basic steel, which is excessively oxidized, and consequently especially difficult to obtain sound: "Thirty pounds of decarbonized basic metal poured into a crucible remained perfectly quiet for a few moments, but on solidifying suddenly boiled up, and although the crucible was not full originally, 75 per cent. of the metal boiled over. No such action was observed in any mixture to which silicon had been added. An addition of only .04 per cent. silicon, and of which

.02 per cent. remained in the ingot, was sufficient to prevent boiling over on solidification." Therefore, in face of this and of the opinions of both practical and scientific experts who agree with me, I think it is rather a strong assertion to make that the effect of aluminum is twenty times as powerful as that of silicon.

A further striking fact has recently come under my notice in confirmation of the analogous effect of silicon and aluminum. At a recent meeting of the Alloys Research Committee, Professor Roberts-Austen drew my attention to the fact that my claim as to silicon and aluminum producing similar effects when alloyed with other elements was confirmed, first, by the fact that that the atomic weight and volume of the two elements are almost identical; and secondly, by a still more striking confirmation, viz., that he had found that when either of these two elements was alloyed with gold, the same effects were produced. In the cooling curves and other data obtained by means of an ingenions combination of the Le Chatelier pyrometer with a photographic arrangement, and by which absolute accuracy is obtained, the above facts were most clearly brought out. A stronger proof could not be found that silicon and aluminum when alloyed with other elements produce analogous effects. The results of the committee will be published in due course. The consensus of opinion of both scientists and practical workers therefore clearly confirms my views. In the former I would specially name Professor Roberts-Austen and M. Osmond, whose opinions are entitled to great weight from their special experience in these scientific investigations. Mr. Keep, of Detroit, Mich., has also shown most clearly that silicon and aluminum act almost precisely alike upon white iron by causing the combined carbon to become changed into the graphitic form. The same thing was named in my paper as to the effect of aluminum turning a great part of the combined carbon in spiegeleisen to graphitic carbon.

I believe that Professor Arnold's interesting experiment of blowing pure carbonic oxide through a crucible containg molten steel, by which means he obtained an increase of 35 per cent. on the carbon present, would be duplicated if molten steel containing silicon were substituted. Mons. Gautier, in his paper on "Solid Steel Castings," read before this Institute in March, 1877, said, "Silicon decomposes the oxide of carbon, or, which is the same, prevents it from forming; therefore, if blow-holes are filled with oxide of carbon they will be made to disappear by an addition of silicon: $Si + 3CO = SiO_3 + 3C$ [or, as we should now formulate it, $Si + 2CO = SiO_2 + 2C$].

The carbon is deposited and dissolved in the steel, and silica is formed." Substitute aluminum for silicon, and we get $2Al + 3CO = Al_2O_3 + 3C$, and it seems to me we have a most convincing proof from Professor Arnold's own experiments of the analogy between the elements.

This similarity of behavior was the reason that I ventured, whilst quite aware of the technical inaccuracy, to describe the effect of the "*metal*" silicon. I cannot but think we shall eventually find this confirmed by those able to decide. Silicon has not yet been obtained in a sufficient mass to determine its commercial properties. Some text-books describe it as "a non-metallic substance (*at present*)."

Similarities.—To sum up, silicon and aluminum produce soundness; have almost the same atomic weight and volume; have very similar calorific power; deposit carbon in the presence of carbonic oxide; turn combined into graphitic carbon; produce almost precisely the same effect when respectively alloyed with gold, as shown by Professor Roberts-Austen; produce similar effects on carbonless iron, as proved by mechanical tests; have similar fractures both in the cast and forged state; produce coarse-grained metal.

Differences.—Aluminum is more powerful than silicon; how much more, it is at present difficult to determine exactly. Aluminum-steel in its cast state will carry a somewhat larger percentage of its metal before becoming brittle. Aluminum-steel seems less red-short.

Professor Arnold refers to piping and contraction. Steel may contract without piping much. For instance, unsound decarbonized iron is subject to linear contraction, although it does not settle or pipe. On the other hand, cast-iron, if poured into an ingot-mould, will pipe at the top almost as deeply as sound steel, but its contraction is only half that of the latter.

Finally, it should be borne in mind that Professor Arnold's experiments apply to only one kind of steel, viz., crucible tool-steel ingots. For the majority of steel work, I do not think that at present the metal aluminum can compete with the results obtained from cheaper mixtures and methods. Even if aluminum costs only 5*s.* ($1.25) per pound, and is employed in quantities of $\frac{1}{10}$th per cent., to a maker of say fifty tons of steel castings per week, an extra cost is involved of £1500 to £2000 ($7200 to $9600) per annum; rather a serious item in a steel-founder's bill.

I regret, that under the rules of discussion, Professor Arnold will

not have an opportunity of here replying further to my remarks. If the points on which we differ can be cleared up, he will find me willing to amend the errors of my ways. As a steel-manufacturer, one is taught not to dogmatize, and on several points raised by the professor, and in my own paper, there is much room for more light.

I should like to draw special attention to Professor Ripper's remarks at the close of Professor Arnold's lecture, with reference to the mechanical worker in conjunction with the metallurgical engineer. The former often designs and sends out patterns which are most difficult to cast successfully in steel. If a little more care were given in adapting the article to the material, very many of the troubles, such as unsound work, wasters, etc., would be avoided.

If members could see some of the patterns that are sent to a steel-founder—patterns that would be even difficult to make in cast-iron, where the contraction is only $\frac{1}{8}$ of an inch per foot—he would understand why a steel-founder is not the sweetest-tempered person in the world.

Mr. Dagger contributes some interesting information; but is there not some mistake as to his remarks respecting hardness, where he mentions that the sample of 7 per cent. aluminum steel would scratch glass? Probably there was much carbon also present; in that case, would not the hardness be rather due to the latter? The carbonless samples mentioned in my own experiments with 5 per cent. and 9 per cent. aluminum respectively were *exceedingly soft*, and could be easily drilled.

As regards the effect of aluminum upon the magnetic properties of iron, my own results entirely differ from those mentioned by Mr. Dagger. Samples containing as high as 5 per cent. and 9 per cent. of aluminum were exceedingly subject to magnetic influence; and ferro-aluminums containing 11 per cent., 14 per cent., and 16 per cent. aluminum were also decidedly as magnetic.

I have again tested and confirmed this since receiving Mr. Dagger's remarks. A small magnet will lift two or three ounces of each of the foregoing percentages of the alloy.

As regards M. Pourcel's remarks, I think that he has overlooked the fact that practically the same results as to soundness are obtained either with metallic aluminum or with ferro-aluminum. My own experiments with alloys of silicon and iron seem clearly to prove that if silicon and aluminum are added in the former case as silicide of iron or silicide of manganese, and in the latter as ferro-aluminum or aluminum, they act similarly in producing soundness. It seems,

therefore, only fair to assume that if such results obtained with ferro-aluminum correspond with those obtained with ferro-silicon, then if a pure silicon addition could be made—that is, the element itself—similar results to an addition of the metal aluminum would follow. As pointed out in my own paper in Paris, it would in every way be an advantage to have this experiment actually made. If any one will furnish me with pure silicon, or say where it can be procured in sufficient quantities, I shall be glad to test and settle this point.

The assumption is further supported from the results obtained by adding 20 per cent. of ferro-silicon, which at present is the nearest approach to an addition of the element itself to carbonless iron.

It seems to me, therefore, that the point as to whether the "addition" or "physic" is made by means of the elements themselves or their ferro-alloys, is not of such importance theoretically. Practically of course there is an advantage in using less bulk, and of not chilling the molten baths to which they are added, though this latter point is obviated by *molten* ferro-additions.

In reply to Mr. Sample's remarks, it would have been better if he had mentioned the special advantages obtained by Professor Langley. At a recent interview with that gentleman during the Iron and Steel Institute visit to Pittsburgh, I understood that he found difficulty in fixing the precise advantage obtained by the use of aluminum in steel-manufacture.

I note Mr. Sample says, that whilst $\frac{1}{2}$ pound of aluminum has a quieting effect upon steel, if it is "wild" 2 to 3 pounds per ton would be wanted. Thus, even with aluminum as low as 5*s*. ($1.25) per pound this would mean an extra cost of 15*s*. ($3.75) per ton! Besides also there is clearly some reason for these "wild" heats, which are the exception and not the rule. With proper care, and increased knowledge in the behavior and treatment of molten steel, such heats will be no doubt eventually avoided. As a proof of this, see the interesting discussion that ensued after the reading of Mr. J. Davis's paper on "The Manufacture of Basic Open-Hearth Steel" before the Cleveland Institution of Engineers in December, 1890.

With reference to the records of Mr. Spencer's valuable experiments, I would draw attention to the fact that his general conclusion as regards the effect of aluminum upon the properties of iron and steel is, that such addition is made at the expense of ductility; in fact, in higher-carbon steel he particularly points out that its use is disadvantageous.

SPIRALLY-WELDED STEEL TUBES.

BY JAMES C BAYLES, NEW YORK CITY.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute.)

THE ideal pressure-tube is obviously the one which combines the greatest strength with the least weight of material consistent with the uses for which it is designed or employed. The inside of the pipe is the first concern of the engineer. It should be cylindrical, smooth, and of the requisite capacity for the purpose in view. It should be in lengths as great as can be conveniently handled, so that the couplings may be as few as possible. With the walls of the tube the engineer has no further occasion to concern himself than to make sure they are adequate to carry the internal and external pressures to which they will be subjected. It is as true of a pipe as of a truss, that when it is strong enough to give the requisite factor of safety, nothing is gained by having it stronger. Anything more than this means dead weight, which has to be paid for many times, first as material, and again and again in the costs of transportation and handling, until it reaches its place in the trench or on supports. In use it must be and remain tight against all leakage, and must sustain whatever is placed over or upon it, or suspended from it, without deformation or injury. Finally, in addition to meeting all the immediate requirements of service, it must possess qualities of durability which warrant a reasonable expectation of a long life of usefulness. The wise engineer no longer regards it as reasonable to depend upon mere bulk of its own material for the protection of a metal as readily impaired or destroyed by oxidation as iron or steel. The metal must have a coating which will be and remain impervious and indestructible. When all these conditions are met, we have as near an approach to the ideal tube as can be attained with the materials now available for their manufacture. Steel used constructively, to the best advantage, meets these requirements most perfectly. Gold alone would do it better. If we search through the whole list of metals and alloys we find nothing available which so perfectly meets the requirements as a pipe material as steel.

Mr. Hamilton Smith, Jr., in a paper on the expedients adopted on

the Pacific Coast in the extensive schemes of hydraulic engineering, for which that part of the world has long been famous, thus states the conclusions to which his observations have led him:*

"The ideal conduit for high pressures is a welded steel tube. Such tubes could probably be subjected to a tensile strain of 25,000 pounds with perfect safety, and would be much preferable to riveted pipe—not only on account of superior strength, but also by reason of almost perfect interior smoothness. . . . . The adaptation of a superior and cheap metal, such as mild steel, for conduits, will permit the construction of hydraulic works in many parts of the world which now appear to be impracticable, owing to the cost of many of the methods still in use for the transportation of water."

The accuracy of this generalization is not surprising, considering the opportunities afforded Mr. Smith for the study of this question, where substitutes for the hitherto impracticable light welded pipe of iron or steel have been used most extensively and successfully. The manufacture of light steel tubes by the processes employed in making the heavy tubes to which we have become accustomed, has been found impossible for mechanical reasons. With the weight of metal needed to carry a welding-heat from the furnace to the rolls, the pipe produced is necessarily heavy. Hence the extensive employment of riveted pipes during the past few years, not only on the Pacific Coast, but all over the world. The best that can be said of them is that they have been used simply because nothing better was available.

Some twenty years ago an ingenious American engineer and mechanic, the late John B. Root, conceived the idea of making light and strong tubes by spirally winding bands of iron or steel so that the edges should overlap, and of heating the overlapping edges and welding them together during the forming process. The novelty of this idea consisted in confining the heat to the edges to be united, and welding the seam as the pipe was formed. The obviously greater strength of a tube thus made over one having any form of longitudinal seam, had already attracted attention. Many years before, as appears from English Patent Office records, an attempt had been made to produce a pipe having a helical weld. The blank was wound cold, with overlapping edges, and placed in a furnace to be heated. When the welding-temperature was reached it was drawn out, and an attempt was made to weld it on a mandrel by the

* On Wrought-Iron Conduit-Pipes.—*Journal of the Iron and Steel Institute*, No. 1, 1886, pp. 133 to 139.

pressure of rolls, externally applied. The experiment, though interesting, was hopelessly impracticable. To carry a welding-heat from the furnace to the rolls, the material operated upon needed to be of a gauge so heavy as to render it impossible to cold-wind it. Furthermore, it was impossible to keep the diameter uniform, as with every turn of the skelp it gained in size an amount equal to the thickness of the metal used. The problem was not at all akin to that of making the twist gun-barrel; and after it was found that tubes could not be made in this way, the experiment was abandoned.

When Mr. Root began his experiments he probably had very little idea of the difficulties involved in accomplishing what now seems so simple. He began by cold-winding, and his first invention in this line was one which has had great value; I refer to the spirally-riveted pipe. The machines for making this form of pipe, though never quite automatic, were very ingenious, and have done good service. They wind the metal, punch holes in the overlapping edges, and as the rivets are dropped in by hand, head them by compression. The process is obviously limited to the use of small and soft rivets, which can be "upset" cold by compression sufficiently to hold them in place. That pipe thus made has any greater utility than ordinary stovepipe is due to the fact that its seam is so disposed that it is not readily opened by an internal pressure exerted circumferentially. This form of riveted pipe is made of material too light to be caulked, consequently it depends for tightness wholly upon the coating of zinc or coal-tar given it. As a preservative coating for either iron or steel, zinc has little value; but it serves very well as a solder to stick the lapped edges together and make a riveted pipe tight against any pressures not exceeding the strength of the material at the line of the rivets. The black coating, when properly made and applied at a temperature sufficiently high, also answers very well to close the seam against light pressures. In some places this pipe has rendered excellent service; in others it seems to have failed conspicuously. I presume its life depends largely upon how it is laid, and upon a light and measurably constant pressure. From its construction I should not expect it to stand shocks without developing leaks. As a matter of fact, however, the spiral riveted pipe has been largely employed, and it is still manufactured extensively.

During some ten years Mr. Root continued his experiments amid many discouragements. It was not until 1883 that he succeeded in building a machine which, in a rude way, demonstrated the practi-

cability of making pipe from strips of iron wound spirally, heated along their overlapping edges only, and united by hammering. This machine, which is carefully preserved as a mechanical curiosity, is a marvel of ingenuity, but not at all practical commercially. Mr. Root had throughout made the mistake of supposing that he was dealing with a mechanical problem simply. That he overlooked the chemical factors is evident to those who, after his death, took up his work and made it successful. It is true, however, that he produced some pipe in short lengths; and curiously enough, owing to the slip of the feed and other mechanical defects in his machine, some of this pipe was of surprising smoothness and beauty. That very little of his experimental pipe was really welded was due to causes for which, as a mechanic, it probably did not occur to him to look.

At Mr. Root's death, which occurred before he had fully worked out the problem to which he had devoted so many years of patient labor, it was found desirable to depart radically from the lines of his experimentation, and to supplement his work by original investigation in many directions. For the purposes of this paper it is sufficient to describe the method of manufacture now employed, which represents the present state of the art.

The only material now used in the manufacture of spirally-welded tubes is mild steel. This is at once stronger, cheaper, and more uniform in quality than any iron available for the purpose, and owing to its homogeneity and density it resists destructive oxidation much better than iron of fibrous texture. Either Bessemer or open-hearth steel is used. The specifications as to composition are not arbitrary. Phosphorus is not considered, being low enough in any steel yet tried. The carbon must be so low that the metal will not harden or become brittle when cooled with water, as suddenly as possible after the weld is made. The sulphur must not be high enough to defeat welding, and copper must not be present at all. The steel employed has an average elastic limit of 35,000 to 40,000 pounds, and an ultimate strength of 55,000 to 60,000 pounds. Anything conforming to these very general specifications has been found to weld satisfactorily and make excellent tubes.

The steel is rolled to the required gauge in sheets which will square 20 feet long, and is slit into bands 12 inches or 18 inches wide. Theoretically, there would be an advantage in establishing and maintaining a fixed relation between the diameter of pipe and width of skelp, but in practice it has been found wise to use two widths only for the diameters thus far produced. Tubes of 6 inches,

8 inches, and 10 inches diameter are made from skelp 12 inches wide; those from 12 inches to 24 inches diameter are made from skelp 18 inches wide.

As skelp in 20-feet lengths is not long enough to make tubes of the length most convenient for use and economical for shipment, the strips are united by welding the ends together. The machines for this purpose are very simple in construction and operation, and do their work very rapidly. The resulting weld is so smooth as not to be visible in the finished pipe. As the stock is thus prepared by what is known as "cross-welding," it is reeled and passed to the tube-machines. By this means the pipe-maker is rendered independent of the limitations of the rolling-mills, which, in the gauges demanded, cannot, with present appliances, advantageously roll more than 20-feet lengths. So cheap and rapid is the cross-welding process, however, that it is doubtful if any advantage would be found in having the strips longer. They would be more costly to transport and handle. The fact that, with cross-welded stock, tubes 100 feet long have been made, and that if it was of advantage to do so they could be made 500 or 1000 feet long, shows how little the pipe-maker is dependent upon the rolling-mill manager. It is, however, useless to make pipes exceeding in length existing facilities for transportation. Thirty feet is the average car-length, and on certain patterns of platform cars, 32 feet can be loaded. To exceed this length would be to increase the difficulties of transportation and handling far more than the value of the couplings saved. The cross-welded stock is passed to the tube-machine, and one end is laid on the guide-table, which, being adjustable, is set to the angle due to the width of skelp and the diameter of the pipe to be made. On this table are the feed-rolls, geared together, which draw the stock into the machine and pass it forward. As the skelp enters, it moves across the face of an anvil-horn, which is rounded to conform to the shape of the pipe. The curvature of the stock is effected by the movement of an oscillating jaw, which presses the metal against the face of the anvil. The impetus of the feed-rolls is intermittent, moving the metal forward when the shaping-jaw is raised, and leaving it at rest when pinched between the fixed and moving parts of the shaping-mechanism.

The source of heat is a small furnace of cast-iron, scarcely more than a cubic foot in dimensions. This furnace is lined with refractory material, having passages of such shape as to deflect the gas and blast where they are needed. The furnace contains two or more

blow-pipes, as may be required. The fuel is a gas of a quality never before employed in any metallurgical operation. It is composed, as nearly as may be, of equal parts of hydrogen and carbonic oxide, is free from carbonic acid and sulphur compounds, and has a flame temperature of 3800° Fahr. It is delivered to the machines under a considerable pressure, and mingles with the requisite volume of blast at the point of ignition. The practice in burning the gas, as in making it, is different from that followed anywhere else, so far as I am aware, but it has been found to be the one best adapted to the special requirements of the industry.

Heating and welding are, of course, the only elements of the process involving any particular difficulty. The nature of this problem can be judged from the fact that it required more than two years of very costly and anxious experimentation, in directions in which previous experience furnished no guide. To pass cold steel into a flame in which iridium will fuse, bring it from atmospheric temperature to that of welding in about five seconds, weld it and quench it with water at once, without burning the metal or producing any appreciable scale, is a metallurgical operation of great delicacy, and one which demands that all the conditions shall be right. If the flame assumes an oxidizing character for an instant, the weld is missed ; if the stock binds, the feed slips, or for any other reason the steel remains in the heat an instant longer than is needed, it will run as fluid as water and leave a hole in the pipe. When I say that a confident operator, who understands the conditions under which he is working, will often leave his machine and attend to something else while it is making pipe at the rate of 18 to 24 inches per minute, the perfection of the machine and process will be appreciated. It is not, however, so automatic that the operator can leave his machine with entire safety. With a constant pressure of air and gas, it would only need to be started right to finish a pipe without further attention. But as machines are started and stopped, producing slight variations in the engine-speed, and for other causes the pressure of air and gas varies, the operator needs to increase or diminish the speed of his feed. This is accomplished by a modification of the link-motion, which enables him, while the machine is running, to stop the feed altogether, or increase it so that it shall drive the skelp forward at the rate of more than 200 inches a minute. The average running-speed in pipe-making advances the skelp from 90 to 100 inches per minute. I have no doubt that further improvements in the plant will make the process so nearly automatic that, after the

operator has run out a foot or two, he need give his machine no further attention.

It is essential that the gas and blast should be as perfectly under control as the feed. There are always two edges to be equally heated—one at the end of the already formed pipe, and the other the edge of the flat or entering skelp which is about to be added on. For this reason the operator must be able to vary the proportions of gas and blast in each blow-pipe. This he can do without leaving his position or abandoning the lever by which he controls the feed mechanism. The direction of present experiment is to make the pressure of gas and air so constant that the operator, having ascertained the proper rate of feed, can let one reel of skelp follow another with no concern for his product unless something breaks.

The machine having these complex functions of feeding, heating, welding and discharging the pipe is quite simple in construction, and has but few working-parts. It is of unusual interest to see cold skelp drawn in, heated, shaped, welded, and passed out as finished pipe, at the rate of a foot to two feet per minute, with all the operations by which flat steel is converted into pipe, performed within a space which could be covered by one's hat. I venture to say it is wholly unlike any industrial process to be seen in this or any other country.

The welding is done under a light hammer, which strikes about 320 blows per minute. The hammer is so arranged as to give what is known as a dead blow. The weld thus secured is vastly superior to that attainable by the pressure of rolls. A hammer-blow can be delivered at the mouth of the furnace, and does not rob the metal of the heat necessary to produce a perfect union. Pipe can be made without the hammer, by the contact of the jaws which bend and form it, but the hammer is more certain. The welds produced by hammering are as perfect as are ever made on the anvil by a skilled blacksmith. Test-pieces subjected to tension show that the metal tears before the weld separates. As the direction of the weld is such that, under pressure, it is never subjected to a strain tending to open it, it serves as a spiral reinforcement, and pipes made by this method, unless they show a leak in the initial test, can be depended upon to develop a strength equal to the ultimate strength of the metal composing them. The spirally-welded pipe is an illustration of the best constructive use of material known in the arts. It is, in fact, stronger than a seamless drawn steel tube of the same diameter and thickness of wall.

The weight of steel used in making spirally-welded tubes ranges

from No. 14 to No. 6 B.W.G. The diameters now made include all sizes from 6 inches to 24 inches, and will probably be extended to include smaller and larger sizes. The limit in small diameters is that which will admit an anvil-horn stout enough to withstand the heat and hammering to which it is subjected, and remain rigid under the compression of the shaping-jaw. There is no limit in the direction of large diameters except such as is imposed by the present difficulty of obtaining stock of suitable widths and lengths, and the cost of transporting large and light cylinders. When the demand for pressure tubes of larger diameter than 24 inches shall justify provision for their manufacture, there will be no difficulty in making them.

After the pipes leave the machines the ends are squared on a trimming-machine, and they are tested in the usual way. They are then provided with their couplings, and are finished by dipping in a bath of Trinidad asphalt. The coating is hard, insoluble and extremely tenacious, and as a protection against rust is very durable.

This industry has been established scarcely long enough to have a history, but it has experienced a very rapid development. Its product has gone into every use for which pipes of diameters of 6 inches and upwards are employed. They are in successful use in steam and compressed-air conduits, gas mains, water pipes, petroleum lines, and to some extent in architectural and engineering constructions. In mining-operations they have met every requirement of service—the saving in transportation-charges often amounting, at remote and inaccessible points, to a very large item. They have also been found useful for many purposes for which timber has hitherto been employed—as in trestles, tramway-trusses, supporting columns, etc. A 12-inch tube, 30 feet long, with couplings, which would be safe under a working pressure of 500 to 700 pounds per square inch, will weigh, according to the thickness of metal used and the coupling provided, from 475 to 490 pounds. Thirty feet of lap-welded tube, 12 inches in diameter, will weigh not less than 1700 pounds, and 30 feet of 12-inch cast-iron pipe, strong enough to carry a working pressure due to a head of 100 feet, and lighter than that usually demanded in water-works construction, will weigh from 2700 to 3000 pounds, and perhaps more. The spirally-welded steel tube certainly has a range of utility greater than that of any tube previously known, and it places at the command of the engineer a material which broadens his field of operations in all directions, and renders practicable undertakings of great magnitude which have hitherto seemed impracticable on the score of cost.

NOTES ON THE BESSEMER PROCESS.

BY HENRY M. HOWE, A.M., S.B., BOSTON, MASS.

(From the Proceedings of the New York Meeting of the Iron and Steel Institute, Oct., 1890.)

THE striking features of American Bessemer practice are its large output and its low initial silicon and initial temperature. These are interdependent. Large outputs implies short blows and short intervals between blows. Short blows imply low silicon and fast blowing. Low silicon implies short intervals and fast blowing. Our blows are short because we have little silicon to remove, and because we remove it fast by supplying blast rapidly. We get enough heat even with little silicon, because our blows and the intervals between them are so short that relatively little heat escapes from our vessels. Were we to lengthen our intervals, or to blow more slowly, we would have to meet the increased loss of heat from our vessels, either by using more silicon, or by melting hotter, *i.e.*, by burning more coke, in the blast-furnace or in the cupola. Finally, our short intervals imply powerful machinery, efficient organization, and extreme specialization and subdivision of labor, which can be profitable only when the output is large.

Let us now take these points up in more detail, first glancing at some examples of our large outputs, given in Table I., and of the composition of our iron.

Output.—On presenting such results to some foreign metallurgists, I have been told, "Oh yes, we quite understand that. You really turn out no more steel than we do. As a matter of personal pride, your manager gets up a tremendous spurt for a day, or even a week, crowds on men wastefully, rushes through great quantities of metal, and then for months afterwards has to recuperate. The machinery has been strained; the owner's money has been wasted; the men have been prostrated—all to gratify the manager's vanity. Your average output is no larger than ours."

Personal pride has, doubtless, played its part; without it we would still be savages, or apes, or mollusks, if you prefer. These baser motives are the mudsills of progress, the soiled roots of that growth of which civilization is the flower. Great outputs were made

TABLE I.—*Output of American Bessemer Works and Rail-Trains.*

	Date.	Name of Works.	Ingots from One Pit.								Number of Shifts in the Month.	Rails.						
			Tons.				Heats.					Tons.				Rails.		
			Per 12 Hours.	Per 24 Hours.	Per Week.	Per Month.	Per 12 Hours.	Per 24 Hours.	Per Week.	Per Month.		Per 12 Hours.	Per 24 Hours.	Per Week.	Per Month.	Per 12 Hours.	Per Week.	Per Month.
2-Vessel Plants.	1889.	Homestead rail steel	477	891	4,477	19,572	91.5	170	809	3,636	71	...	...	...	...	...	...	...
	...	Homestead soft steel	372	619	...	13,291	69	116	...	2,436	71	...	...	...	...	...	...	...
	March, 1889.	Scranton . . .	...	...	...	18,303	...	...	...	...	47	...	...	...	...	...	...	15,879
	July, 1889.	Union	...	1,108	7,024	...	...	...	...	...	...	...	1,149	5,783	...	...	19,392*	...
	Oct., 1889.	Scranton . . .	488	...	...	...	76	...	...	...	...	454.5	...	...	...	1,818	...	...
	Oct., 1889.	Union (10 tons) .	852	1,639	8,549	36,232	85†	164†	855†	3,623†	54	660	1,312	6,766	28,491	...	...	...
	Nov., 1889.	Scranton ($6\frac{1}{2}$ tons)	491.8	975	...	...	71	141	...	...	...	...	...	...	...	...	...	...
	Dec., 1889.	Scranton . . .	501	...	...	...	73	...	...	...	...	...	...	...	...	...	...	...
	March, 1890.	Scranton . . .	...	...	5,456.7	...	...	...	...	...	...	...	...	...	...	...	...	...
3-Vessel Plants.	...	Harrisburg . .	544	1,029	5,110	20,947	75	141	678	2,929	48	...	...	...	...	...	...	...
	...	South Chicago .	701	1,303	6,402	27,427	60	119	556	2,441	50	...	...	...	...	...	...	...
	Oct., 1889.	Edgar Thomson	...	...	7,225	...	...	...	664	...	...	...	...	...	...	...	12,073	...
	Jan., 1890.	Edgar Thomson	...	...	...	...	...	...	...	...	...	...	1,250	...	...	2,502.9	...	...
	Feb., 1890.	Edgar Thomson	...	...	...	...	...	...	...	...	...	...	...	...	25,600	...	...	...
	March, 1890.	Edgar Thomson	...	1,648	...	...	...	...	...	...	...	...	1,417	...	...	...	...	...

* Calculated. During ten out of the eleven shifts 66-lb. rails were rolled, and on the eleventh shift 75-lb. rails. Assuming that the average weight of the rails was $\frac{66 \times 10 + 75}{11} = 66.8$ lbs., the number of rails would be 19,392.

† Calculated, assuming that the average weight of ingots per heat was 10 tons, which is the nominal weight.

and published; generous rivalry sprang up. Each showed the other how he made his great output, each gained the experience bought by the other's toil, till the average output of to-day is far greater than the boasted record-breaking output of our centennial year. Whatever may be the merits of our tariff-system, we have pushed free-trade far in some important directions—free-trade in ideas, free-trade in land, free-trade in labor.

Is the charge true, that our great outputs are but isolated *tours de force*, rising like Teneriffe above the dead level of our every-day mediocrity? I have before me the returns from the Adonde* (American) Bessemer Works, from July 1, 1889, to June 30, 1890, a period of fifty-two weeks and one working day. As we work eleven shifts a week, this gives a total of 574 shifts available for work. But the period actually comprised in the returns covers four additional shifts, or a total of 578 shifts. Out of these, the Bessemer mill ran during 542 shifts, leaving thirty-six idle shifts to cover holidays, strikes and stoppages for repairs and lack of orders. Three of these were for celebrating the national holidays; thirteen at the winter holiday season for general repairs; nine owing to a strike, and eleven owing to an order from the directors, for reasons of which I am ignorant. The works have two 10-ton vessels. The total annual output was 318,635 tons, which implies an average output of 26,553 tons per month, or of 1176 tons, or about 118 blows per twenty-four hours, or of one blow every 12.2 minutes the whole year through. During the same period 260,881 tons of rail were made, which implies an average output of 489.5 tons per shift, or 979 tons per twenty-four hours, or 5384 per week. Less than ten years ago Captain Jones described to you his work during the preceding year, in which he had made 123,303 tons of ingots. He gave it as his belief that we were then on the verge of the elastic limit of production, though he thought that in a long month 15,000 tons might be made. Thus, in about nine years our output has increased more than two-and-a-half-fold. Captain Jones' supposed elastic limit has proven to be no breaking-down point: under a stress more than twice as great no contraction of area occurs, and our ultimate strength is not yet in sight.†

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* I use cipher-names expressly to avoid indicating what works are referred to. For a like reason, I decline to state whether I always refer to the same works in the same way, or not.

† *Journal of the Iron and Steel Institute*, 1881, i., p. 129.

It is reported that the Scranton works, with two 6.5-ton vessels, made 104,441 tons of rails during the first six months of this year. This implies an average of about 144 heats per twenty-four hours, or a heat every ten minutes during the whole of this period, and of at least 4017 tons of rails per week.

*Proportion of Silicon in the Vessel-Charge.**—While there are American mills, and good ones, which habitually have 2 and even 2.25 per cent. of silicon in the vessel-charge, and while I know cases in which the silicon-content has reached 6.9 per cent., the majority of mills have less than 1.78 per cent.; and what appears to me the most characteristically American practice† has habitually but from 0.6 to 0.9 per cent. of silicon in the vessel-charge. I have before me returns from a number of successive heats, whose composition lay between the following limits; they represent the usual practice at Ningunaparte, except that the manganese is a little higher than usual.

Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.
3.36–3.66	.64–.80	.49–.70	.079–.086	.053–.075

In another case the silicon in ten consecutive heats lay between 0.54 and 0.78 per cent., averaging 0.668. The iron was melted in cupolas with one part of coke to 11.77 of metal.

	Per cent.
The loss for the month (including cupola loss) was . . .	7.89‡
The proportion of scrap was	5.09
The yield of ingots per 100 of cast-iron was	87.02
	100.00

* I invariably refer to the average percentage of silicon of the whole vessel-charge, scrap included, taken as it goes to the vessel itself. From the numbers which I give, the loss of silicon in the cupola has already been deducted.

† The words "characteristically American" must be taken in their exact sense. I do not mean that this is the habitual American practice. It is not. But it is most characteristically American to carry on operations so extremely rapidly, and hence with so small a proportion of loss of heat, as to permit this low silicon content, and, unhampered by precedent, to use it boldly.

‡ It is certain that this is the monthly loss recorded in the books of the establishment; but as it represents the returns for a single month only, errors in weighing and in estimating stocks may have distorted it somewhat. It is given here simply to show that, at least, no excessive loss occurred, and that the remarkably low proportion of silicon in the pig-iron cannot be explained away by supposing that an excessive quantity of iron was oxidized in its stead to generate heat, either in cupola or converter. The loss at this same mill for the third quarter of 1890 was 8.8, the scrap 3.2, and the yield of ingots 88 per 100 of pig-iron. Very careful attention is now paid to weighing, and the management is confident that these numbers are accurate.

At this mill less than 16 pounds of coal per 2240 pounds of ingots is used in heating the vessels and ladles, according to the yearly returns.

In order to blow iron with so little silicon successfully, the heats must follow each other quickly, and vessels and ladles must be hot. In running at the rate of five heats an hour, the metal whose composition I have given with 0.64 to 0.80 per cent. of silicon might be further diluted with 5 per cent. of scrap, reducing its silicon-content to between 0.61 and 0.76 per cent. If, on the other hand, only 1.5 heats were blown per hour in each vessel, the iron should have about 1.25 per cent. of silicon. Indeed, I think that most of those who use less silicon than this still admit that, as far as mere convenience of blowing is concerned, 1.25 per cent. is the best proportion.

Iron with much less silicon has actually been blown. It is stated that white iron containing 0.12 per cent. of silicon and but little manganese was once blown; and I know that iron with 0.25 per cent. of silicon and 1 per cent. of manganese has been blown intentionally, sandwiched between normally hot heats.

Metal with only 0.50 per cent. of silicon has been blown in Sweden; * but I understand that this was done only when the initial temperature was decidedly high. The practice which I have described, however, is that which Dr. Müller, in his admirable study of the German Bessemer process, called "the fourth and practically impossible variety of the Bessemer process, low initial temperature and low silicon-content."†

Low-silicon pig has, besides the advantages already referred to of being a relatively cheap pig to make, and of permitting large output, the further advantages of implying small loss of weight in conversion, little wear and tear of refractory materials and little power for blowing.

It costs much to put silicon into pig-iron, and much to take it out again. Not only does its own removal in the Bessemer process mean a loss of weight, but it drags much iron off with it into the slag. Further, by lengthening the blow it increases the wear and tear of the tuyeres, and of the refractory lining of the vessel and bottom, and every pound of silica thus abraded or corroded implies further

* Tchernoff, *Metallurgical Review*, vol. i., p. 352.

† "Die vierte Variation, niedrige Anfangstemperatur und niedriger Siliciumgehalt, ist practisch unmöglich."—*Zeit. Vereins Deut. Ingenieure*, 1878.

scorification and loss of about a pound of iron, besides implying expense of buying, preparing and setting in place new refractory material in its stead. Finally, the blast needed for its removal implies expense for fuel, for wages of engineers and firemen, and for wear and tear of engines and boilers.

For all these reasons, the less silicon we can get on with the better, and much the same may be said of manganese. Coke is cheaper than silicon. It pays to burn an excess of coke in the cupola, if by thus increasing the metal's actual heat we lessen proportionally the potential heat—that is, the silicon—which it must have. But in our practice this is unnecessary, thanks to our quick blowing and short intervals; we use only enough coke to thoroughly liquefy the iron.

Rate of Removal of Non-Ferrous Elements—Table II., which gives the composition of the metal and the slag at different periods in a typical American blow, shows the results of a laborious investigation carried out for this paper by Mr. Frank Julian, chief chemist of the Illinois Steel Company's works at South Chicago. Our hearty thanks are due him.

There appears to be a slight removal of phosphorus. As the loss of weight in conversion was 9.37 per cent., the resulting steel should contain 0.116 per cent. of phosphorus, if all the phosphorus present in the initial charge were concentrated in it. Actually the steel contains 0.109 per cent., the discrepancy being 0.007 per cent., or 1.59 pounds of phosphorus; certainly a very small quantity when distributed over ten tons of metal. The slag contains a small quantity of phosphorus, which increases up to the time of adding the spiegeleisen, when it again falls off. Unfortunately the weight of the final slag is unknown; but, from internal evidence, I estimate that it was probably between 1200 and 1900 pounds; and hence that, containing as it did 0.010 per cent. of phosphorus, it should have held between 0.12 and 0.19 pounds of this element.

Can so siliceous a slag contain even this trifling proportion of phosphorus, in presence of molten iron? It is possible that the phosphorus was not uniformly distributed throughout the slag, but was scorified locally from shots of iron during and after the removal of the slag from the vessel. These shots, suspended in the slag, would oxidize, surrounding themselves with a little shell of slag more highly ferruginous than the great mass in which they were suspended, and from them phosphorus would pass into this locally basic layer of slag. It is true that metallic iron and magnetic oxides and silicates were removed by means of a magnet before

analyzing the slag. But while the layers of slag immediately against the shots of metal might well be sufficiently magnetic to be thus removed, some of the phosphorus might well pass beyond these magnetic layers into outer ones, which, though still basic enough to retain phosphorus, were not ferruginous enough to be readily separated magnetically. The fact that the weight of phosphorus accounted for by the slag is only about one-tenth of that apparently lost by the metal, agrees with this view; most of that scorified remaining in the magnetic inner layers of slag next the shots, and being separated thus by the magnet.

A very appreciable quantity of phosphorus may, however, exist in a siliceous slag under favorable conditions. Thus, a heat was very badly overblown at the North Chicago Steel Works in 1878, so badly that only two ingots were made; but the full charge of spiegeleisen, intended for normal conditions, was added on recarburizing. The composition of the resulting metal and slag was as follows:

Metal.	Per cent.
Carbon,	.70
Silicon,	.085
Phosphorus,	.157
Manganese,	.93

Slag.	Per cent.
Manganous oxide,	2.68
Ferrous oxide,	46.20
Silica,	41.30
Alumina,	5.50
Phosphorus,	.03
	95.71

As the spiegeleisen doubtless dragged much of the phosphorus back from slag to metal, there is every reason to believe that this decidedly acid slag, before recarburizing, held much more than 0.03 per cent. of phosphorus.*

The gradual increase of phosphorus in the slag till the end of the blow, and its sudden fall on recarburizing, may possibly be connected with the growing viscidity of the slag, as its proportion of silica increases from 42.40 to 63.56 per cent. towards the end of the blow, and with its again becoming more liquid, as its content of manganous oxide rose from 8.88 to 13.72 per cent. on recarburizing, the quantity of shots suspended increasing with the viscidity of the slag. But I think that a more important factor

* R. Forsyth, private communication, October 18, 1890.

is the progressive diminution in the proportion of carbon and silicon contained in the metal, which thus held its phosphorus less and less tenaciously, and acted less and less energetically in deoxidizing phosphates in the surrounding slag. So, too, the gain in carbon and manganese on recarburizing, with consequent increased resistance to scorification of phosphorus, goes to explain the sudden fall in the phosphorus-content of the slag at the end.

TABLE II.—*Removal of Non-Ferrous Elements.*—F. JULIAN.

Description.	Initial Charge.			Time of Actual Blowing.						
	Molten Metal.	Steel Scrap, Estimated.	Average.	2 Min. 0 Sec.	3 Min. 20 Sec.	6 Min. 3 Sec.	8 Min. 8 Sec.	9 Min. 10 Sec.	9 Min. 20 Sec. A.	After adding Spiegel eisen.
Carbon,	3.10	0.36	2.98	2.94	2 71	1 72	0.53	0.04	4 64	0.45
Silicon,	0.98	0.08	0.94	0.63	0.33	0.03	0.03	0.02	0.35	0.038
Manganese,	0.40	0.97	1.43	0.09	0.04	0.03	0.01	0.01	14.90	1.15
Phosphorus,	0.101	0.10	...	0.104	0.106	0.106	0.017	0.018	0.139	0.109
Sulphur,	0.06	0.08	0.06	0.06	0.06	0.06	0.06	0.06	...	0.059
Silica,	...	...	...	42.40	50.26	62.54	63.56	...	...	62.20
Alumina,	...	...	...	5.63	5.13	4.06	3.01	...	...	2.76
Ferrous oxide,	...	...	...	40.29	34.24	21.26	21.39	...	...	17.44
Ferric oxide,	...	...	...	4.31	0.96	1.93	2.63	...	...	2.90
Mangan. oxide	...	...	...	6.54	7 90	8.79	8.88	...	...	13.72
Lime,	...	...	...	1.22	0.91	0.88	0.90	...	...	.87
Magnesia,	...	...	...	0.36	0.34	0.34	0.36	...	...	.29
Phosphorus,	...	...	...	0.008	0.008	0.010	0.014	...	...	.010
Sulphur,	...	...	...	0.009	0.009	0.014	0.008	...	...	.011
Appearance of flame,	...	...	...	silicon flame	brightening	moderate carbon flame	full carbon flame	flame drops	blown 10 seconds after drop	...
Cubic ft. of air	...	...	...	34,502	30,628	53,481	45,265	26,430	1,868	...
Cubic ft. of air per minute, B.	...	...	...	17,251	22,971	19,691	21,810	25,685	11,208	...

Date, 1890.	Turn.	No. of Heat.	Converter Record.							Record of Blow.			
			Molten Metal.	Steel Scrap before Blow.	Steam Equal to Scrap.	Spiegel.	No. of Heat on Bottom.	Tuyeres.		Temperature.	Rate of Flowing.	Turned Down.	Spiegel Reaction.
								No. 3/8"	No. 1/2"				
Aug. 13	Day	18,398	21,500	1000	800	2500	4th	11	3	Medium	Quick	Fast	Medium

Date, 1890.	Turn.	No. of Heat.	Ladle Record.						Ingot Record.					Bloom Record.	Rail Record.		
			Rate of Flowing.	Nozzle. Size.	Nozzle. Heat on Nozzle.	Behavior in Ladle.	Behavior in Mouldings.	Temperature of Moulds.	Number of Ingots Cast.	Kind of Ingots.	Bend Test.	Time in Git. Hours.	Time in Git. Min.		Weight per Yard.	No. Rolled.	Seconds, %.
Aug. 13	Day	1839	Fast	1¾″	3d	Quiet	Quiet	Warm	7⅔	80%	Left R. A.	1	23	Clean, without cracks.	80%	23	0

Note.—*A.* Sample V. was taken after blowing 9 minutes and 10 seconds. The vessel was then turned up again, and blowing was continued for 10 seconds more, when the spiegeleisen was added, no sample of either metal or slag being taken before adding it. After adding it, Sample VI. of both slag and metal was taken.

B. The volume of blast is calculated from the volume of the blast cylinders, without deducting for leakage or clearance.

The barometer in the engine-room read 29.79 inches; the temperature was 36° C. (96.8° F.) in the engine-room. The blast-pressure was 27 lbs. per square inch.

Each of the fourteen tuyeres has twelve holes. Those in eleven of the tuyeres are ⅝ inch in diameter; those in the remaining three tuyeres are ½ inch in diameter.

Length of Blow.—This depends on the composition of the iron, the blast-pressure, the area of the tuyere-holes per ton of metal in the charge, and the depth of the bath of metal in the converter. Our blast-pressure usually lies between 20 and 25 pounds per square inch, and we provide from 2 to 4 square inches of tuyere-hole area per ton of metal. These conditions are thus about the same as in modern European practice. The chief reason why our blows are so much shorter than the European ones, then, is that our iron is so relatively free from silicon; but our bath of metal may be enough shallower than the European to contribute to the result. The length of the blow in our fast-running works is usually from eight to fourteen minutes, though it is rather longer than this when direct-metal is used. The Union, Scranton and Homestead works have each run for a whole shift at the rate of one heat in 9.86, in 9.23 and in 7.87 minutes respectively. The depth of metal in our newer vessels is usually from 12.5 to 15.5 inches.

Heavy blast-pressure, of course, costs more than light for steam, and for wear and tear of boilers and engines. It has the compensating advantage of prolonging the life of the bottoms. On the one hand it protects the bottom by lifting the metal, there charged with most corrosive iron oxide, away from the tuyere-ends; and by cooling those tuyere-ends, both by holding the hot metal away from

them and by causing vaster volumes of rapidly expanding, and hence cooling, air to pass through them. On the other hand, the increased velocity of the air tends to increase the abrasion or corrasion of the tuyere-ends. As might be expected, experience indicates that its protective greatly outweighs its injurious effect; and that increasing the blast-pressure lengthens the life of the bottoms. Up to what pressure it would, on this account, pay to compress the blast, must depend on the relative cost of steam and of bottoms. I can only say that our best engineers think that we have not passed the limit of economy. The fact that, though we use non-compound and not particularly economical blowing-engines, the total fuel-consumption for steam is sometimes only about 200 pounds* per ton of ingots, or 9 per cent., harmonizes with this belief. Usually, however, the boiler coal-consumption per ton of ingots is larger than this: the yearly returns for one very well-managed works making an enormous output, and using, I believe, the poor Illinois coal, give it as 309 pounds; those of another, also making an enormous output, but not so well managed, gave it as 350 pounds. In still another old but well-managed mill it rises to about 400 pounds.

Large Outputs.—Turning next to our large outputs, one naturally asks, "How are they made?" and "Do they pay?"

One of the two essentials of large output, short heats, we have just considered. The other, short intervals between heats, implies such arrangements for preparing the cast-iron, for conversion, for teeming the steel, and for stripping and removing ingots and setting moulds, that no one of these operations shall be greatly delayed in waiting for another; and enough pit-room to accommodate the ingots resulting from several blows, so that each ingot may stand in its mould long enough to cool and solidify, without delaying the teeming of subsequent heats. If we seek very large output, then, as soon as a heat is blown in one vessel, and before it is recarburized, a second must begin blowing in another vessel; and the first vessel, or the third if there be three, must be ready, so that a third heat may begin blowing as soon as the second is blown. The casting-ladle must receive and deliver its steel, undergo its repairs, and be back ready to receive the second heat as soon as the second heat is ready to be poured into it. So, too, a second set of moulds must be ready to

* The manager of one of our largest steel-works writes me (August 7, 1890) that a careful estimate, covering a continuous run of eighteen months, gave the coal, which was anthracite of a common grade, used for steam-raising in the (Bessemer) converting department, as between 205 and 206 pounds per 2240 pounds of ingots.

receive the second heat as soon as the casting-ladle has received this second heat and swung around to where the moulds stand, and so on.

Five large cupolas, with abundant blast, or ample arrangements for bringing metal direct from the blast-furnace, according to whether cupola- or direct-metal be used; a swift moving casting-crane; at least 60 feet along the rim of the casting-pit available for ingot-moulds, or room for at least four sets of ingot-moulds; four ingot-cranes; blowing-engines, pumps and hydraulic apparatus, capable of running continuously for a week, without a moment's stoppage; hydraulic pressure-pipes that can neither burst nor freeze; ample boiler-capacity—these are what the mechanical engineer must supply. The administrative head must arrange that sufficient stores of pig-iron, ferro-manganese, and spiegel; of coal and coke; of refractory materials, and of moulds, shall be ever at hand, and reach promptly the cupolas, boilers, repair-shops, and mould-yard. He must organize the many groups of workmen needed (each group self-contained and looking to one definite director within or without its own limits, yet each co-operating with the rest) for stocking the cupolas, for charging, tapping, and repairing them; a group charged with the care and repairs of the vessels; another with making the vessel-bottoms; another with the recarburizer; another controlling the length and temperature of the blow—but the Appendix to this paper gives a list of the thirty-four distinct groups of men employed on each shift at the Mugby Junction (American) Steel Works, besides the six groups employed for current repairs on Sundays.

But note that there is absolutely no mystery about it. Almost no chemical knowledge is needed. A good mechanical engineer to build, a good administrator to manage—to select his men with judgment, to treat them with gentle, just firmness. One who can, now by appeal to pride, now to cupidity, identify the interests of men and of masters; who, with contagious zeal, can extract earnest, intelligent, and prolonged efforts from all, while overtaxing and oppressing none; whose deft, velvet-gloved iron fingers can hold a thousand reins, keeping at each moment the needed but varying tension on each; who, ever watching each item of cost, pares each to the limit of safety. All this you knew before; here is no more mystery than in putting up a 100-pound dumb-bell. Given the strength of mind or of muscle, and it is simple enough. What mystery in Patti's carolling, in Napoleon's conquests, in Shakespeare's master-works, in your cricketer's batting? Merely skill, or if you prefer, skill and strength, now of one kind, now of another.

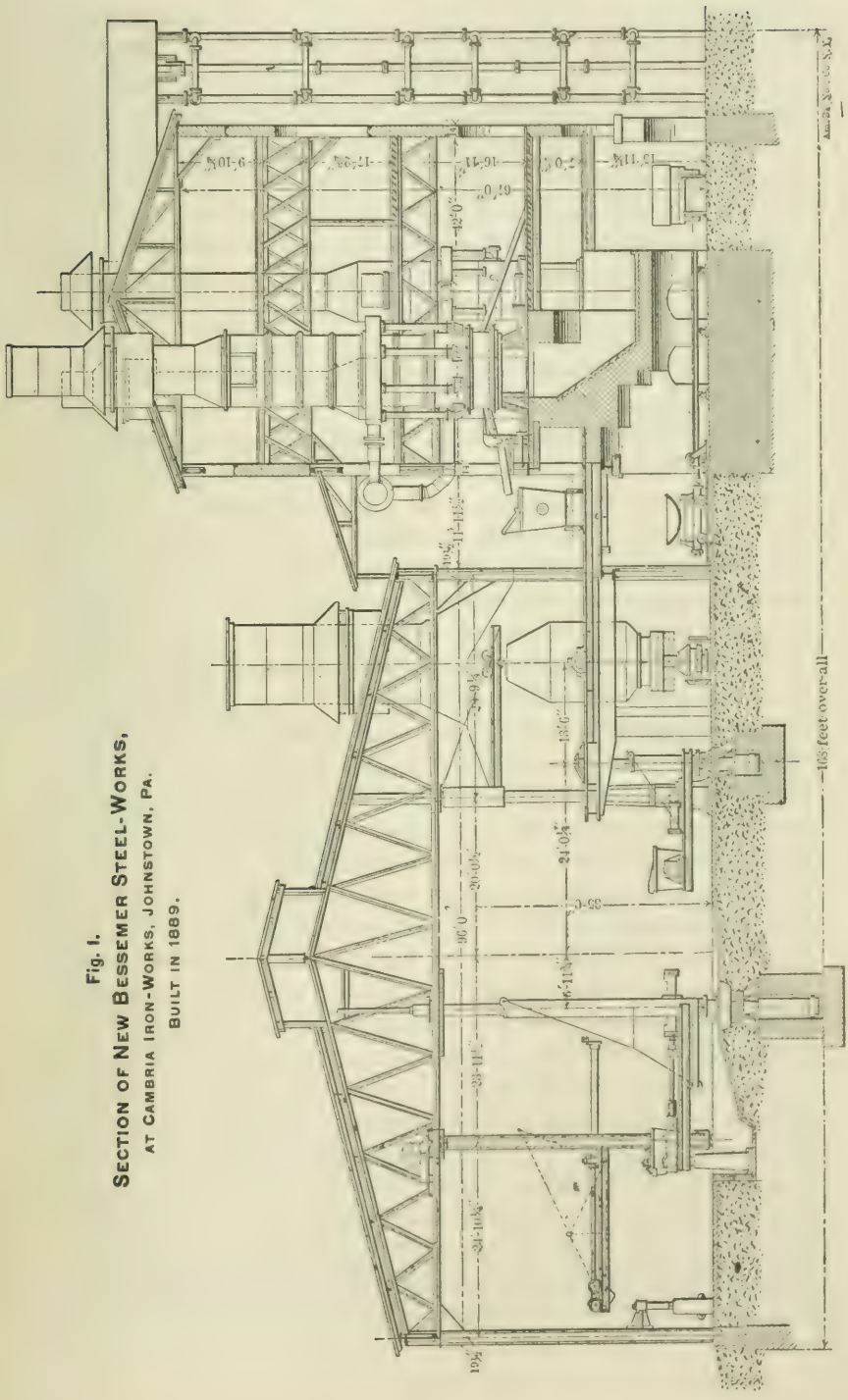

Fig. 1.
SECTION OF NEW BESSEMER STEEL-WORKS,
AT CAMBRIA IRON-WORKS, JOHNSTOWN, PA.
BUILT IN 1889.

But does it all pay? Excessive pace kills both horse and man. Remember the hare and the tortoise. He who works fourteen hours a day, seven days a week, and fifty-two weeks a year, so shortens his working-life that he accomplishes less than the more moderate worker. *In medio tutissimus ibis.* You Americans go at such a pace that you wear out your lives and your machinery. Your large outputs cost you more per ton of ingots than our small ones.

Such similes, such rebukes are familiar to most American metallurgists who have visited European works. While there may be a certain residuum of truth in them, there is certainly an incomparably greater mass of fallacy.

Effect of Rapid Running on the Machinery.—Let us consider first the machinery; next the men.

Realize, first, that greater output does not necessarily imply that any of the machinery moves faster; it may be, and in great part is, due to shorter intervals of rest and disuse. It means more nearly continuous, rather than swifter movements. The frequent and long repose that animals need, "that knits up the ravelled sleave of care," machinery needs not, and is not benefited by. It is not true that an engine which runs continuously from Monday morning to Saturday night will necessarily make fewer revolutions before wearing out than one which makes but one-half or one-third as many like revolutions at like speed each week. Indeed, some might go farther, and, pointing to the settling of foundations, the accumulation of grit on bearings, the rusting which occurs during rest as well as during work, and to the stresses set up by heating and cooling, expansion and contraction, each time the engine is started and again set at rest, might claim that, given occasional intervals for alignment and current repairs, the continously-running engine should make more revolutions before wearing out than the one used intermittently.

Again, faster blowing implies delivering more blast per second; but it does not imply that the engines must run faster, for it may be attained by making them larger.

Holding thus, that our system of enormous output, from works intelligently designed for it, implies no necessary waste or injury to machinery, I must admit that it tends strongly to induce more rapid movements of the machinery, which surely tend to shorten its life. But is not the tendency of modern engineering towards greater speed? Is it not thought that the saving in interest, labor and steam, outweighs the cost of the increased wear due to greater speed?

But I anticipate the argument. Finding deductively no strong

reason to expect that our larger outputs imply wasteful use of machinery, let us turn to induction, and see what the wear and tear of our machinery actually is.

The first American Bessemer works were built at Troy and Wyandotte in 1864. The Wyandotte works were abandoned in 1869; but one of their old cranes is still in vigorous activity at the Bessemer works at North Chicago, which for a time excelled all others in size of output. This archaic crane has, indeed, received a new plunger, but has undergone no other renewals, and, after more than a quarter of a century of our American high-pressure system, does two-thirds of the work of its mill.

The Troy works, burned down in 1868, were rebuilt in 1869, still in an experimental way. The subsequent doubling of the weight of the charges has necessitated changing most of the machinery; but the old blowing-engines are still in constant use.

The same cause, increasing the weight of the charges, has led to the renewal of much machinery in most of the older works. But at the North Chicago Bessemer works, started in 1872, the original cupolas (both for cast-iron and for spiegeleisen), vessels, cranes, ladles, pumps, and engines, are still in service without renewals, save the single crane-plunger already noted, and the center-section of each vessel. The bodies of the iron cupolas have been repaired, but the rest of the cupolas have needed nothing but an occasional coat of paint. All the original ladles are not only in use, but promise yet to do as much work as they have already done.

The Cambria Bessemer works were built in 1868–69. The annual output has grown from 30,000 tons in 1873 to over 120,000 in 1880, and 210,000 in 1888. They long disputed with North Chicago the palm for great output. The greatly increased output and weight of charge have led to the abandonment of the original antiquated cupolas and of the ladle-crane; to enlarging the bottom-ovens, the stock-hoist, and the end-sections of the converters; and to erecting additional boilers and engines. While the original ladles have been worn out, the original ingot-cranes, boilers,* pressure-pumps and blowing-engines are still serviceable and in use. The original reverberatory spiegel-furnaces, pressure-pumps, and fan-blowers have given place to cupolas, compound pumping-engines

* The original boilers have been retubed; the old pressure-pumps are now used as boiler feed-pumps; and the old ingot-cranes have been strengthened to meet the fourfold increase in the weight of the ingots, from about 1500 to 6500 pounds.

and Baker blowers. But for the ladles, one might say that all the original permanent plant was still in use, except what had become unbearably antiquated, and what could not be used because of the greater weight of charges—in short, except only what was outgrown.

So far as my inquiries have gone, in the great majority of cases, machinery has been replaced, not because worn out, but, though still fit for the work for which it was designed, either because it was unequal to the enormously and unexpectedly increased duties due to increased weight of charge, or because it could be replaced advantageously by more efficient machinery of later, or at least better, design. In this day of rapid mechanical progress, it must often happen that machinery becomes antiquated long before it is worn out. The mammoth office-buildings erected in my boyhood, with all the then modern improvements, are torn down to-day, not because worn out—they would stand for centuries—but solely because they are unsuited to to-day's needs. To the eye of progress we are still in the middle ages.

Now the length of time needed to render a machine antiquated, and to justify replacing it with a better, depends not on how much it works, and how much it rests each day, but on the progress of the rest of the world.

Suppose that we have an engine which can make just one billion revolutions before wearing out. If mechanical progress be such that it compels us to displace this engine at the end of ten years, the proportion of the engine's first cost chargeable to each ton of product will be only one-tenth as great in case it has then already made its billion revolutions, as if, thanks to leisurely interruptions, its average work has been at the rate of one billion revolutions in one hundred years. If a machine is to be thrown out before it can wear out, the charge for its cost against the unit of its useful work is inversely proportional to its average rate of work.

The same is true of the charges for interest, insurance and taxes; while the fact that general charges, such as those for sale, advertisement and collection, diminish with the volume of business, need not be dwelt on.

The Report of the United States Commissioner of Labor gives the cost of repairs and supplies at a single American rail-steel Bessemer works (and a slow-running one at that, making only 479 tons daily), as $0.453, against from $0.541 to $1.52, or an average of $1.07 for eight British and Continental rail-steel Bessemer works,

three of them basic, five acid.* But such comparisons must be received very cautiously, owing to differences in book-keeping.

In the only case in which I have exact trustworthy data, the actual cost of repairs to an American Bessemer converter-department, including cupolas and ingot-locomotives and cars, is $0.174 per ton of ingots.†

To sum up, our system of large outputs then, on the one hand, certainly lowers the charges for installation, interest, taxes and insurance; on the other hand, the actual history of the machinery itself, and the actual cost of repairs in some of our oldest and most rapid works, by no means suggest that our high-pressure system causes excessive wear and tear, or that it lessens the grand total life's work of the machinery and plant. And this is easily understood on remembering that great output is due rather to shortened intervals of disuse than to rapid motion.

Effect of Rapid Running on Labor.—So much for the machinery. Turn we now to the men.

Larger outputs imply a larger number of men, without materially increasing the number of kinds of operations to be performed. Hence each man should become more skilled in the smaller range of operations to which his work is restricted. These he should perform not only more quickly, hence accomplishing more useful work with a day's labor,‡ but with greater economy of power, of the fuel or other material which he uses, with less wear and tear of tools and machinery, and in general better, with results fitting more closely the general needs, and so lightening the labor of his co-workers, and lessening the needed supervision and instruction.

Or, if the sum of his useful work be not increased, its smaller range should permit us to employ less intelligent and costly men. In either case we have here a first gain.

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* *A Preliminary Report on the Cost of Production of Pig-Iron, etc.* By the Commissioner of Labor, Washington, 1890, pp. 31-33.

† The cost of such repairs consists chiefly of the wages of machinists, blacksmiths, etc. In the neighborhood of these works, laborers are paid $1.68 per twelve-hour shift; enginemen suitable for small locomotives, $2.50; enginemen for large engines, $3.25; blacksmiths, $2.50; handy-men, attending to current repairs to machinery, $2.75.

‡ Let us take as the unit of work a day's labor, that is to say, the labor of one man for one shift. It varies in value and cost, just as coal does; still, like the ton of coal, it is a better unit than the dollar's-worth, since it is free from one source of variation, the difference in the money-value of like kinds at different times and places.

Next, as operations succeeded each other more rapidly, so should a smaller proportion of the workmen's time be lost between them, if not in rest, at least in slackened activity.

Again, we cannot employ fractions of men. Where given work is two and a quarter times as much as one man can do, we must actually employ three. Here we have what I may call a *fractional excess* of three-fourths of a man. Now it should be clear that when, as in rapid running, many men are employed at a given task, this fractional excess tends to bear a smaller ratio to the whole, and hence to the output, than when few are employed.

Finally, there is a certain nucleus of skilled men—blower, chemist, blacksmith, timekeeper, foreman, etc.—whose wages resemble the fixed charges of the railway more than do those of the other workmen, which rather resemble the working-charges. I do not insist on an exact, but merely on an approximate resemblance. Now, clearly, the greater the output the less heavily do these quasi-fixed charges fall on the unit of product.

It is not easy to see how rapid running should increase, to any considerable extent, the work to be done. At first sight one might suppose that there would be more work connected with the moulds, since in rapid running they must be specially cooled, while in leisurely work they cool soon enough spontaneously. But even with the enormous output at Adonde, two boys on each twelve-hour shift suffice for cooling the moulds. The labor connected with setting and stripping the moulds is no greater when they are artificially cooled between leaving and returning to the pit, than when they return to the pit immediately after leaving it. In either case the same number of operations is performed, and in the same way. The mould must be lifted from the ingot, set down, left while its ingot is taken from the pit, and then returned; in one case after cooling, or even after an excursion on its car to the mould-yard and back, in the other case immediately. In either case the ingot should be stripped as soon as possible, so as to save its initial heat; hence the trouble from temporary stickers should be the same in either case.

Next, when the intervals between heats are short, the vessel-linings and bottoms have little chance to cool; hence they should wear out more quickly, or at least the labor needed to keep them in order, by patching, blinding tuyeres, etc., should, it may be claimed, be greater than in case of more leisurely running. A partial answer to this is that the short intervals permit the use of low silicon, and

hence short blows, and that the destructive effect of each blow must increase with its length.

A better answer, however, is that actually the number of days' labor of the vessel-men per ton of ingots, is less in swift-running American than in European Bessemer works, at least so far as my investigations go; indeed, in the case of this group our labor-consumption compares exceptionally well with the European.

Again, it may be said that to move objects more rapidly calls for a greater expenditure of strength for each motion, and hence ultimately for a greater labor-consumption, by calling either for a shortened shift or for additional help. In the case of most of the operations connected with the converting-department, however, greater output does not require that any one manual operation be performed more quickly, but merely that one succeed another more quickly. Double the quantity of iron to be melted, and you must double the number of stockers and chargers, each of whom will still move at the same rate as before.

But where, as in the case of the cranes, the number of instruments available is limited, then, indeed, greatly increased output, while in part due to shorter intervals of rest, may well lead to more rapid movement. This objection to large outputs is met, however, by using mechanical appliances for rotating the cranes, as well as for lifting them. When this is done, and in many other cases, those of the engines, pumps, etc., in which the work is done by machinery, the workman simply directing, more rapid movement merely means that each workman directs more useful work per shift.

This question, then, whether increased output, in that it tends to induce more rapid movements, really calls for a considerably greater consumption of labor per unit of product is one which each must answer for himself; but no valuable opinion can be formed without great familiarity with the work in hand, and careful observation. Personally, I am confident that it does not; but that, by compelling us to keep the machinery in the best order, and by leading to the adoption of better mechanical appliances, it eventually lessens the labor-consumption.

In considering this question, those inclined to the opposite view must bear a further distinction in mind. In order that the greater speed of the working shall actually cost more, it must be so extremely excessive as to lessen the amount of useful work which each man accomplishes in a working day. Proof that the pace was such

as to shorten his years of physical vigor, and thus lessen the amount which he can accomplish in his lifetime, would, I apprehend, count for little with directors and managers. The workman is paid, not for his lifetime, but for the number of days which he actually works, and at a rate which is governed chiefly by the law of supply and demand, by the attractiveness of his occupation, and perhaps by the endeavors of the trade-union—a point that we need not discuss—and but remotely and slightly, if at all, by the probable number of his working-years. As humanitarians, we might regret such diminution of the gross sum of the workman's working-power. As managers, hired by others to make money for them by metallurgy, we would be no more justified in diminishing our employer's profits by alloting to the workman less work than he willingly and knowingly, without deceit or pressure on our part, sought or accepted, than we would be in substituting machine for hand-puddling, knowing the former to be less profitable, or in shortening the puddler's shift to four hours. The workman is a free agent.

But you will find no signs of unwholesome exhaustion in our Bessemer works. Go there unobserved; the cheery tones, the passing jest which you will hear, predict no early decay. No; in visiting our works, I recognize the faces which I knew there twenty years ago, still full of force.

Leaving now the quantity of labor, note that the average degree of intelligence needed, and hence the average cost of a day's labor, should be very much lessened by rapid running. One skilled chemist, one skilled mason, you must have, be your output large or small. One blower, one skilled engine-man is needed, and suffices, whether you blow one or one hundred heats per shift, whether your engine runs continuously or but a tenth of the time. The difference is that, in case of small output, much of the time of these men, which you must pay for in full whether you use it or not, is either wasted or devoted to work which, in the case of larger output, is performed by less intelligent, less skilled, less costly men. Shorten your blows, shorten your intervals, double your output, and you hardly need more of these highly-skilled men. Most of the additional men are less skilled; many, indeed, are but assistants of the skilled nucleus which is as necessary, and nearly as large, in the case of small as in that of large output. And though the chemist or engineman who is to direct others must add executive ability to the qualifications he would otherwise need, and hence commands higher pay, yet the extra expense thus caused should raise the average

cost of the daily wages much less than it is lowered by the employment of the large number of relatively unskilled assistants, due to the greater output.

There are many trivial and very simple duties, which call for little intelligence. With large output, those of each kind recur so often that special, cheap men can be fully occupied with them. The expensive men, in cases of small output, do these trivial simple acts with their costly labor, which, in cases of large output, is restricted to the difficult tasks which require it, and which fully occupy it.

Or if, justified by the greater output, on the unit of which their pay forms a relatively small charge, you employ additional skilled and costly men, they are not mere duplicates of those you had before; they bring a different skill and additional knowledge to your aid, permitting economies, and devising improvements otherwise unattainable.

To sum this up, the greater skill due to the smaller range of duties; the shorter slack intervals between successive repetitions of the same work; the smaller ratio of fractional excess and of fixed charges, to all of which an increase of output should lead, should lessen the quantity of labor per ton of product. And while more rapid work might at first suggest more rapid, and hence more fatiguing, motions on the part of the men, and thus an increase in the labor-consumption, on closer view it is seen rather to mean more men, each moving at the same old rate, improved mechanism, and fuller substitution of machine for muscular power.

Further, increase of output, causing us to employ many relatively unskilled assistants to the costly skilled nucleus, should lower the average cost of a day's labor; or, giving us additional kinds of skill and knowledge, should lead to economies and improvements.

Turning now from deduction to induction, I must admit that owing to the entanglement of causes, it would be difficult to prove by actual numbers that any saving in the consumption of tools and materials was due to the increased specialization of labor, rather than to other co-existing causes; and, further, that I have not at hand data sufficient to prove directly that the greater output permits a lower average degree of intelligence, and thus a lower average price of a day's labor. But it is certainly very striking that, in the Mugby pay-roll in the Appendix, the rate of pay of 52.8 per cent. of the workmen indicates that they are wholly unskilled, mere laborers set at hard work. Their highest rate of pay is only 14 per cent.

above that of the very cheapest men, the mere sweepers. This 52.8 per cent. includes less than half the stockers, none of the cupola-chargers, and none of the thirty-four pitmen proper.

Making a like study of the pay-roll of a Continental establishment in 1881, I find that only 13 per cent. of the workmen can be classed as laborers, placing in this group all who do not receive more than 14 per cent. higher wages than the mere sweepers. This 13 per cent. includes one of the runner-men, an engine-tender, and all the blacksmith's helpers; indeed, quite as high a class of men as I include in this group at Mugby.

On the one hand, a single case, especially when at so remote a period, does not offer convincing evidence; on the other hand, the establishment is one of the most famous, and reputed to be one of the best managed, on the Continent.

Sir Lowthian Bell * gives the wages of British and American cupola and vessel-men, ours being from 67 to 160 per cent., or on an average 123 per cent., higher than the British. Yet the average wages of all the men at the less slow of the two American mills which he quotes is only 30 per cent. higher than at the British, which certainly seems to imply that a much higher proportion of our men than of the British are unskilled.

If we take the average pay of laborers about the British Bessemer works as 70 cents, then the average British Bessemer day's wage of 5*s.* 11¾*d.* given by Sir L. Bell is 113 per cent. above that of laborers, while at Mugby Junction the average daily wage is only 64 per cent. above that of the lowest grade of laborers, a fact which strengthens the suggestion of the preceding paragraph.

Turning now to the quantity of labor, *i.e.*, to the number of days' labor per ton of ingots, I must first insist on the extreme caution needed in making comparisons. Not only does the quantity of labor needed vary greatly with the method of working, wholly apart from its swiftness, but the limits of the truly Bessemer labor—the limits of the converting-department—vary greatly in different establishments.

The consumption of labor may be purposely increased, if a more than equivalent money-saving can be thereby effected in some other respect. Witness the adherence and even the return to the use of cupola-metal in this country. So, too, it may, and often does, pay to employ men who are not absolutely necessary, that they may pro-

* *Principles of the Manufacture of Iron and Steel*, 1884, p. 573.

long the life of the moulds, bottom-ladles, etc.; or that they may prevent the cracking of ingots, promote regularity, save metal, etc. Liberality in this respect is wise.

Next, the limits of the converting-department are a matter of book-keeping; no natural exact limits exist. In comparing the nominally full lists of all the converting-department men given me by some European managers, whose works I have lately visited, with the Mugby Junction pay-roll, I find that from 32 to 38 per cent. of the men classed at Mugby Junction in the Bessemer department are in other works classed outside it. And if we would compare the pay-roll at Mugby Junction with those of some foreign mills using direct-metal we would have to exclude in one case 52 per cent., in another 59 per cent., of the Mugby men.

Among the doubtful classes, included at Mugby, but excluded from the Bessemer department at many other works, are:

1. Those bringing iron, coke and refractory materials from the stockyards or cars into the steel-works.
2. The superintendents, weighmasters, clerks and time-keepers.
3. The switchmen and the locomotive-men, moving moulds, ingots and bottoms.
4. The men engaged in crushing and roughly preparing the refractory materials.
5. The men at the drop.
6. The enginemen and stokers.
7. The blacksmiths.
8. Those carrying cupola-slag away from the Bessemer department.
9. The machinists and others engaged in current repairs.
10. Those engaged in repairs and other work on Saturday night and Sunday.

With this caution I submit the following statements concerning the labor consumption at American and foreign mills:

In 1876, when our output per pit per 24 hours was about 225 to 250 tons, about 0.67 day's labor yielded a ton of ingots at American Bessemer mills, according to Kupelwieser's data, or 0.64 day's labor in the case of the Edgar Thomson works.*

* Trasenter, "*L'Industrie Sidérurgique aux États Unis d'Amérique*," 1886, pp. 232–236.

In 1879, the number of days of labor per ton of ingots at one British Bessemer works was 9.47 per cent. greater, and at another 32.14 per cent. greater, than in current American practice, according to data furnished by Holley.*

In 1880, comparing the United States census returns and the Belgian mineral statistics, Trasenter finds that the annual output of finished product per workman was:

In the United States (Bessemer and open-hearth processes), 82 tons.

In Belgium (Bessemer process), 48 tons.

76 per cent. of the American and 72.5 of the Belgian output was rails.

According to data collected by Trasenter, apparently in 1884, the number of days of labor per ton of ingots in Bessemer works was as follows:

	Direct-Metal.	Cupola-Metal.
American:		
Works A, direct-metal	0.41	...
Works B, direct-metal, stokers, blacksmiths and refractory-materials-men excluded	0.267	...
Works C, cupola-metal	...	0.476
Works D, cupola-metal	...	0.41
Belgian:		
Direct-metal	.3–.4	...
Cupola-metal	...	0.4–0.5

In 1884, Sir Lowthian Bell instanced a rather slow and a very slow American Bessemer mill, in which the output per day's labor was 39 per cent. and 9 per cent. less, respectively, than in current British practice; and he seemed inclined to think that our swift work called for great expenditure of labor, though apparently meaning to imply to the reader that these were but impressions, and that his data did not warrant decided opinions.†

As the works which he quotes were both slow ones for America, making respectively only 117 per cent. and 65 per cent. of the current British output, their evidence does not bear on the present comparison, which is as to the effect of our very swift running on the consumption of labor.

During the spring of the present year I obtained from the man-

* Reports to the Bessemer Steel Company, No. 2, 1880, p. 21.

† *Principles of the Manufacture of Iron and Steel*, 1884, p. 572.

agers of three British and one Belgian Bessemer works which I visited, what purported to be complete lists of their Bessemer men. They were given from memory, and pretty broad experience has shown me that, under these conditions, the manager seldom remembers all the men; at least, he is almost certain to exclude many who lie near the limit of the converting-department, and who are included in the Mugby list.

Throwing out from the Mugby list those men whose duties are quite clearly excluded from the European lists, I find that the time of labor per ton of ingots at the European works exceeds that at Mugby as follows:

	Per cent.
At Works A by	0
At Works B by	17
At Works C by	45
At Works D by	11
Average,	18.25

Looking a little further, I find that the difference lies just where we should expect it, in those classes of men whose time is likely to be incompletely occupied in leisurely practice. Such are the vessel-men, much of whose time may be wasted when the blows and the intervals between them are long, the enginemen, the blowers and the box- or stage-men. On the other hand, in those classes whose time can readily be fully occupied, even in leisurely practice, such as the stockers, the cupola-chargers and the pit-men, the economy effected by rapid running is less conspicuous.

At the very fast-running American Bessemer works at Middlemarch, in which over 47 per cent. of the *employés* are merely laborers, the labor-consumption during a month of exceptionally great output was only 0.374 day per ton of ingots, when using cupola-metal. In four consecutive months, when using cupola-metal, it was only 0.403 day. In both cases all the doubtful classes already enumerated are included, and the data are most trustworthy. The first is below, the second barely reaches, the lower limit claimed by Trasenter for Belgian cupola-metal practice; and my observations in Belgium indicate that a much smaller proportion of unskilled men is employed there than here, and that the usual limits of the converting-department there exclude many men included at Middlemarch.

Turning back to three complete pay-rolls from European Bessemer works for 1879, I find that the labor-consumption per ton

of ingots then exceeded the present consumption at Mugby as follows:

	Per cent.
At Works C (the same works as that styled C just before) by .	. 65
At Works E .	. 71
At Works F . .	. 130
Average, . . .	88.7

I here add some data as to the cost of labor per ton of ingots, though they cannot well be applied to our present question, for lack of definite information as to the relative daily wages.

Estimate of Cost of Bessemer Labor per Ton of Ingots.

America:	
1884. Trasenter, ...	$1.00–1.75
1889. Jeansville, cupola-metal, very rapid running, *all doubtful men*, smiths, refractory-men, stokers, etc., and all laborers included; the author, . .	1.185
1889. Semitopolis, very rapid running, same classes as for Jeansville included, direct metal; the author, ...	0.95
1889. Report of United States Labor Commissioner, rail-steel, ...	1.522
1889. Report of United States Labor Commissioner, nail-steel, ...	1.923
Great Britain:	
1872. Snelus, ...	$5.11
1879. Holley, repair-labor included, direct-metal, . .	.82
Holley, cupola-metal, ...	.73
1886. Snelus, ...	.75
1888. United States Labor Commissioner, . . .	.56
1889. United States Labor Commissioner, . . .	.65
The Continent:	
1879. Holley, half direct-metal, ...	$.67
1889. Direct-metal, United States Labor Commissioner, .	.44
" Cupola-metal, ...	.85

To sum this up, the evidence offered goes to show that the average grade of intelligence needed for rapid running is lower than for leisurely work, agreeing with the results of our *a priori* reasoning. The data given by Holley agree with mine by indicating a decidedly smaller quantity of labor per ton of ingots in American

than in European practice; those given by Trasenter on the whole point in the same direction; those given by Bell point in the opposite direction, but they do not represent our characteristic rapid practice, nor is there evidence that the necessary steps have been taken to make his data comparable among themselves.

From the nature of the case I must expect that many, especially among those with whom prepossession is nine points of belief, will think my evidence far from convincing. If they are familiar with Bessemer practice, and have access to the detailed pay-rolls of Bessemer mills, I ask, or if they prefer, challenge them to compare these carefully with that of Mugby, and to give the world the results as fully as I have done.

Quality: Is it Influenced by Fast Running?—Finally, we are told that we sacrifice quality to quantity. There are intelligent and otherwise rational men, safely entrusted with important interests, who go so far as to insist on a necessary relation between quantity and quality, as logically necessary as that between the velocity and pressure obtainable from given energy. These lesions of the power of ratiocination, producing these isolated flings of a mental St. Vitus' dance, and reminding us of the sedate oriental engaged in calculating eclipses of the moon while he firmly held that they were due to the bite of a dog, should interest the alienist, but are beyond the scope of this paper.

Now, what are the conditions in the Bessemer process on which the quality of its product depends? The proportion of phosphorus and sulphur, of manganese, silicon, and carbon in the steel; homogeneousness due to mixing the recarburizer uniformly with the blown metal, and to preventing segregation; freedom from external cracks, and from an excessive quantity of blow-holes.

Phosphorus and sulphur we, of course, leave out of the question. All of these elements that the charge entering the vessel contains will concentrate in the steel, no matter how the process be conducted.

The proportion of carbon, silicon, and manganese depends on the accuracy with which the blower interrupts the blow, on the temperature of the blow, on the accuracy with which the recarburizer is weighed, and on the uniformity as regards oxidizing conditions with which it is heated or melted.

Actually the blowing is accurate enough. The proportion of carbon is as nearly constant as the needs of the case require. When greater accuracy is required, as in making soft steel, it is attained without appreciable delay. In a study of the uniformity of our

Bessemer practice a few years ago, I found that at some of our rapid works the greatest deviation of the carbon from the average, in a large number of consecutive heats in a period selected at random, was 0.11 per cent. *

The temperature of the blow is extremely important, affecting not only the composition of the steel, but the soundness of the ingots, and perhaps their homogeneousness. It depends not alone on the composition of the cast-iron treated, and the rapidity of blowing, but on the initial temperature of cast-iron and vessel. Of these the initial temperature alone is likely to be affected materially by the quantity of the output. The variations in this temperature are due chiefly to variations in the heat lost from the vessel between the blows, and on that lost on the cast-iron between the time when it is melted and the time when the blow begins. It seems to me that these variations are likely to be less when the intervals during which they can occur are extremely small, as in our practice, than when, as in more leisurely practice, considerable intervals occur. Indeed, the very fact that, thanks to the extreme subdivision of labor, each man's attention is confined to a very small class of acts, so that he is but a machine, repeating in endless monotony his few stereotyped duties, should foster uniformity. The weighing should be more constant, the conditions of melting the cast-iron, of heating the recarburizer, and of teeming, should be more constant than in leisurely practice.

I find it hard to understand how the rapidity of the blow itself should injure the quality of the metal; or how shortening the intervals between blows should injure it, unless by inducing slovenly casting. To obtain sound ingots, free from cracks and from excessive sponginess, we should teem with care. And here, indeed, one might reasonably fear that our great outputs would lead to unfortunate results.

But rail-steel lies so quietly in the moulds that nothing is gained by teeming slowly. Indeed, in view of the fact that a certain temperature but slightly above the freezing-point of the metal seems to give better ingots than either a higher or a lower temperature, and that the steel in the ladle necessarily cools rapidly, it is best to teem rapidly, in order that both the first and the last ingots of each heat may be teemed as nearly at this best temperature as practicable.

Turning again from deduction to induction, we do not find those

* *Transactions of the American Institute of Mining Engineers*, vol. xv., p. 347, 1887.

specific bad results which might be expected to follow too rapid teeming. The cracks and blow-holes to which careless teeming should lead, would in turn lead to physical defects in the rails, due to cracks; hence should result a large proportion of second-quality rails, of cobbles, of rolling-mill scrap. The proportion of second-quality rails is often less than half of 1 per cent. for a month at a time; and even the average of a year may not rise above this amount.* Making any reasonable allowance for our usually easy conditions of acceptance, this certainly indicates a small proportion of cracked or badly-teemed ingots, and reasonable freedom from sponginess.

Again, we are told that when operations are so rapid, following each other so closely, there is no opportunity to humor the individual peculiarities of each heat, whether in teeming or blowing. Whatever weight this consideration deserves, is, I think, overbalanced by the greater uniformity to which our conditions tend. We do not humor peculiarities, we prevent them. Man after man, each watching the few phenomena under his charge, knowing that he is but a tooth in a wheel, a link in a chain; that his failure to round his little sphere of duties true to the given pitch will so entangle the rest of the vast machine as to lead to prompt and severe punishment; man after man strives to keep the conditions entrusted to him constant. As each is charged with fewer conditions, so can each keep his charge more steady. The uniformity in the carbon-content already referred to is evidence of regularity; so is the small proportion of butt-ingots. In a monthly return before me it is only 1.09 per cent., though the output of the same month broke the record.

Does the fact that a pair of railway-tracks daily delivers 100 trains instead of two, imply any injury to the quality of those trains, whether as to punctuality, speed, equipment, safety, or comfort?

But how is it, then, that in literature, in art, in designing, beyond a very narrow limit, any increase in quantity of one's product necessarily injures its quality? Without trying to lay bare fully the causes of this difference, I may point to a sufficient and obvious one. The painting, the poem, the design depends for its excellence on the long-continued application of a single human mind, which may indeed be assisted mechanically by amanuenses, draughtsmen, etc., but

* In works using direct-metal the proportion is usually much higher than this. The yearly returns from one mill using direct-metal show 3 per cent. of second-quality rails; and I have good reason to think that it has been very much higher than this at some mills.

can share the essential work with others little more than the acorn can the assimilation and growth which yield the oak. Teachers, patrons, critics, help the one; the arborator and nursery-man the other. These matters demand the supreme effort of an organism which, from its very nature, cannot exceed a certain speed.

In other matters, as in rail-making, no one of the things to be done needs or is better for prolonged attention or long time-allowance. Neither rapidity of stocking, of melting, of blowing, of teeming, of carrying to furnace, nor of rolling need injure. Teeming should be swift; furnacing may be as long as you please, given enough soaking-pits. Despatching a hundred trains daily instead of two does not imply, but rather tends to prevent, slighting any important action; implies such organization that a vast number of simple acts, each capable of being done at once well and quickly, may succeed each other at brief exactly-timed intervals.

To what is the particular bent of our practice due? Jointly to the powers of an uncommon man, to favorable conditions, and to the favorable material with which he worked.

The mechanical aptitude which you and we inherit has probably been fostered by the isolation, the ungenerous soil, the inclement climate which the founders of this country braved. Sons of tillers of the wilderness, whose strange, harsh surroundings bade them invent or die, whose scanty tools had to fulfil many purposes, what wonder if mechanical aptitude be one of our strongest characteristics? What wonder if those mobile particles of the mother-mass which such forces as hope of religious freedom or of rapid amelioration of lot, impatience of native restraint, restlessness, sufficed to segregate from those whom the same forces failed to move, have formed, and, immigration continuing, are still forming, a markedly mobile and active race?

Favorable conditions were present in the enormous demand for rails, and in the very easy conditions of acceptance, which led the manager to concentrate his attention more on the quantity than on the quality of his product.

With such material and such conditions, a leader of rare powers, Alexander Lyman Holley, perhaps less by his uncommon mechanical skill than by his hold on the affections of others, and thus by his control over their course, started our Bessemer practice in the path which it has followed. A remarkable group of men, the Fritz brothers, Jones, Forsyth, R. W. Hunt, joined enthusiastically; traditions, standards, aims, were established lastingly; generous rivalry,

Fig. 2.
ELEVATION.

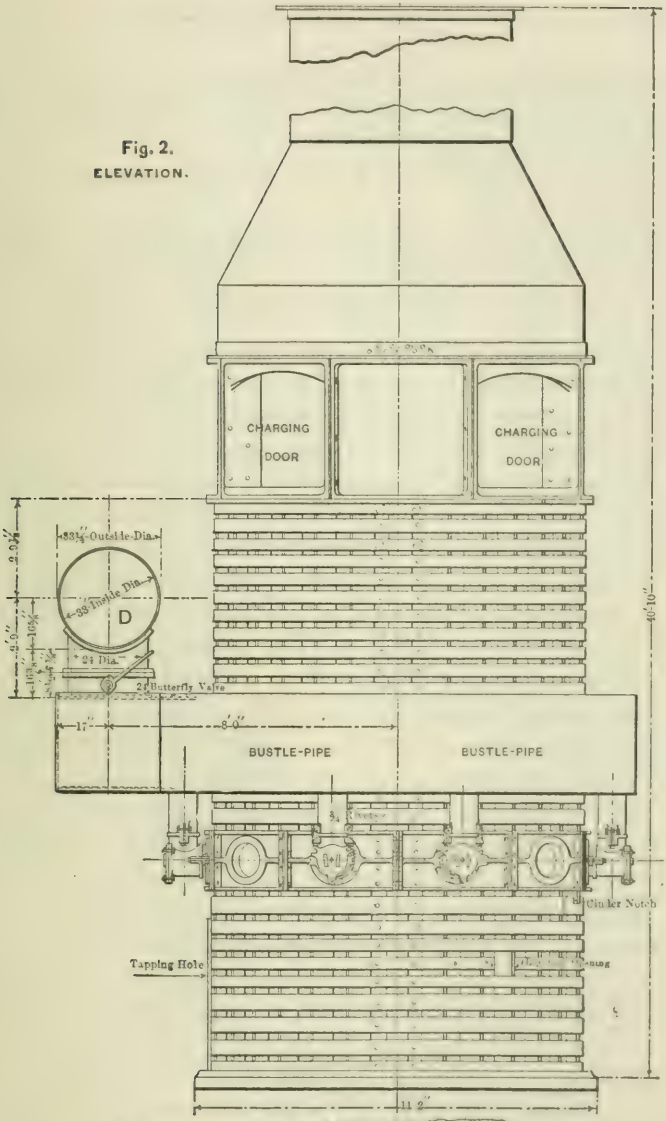
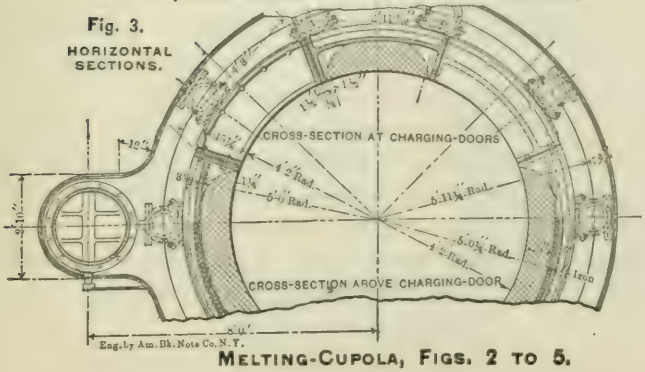


Fig. 3.
HORIZONTAL SECTIONS.



Eng. by Am. Bk. Note Co. N. Y.

MELTING-CUPOLA, FIGS. 2 TO 5.

esprit de corps, free trade in ideas, mutual help, quickness to adopt whatever was good in our neighbor's practice, whether cis- or trans-atlantic, willingness to give and take.

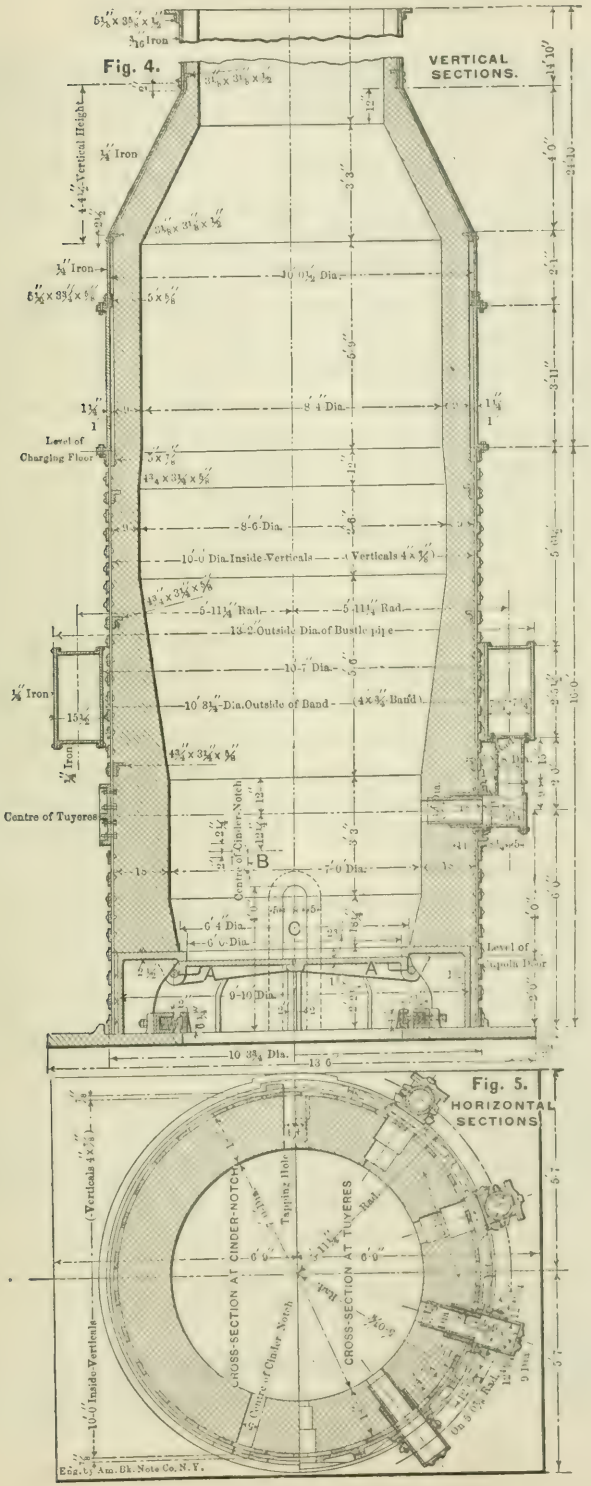
Beyond the features which I have thus discussed, the following deserve notice:

General Arrangement.—The Holley type of plant has usually been adhered to, with its broad shallow pit, served by at least three ingot-cranes, and with the vessels perched aloft. In the Forsyth modification of this, which several important works have adopted, the vessels stand back from the pit, and the casting-ladle is carried from vessels to pit and back over a short intermediate transfer-track. In this arrangement some 250° of the circumference of the pit are available for ingot-moulds, and four ingot-cranes can be used advantageously. Fig. 1 shows one of our latest and best Bessemer plants.

Cupolas.—The cast-iron is usually remelted in cupolas, which, with the exception of a modification introduced at Joliet and whose value is questioned, are very plain, but rather large, from 8 to 10 feet in outside diameter and from 14 to 24 feet high from the bottom inside to the charging-door. With cold-blast at about 10 to 12 ounces per square inch, usually delivered by fan-blowers, but in many works by the positive Baker blower, these cupolas sometimes run a whole week, but in case of very rapid work they are often dumped every forty-eight hours. Some of them melt as much as 16 tons an hour apiece, on an average of a whole month, with a fuel-consumption which occasionally falls to 1 per 15 of iron melted, or 6.67 per cent. The fuel-consumption for a whole month, when but little steel scrap is melted, is sometimes as little as 1 to 14.3 of iron, or 7 per cent. I am credibly informed that, as a *tour de force*, the coke-iron ratio has been for a whole shift as small as 1 to 20, or 5 per cent.

These large swift-running cupolas have the digestion of an anaconda. Even with this very small fuel-consumption, I have known an ingot weighing 3000 pounds, stuck in a mould weighing another 3000 pounds, to be charged in the cupola, causing no more derangement than if it had been a cambric needle.

This very small fuel-consumption is made possible probably only by the heat generated by the oxidation of the silicon of the cast-iron in the cupola itself. Indeed, as much as 0.5, or even 0.6 per cent. of silicon may be thus removed when the fuel-ratio is very small; but, with the common fuel-ratio of 1 to 12, or 8.33 per cent., only about 0.3 per cent. of silicon is removed in the cupola.



MELTING-CUPOLA, FIGS. 2 TO 5.

Figs. 2 to 5 show a lattice-bound cupola, such as has been built at some important works, but the plain boiler-plate shell is generally used.

In the newer works the ladle which receives the molten cast-iron from the cupolas is carried by a locomotive to the vessels, saving running-scrap and the labor of the runner-men.

Direct-metal.—The experience with direct-metal has not been such as to lead to the rapid abandonment of remelting in cupolas. The blast-furnace must be so burdened that the least siliceous iron made shall still have enough silicon to develop the high temperature needed at the end of the process, at least 1500° C., and that the most sulphurous shall be within safe limits. As the silicon varies greatly and capriciously, it results that some heats have a very great excess of silicon, and that the average proportion of this element must be higher than in cases of cupola-remelted iron. In addition to the disadvantages attending the use of iron rich in silicon already referred to, we have as additional ones sudden variations in the casting-temperature and in the proportion of sulphur, leading to increased proportion of second-quality rails, and increased consumption of manganese; variations in the quantity of scrap needed to cool the charge, and consequent increase in the proportion of butt-ingots; increase of converting-works scrap; skulling in vessels and ladles; inaccurate blowing; lessened output; greater consumption of steam; shortened life of bottoms, ladles, moulds, and superintendent. These, in most cases, have seemed to outweigh the obvious saving, which is estimated by different managers at from $0.50 to $1 per ton, and which is due to an important saving of labor at blast-furnace and cupola, of the coke needed for remelting, and of the loss of iron, silicon, and manganese which occurs in the cupola. Indeed, the greater loss in the converter in direct-metal practice goes far towards balancing the cupola-loss in remelting-practice.

All these disadvantages attending the use of direct-metal are due to irregularity in its composition. How great this irregularity is, and how much it is lessened by the use of Captain Jones' metal-mixer, is indicated in Table III. Note that the use of the metal-mixer at Edgar Thomson reduces the range of composition by 37 per cent. and 52 per cent., and lessens the "average deviation from the average" by 43 per cent. and 55 per cent., in case of silicon and sulphur respectively.

Here the mixer has been of great benefit. While it is certainly inconvenient to have even an occasional heat with only 0.68 per

cent of silicon, such heats can be blown, if heat follows heat quickly; while if the ladleful with only 0.264 of silicon had been run into the converter, it could have been got out again only by blowing it on its side, with heavy loss of iron. And it must be admitted that the

TABLE III.—*Uniformity of Blast-Furnace Work, as Deduced from the Composition of Successive Casts.*

	England.		Edgar Thomson	Edgar Thomson.			
				Successive Casts run into Metal-Mixer.		Same Successive Casts run out of Metal-Mixer.	
	I. Silicon.	II. Sulphur	III. Silicon.	IV. Silicon.	V. Sulphur	VI. Silicon.	VII. Sulphur
Date	Feb. 1890.	Feb. 1890.	June 1888.	June 1890.	June 1890.	June 1890.	June 1890.
No. of heats represented { Silicon }	29	...	85	108	...	96	.....
{ Sulphur }	...	29	...	...	107	...	106
Average percentage..... { Silicon }	2.20	...	2.04	1.336	...	1.186	....
{ Sulphur }	...	0.082	...	...	0.023	...	0.019
Maximum percentage { Silicon }	4.12	...	3.35	2.613	...	2.103	...
{ Sulphur }	...	0.149	...	...	0.099	....	0.048
Minimum percentage { Silicon }	1.40	...	0.71	0.364	...	0.684	...
{ Sulphur }	...	0.043	...	...	...	...	Trace.
Total range of composition { Silicon }	2.72	...	2.64	2.249	...	1.419	...
{ Sulphur }	...	0.106	...	...	0.099	...	0.048
Average deviation from average..... { Silicon }	0.328	....	0.497	0.409	...	0.235	...
{ Sulphur }	.....	0.002	.....	...	0.0193	...	0.0084

Note.—I have deduced these numbers from the composition of successive casts at the several works. In the case of columns IV. to VII. inclusive, the composition given represent 108 ladlefuls filled consecutively at the blast-furnaces. The metal was poured from these ladles into two mixers, and from these mixers it was taken to the converting-works, in ladles each of which was filled half from each mixer. Columns VI. and VII. represent samples taken while the iron was running from the ladle into the converter. The mixers were badly skulled at the time, and the manager, Mr. J. Gayley, thinks that later data would yield a more favorable showing. Note that the average proportion of both silicon and sulphur is lower, that of the silicon 0.15 per cent. lower, in the converter- than in the blast-furnace-samples. This has always been found true of silicon, but no previous comparison with regard to sulphur has been made. The removal of so large a proportion of silicon is most surprising.

average proportion of silicon is not much higher, nor is that of sulphur lower, than would be desirable in case the metal were to be remelted in cupolas; nor is the maximum proportion of sulphur so high as to necessitate using more manganese than would be needed

in cupola-practice. It certainly seems as if this simple and characteristic device of Captain Jones has turned the balance in favor of direct-metal practice; and it would be far from surprising if all our rail-steel-making Bessemer works were driven to using direct-metal.

Vessels.—The tendency has been to increase the size of the vessels. I remember no case in which it has been lessened, while in many cases it has been increased, from the old standard size of five, to seven, and later to ten, and even twelve and thirteen, tons.

The newer vessels are usually nearly concentric or symmetrical (instead of being retort-shaped) and very large, so that even with rapid blowing very little slopping occurs. The capacity of the newer vessels is from 50 to 65 cubic feet per ton of iron to be blown. With 65 cubic feet per ton there is little slopping; but much trouble from this source has arisen in cases of some new vessels with less than 55 cubic feet capacity per ton.

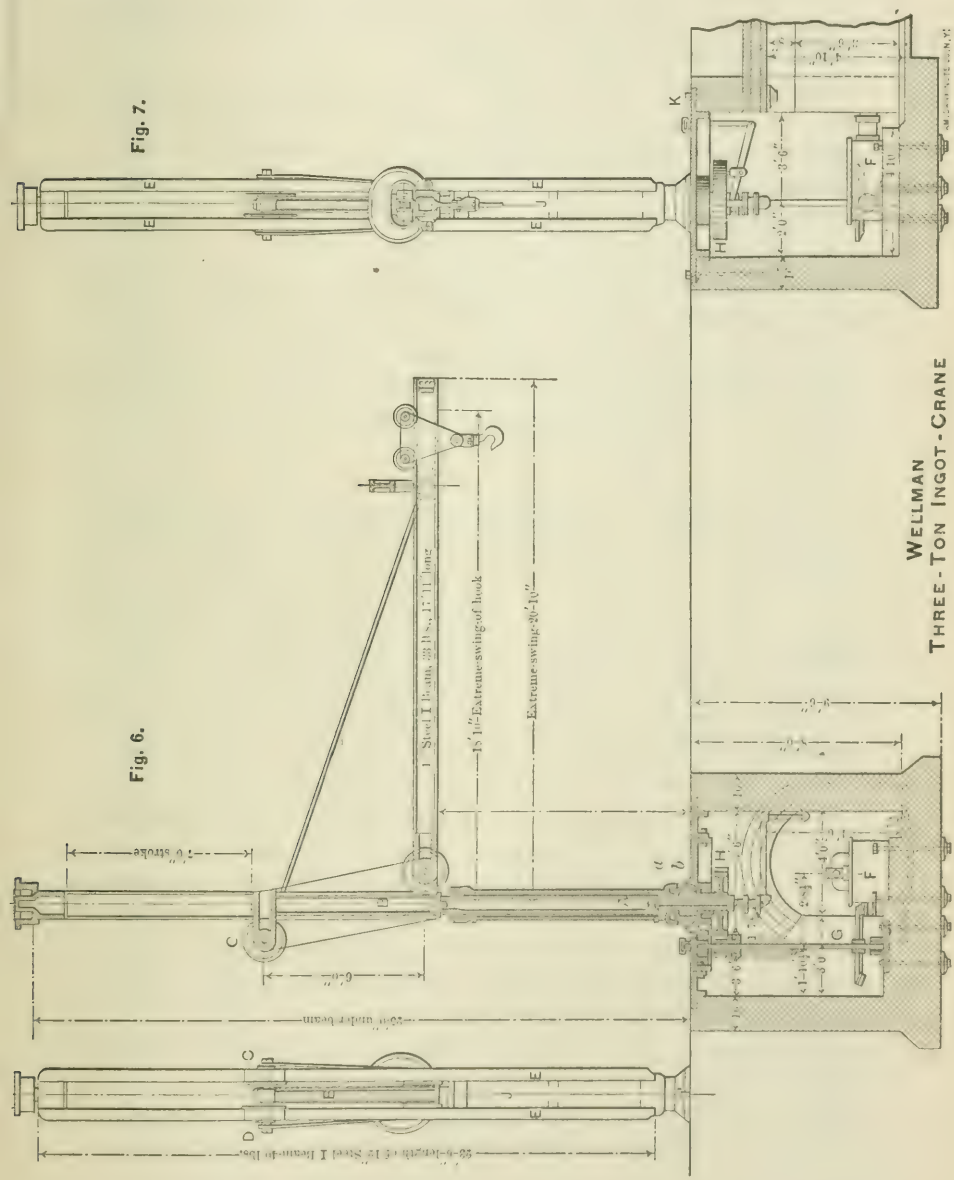
It has not yet been shown what residuum of real value for special conditions underlies the extravagant and often senseless claims made for the comparatively new patented modifications of the Bessemer converter, such as the Clapp-Griffiths and the Robert. They, or at least the Robert, should yield an unusually high temperature, thanks to the greater loss of iron, and perhaps to the fuller oxidation of the carbon; this would seem to recommend them for small foundries, where but few and small heats can be used daily. The same result may be reached with the common vessel by inclining it, though perhaps less conveniently than with these special vessels.

The vessel is almost always turned by a horizontal rack lying beneath a pinion on its trunnion. The lining is usually rammed in place, but good results have been obtained in the Pennsylvania mills with linings of blocks of mica-schist. The linings in many works are knocked out but once a year, at the time of the general repairs in December, but of course they are patched on Sundays. Thus a lining may make over 30,000 blows.

The bottoms are usually dish-shaped, and in many of them long shaped bricks or tiles, standing on end and each reaching completely through the bottom, are used. These increase the first cost, but also the life of the bottom, and facilitate drying. The bottoms themselves are usually rammed very carefully, be it with or be it without these bricks; the mixture used is very dry, barely moist and fat enough to be balled in the hand; and they are very carefully dried, often for 48 hours. In some works the average life of the bottoms is as

much as 28 heats, and single bottoms occasionally last more than 50 heats.

Fig. 7.



Blowing.—The moment for ending the blow is determined by the eye, and extremely rarely, if indeed ever, by the spectroscope. In

making rail-steel it is usual to blow "young;" that is, to interrupt the blow as soon as the first marked shortening of the flame occurs. This course shortens the blow, thus saving time and lessening the consumption of power for blowing, the loss of iron and the quantity of manganese needed for recarburizing.

In making soft steel the blow is, of course, prolonged beyond this point, in some cases till what is locally known as the "second change" appears; that is, till the reddish-gray smoke, which appears about the upper end or point of the flame when it first shortens, and which constantly grows more copious, can be seen along the whole length of the flame, and even at its very base.

The temperature of the blow, while of course controlled in large part by the addition of scrap-metal, is in many works further controlled by admitting great volumes of steam along with the blast. This simple trick prevents delays, facilitates making solid ingots closely of the desired composition, and reduces the proportion of butt-ingots to a minimum.

The Cranes are always top-supported.—Formerly they were usually of the Worthington or so-called Holley type; but of late years the incomparably better Wellman crane has come into general use. Quite a number of these have Mr. H. Aiken's modifications. In several mills these cranes are rotated and their trolley is moved radially by power. The rotation is effected by 3-cylinder hydraulic engines, by friction-clutches, and more rarely by racks and pinions. In some, a boy sitting on the crane controls all three movements—the vertical, the radial, and the rotary.

The chief advantage of the Wellman over the Worthington crane is its smaller water-consumption. This is due to two things—first, to the smaller weight of its rising parts; the mast may be light because it receives no transverse stress, this being transferred by the wheels C and D, shown in Fig. 6, to the steel I-beams, EE, which surround the mast. Again, the mast may be shorter, since it usually need not reach to the top-support. Second, the rise of the rising parts is less resisted by friction, again because the mast receives no transverse stress. This in the Worthington crane and the old Bessemer crane presses the mast against its gland or glands.

The Wellman crane, shown in Figs. 6 and 7, is rotated by a small air-engine, driven by air from the Bessemer blowing-engine, and controlled from the pulpit. The water-pressure used is rather low in most cases, but the tendency is now towards higher pressures.

Engines.—The blowing-engines are almost always non-compound,

and so are the rolling-mill engines. But, except where fuel is very cheap, this is admitted by many of the more thoughtful to be an error. Compound pumps, however, are used in many works.

The ingots are usually cast from the top, and in unsplit pyramidal moulds. In case of rail-steel they are usually 14.5 inches square, and usually weigh from 1.2 to 1.5 tons each; but at one mill they are 19.5 inches square, and weigh nearly 3 tons each.

The moulds stand in a wide, shallow casting-pit, served by three, and in some later works by four, ingot-cranes. The ingots are very rarely allowed to cool, but are transferred at once from the casting-pit either to common horizontal-hearthed heating furnaces or to Hainsworth pits. These may be likened to Gjers soaking-pits, but with auxiliary gas-firing. Few Americans have been able to grasp the advantage of the Gjers pit. Why we should tie our hands in this matter, and neglect to avail ourselves of that complete control over the temperature of the pit which auxiliary gas-firing gives, few of us can understand. When the ingots come hot enough to the pits, gas and air may be shut off; then the conditions of the Gjers pit are reproduced faithfully. In some of our mills the loss of weight in soaking and rolling, from ingots to rails, is reported to be only 1 per cent., and I have cases taken directly from the records of the mill in which the loss is only 1.5 per cent.

Moulds.—Some managers claim an average life of 100 heats for their moulds, and I believe that the average life for a whole year sometimes reaches 87 heats, when rail-steel only is made; 65 to 70 heats, however, is probably much nearer the usual average life, and in some mills it has been as short as 45 heats. A yearly return before me gives the average life as 70 heats. The common hydraulic ingot-extractor and the simple one of Mr. T. James are sometimes used; but many object to the former as likely to tear the ingots.

Ladles, etc.—For lightness, the ladles are always lined with loam-sand. In some works their life is only from 10 to 15 heats; in others it rises to 40 and even 50 heats. Stoppers and nozzles last from 2 to 5 heats each; in one mill the average life of the nozzles for a whole month was 7.11 heats. In another case the yearly return from a mill making an enormous output gave the average life of the stoppers as 1.9 heats, and of the nozzles as 6.25 heats. The consumption of refractory materials per ton of ingots was—

	Pounds.
Quartz,	22
Clay,	16
Sand,	21
Old bricks,	3
Loam sand,	30
	92

0.102 of a tuyere was used per ton of ingots.

Rolling-Mill Practice.—In rail-making the ingots are reduced to blooms about 7 inches square, usually in eleven passes, and almost universally in three-high mills, never under hammers. Cambria has a 48-inch two-high, and Bethlehem a 48-inch three-high, blooming-mill. In the latter the delay in lifting the piece 48 inches has been found considerable; 36 inches is a more common diameter.

The three-high mills usually have the well-known Fritz-Holley lifting feed-tables and ingot-manipulators. Efficient ingot-manipulators, designed by D. Jones and by Critchlow, are in use at two-high blooming-mills.

In some blooming-mills noted for their great output only two men are actively occupied; but to these must be added the engineer and spell-hands.

The blooms are further rolled in the older mills to single, in the newer to double or four-length T-rails, usually in eleven passes, and usually after, but in some mills without, reheating. The rail-trains are almost always three-high, usually in two, but exceptionally in three stands, and have two finishing-passes. The rolls are usually from 23 to 25 inches, but exceptionally 28 inches in diameter; while one two-high rail-train has a pitch-diameter of 32 inches. At some rail-trains with enormous output only five men are actively occupied. In addition there are the engineer and spell-hands.

The three-high rail-trains have ingenious and efficient appliances for giving the rail its four motions, rotating it about its axis, raising and lowering it, moving it parrallel with the rolls, and feeding it into them.

The plate-mills are usually of the three-high Lauth type, with rocking feed-tables, the upper rolls being raised by power, and in many cases by a pair of reversing engines mounted above the housings. Common dimensions for plate-trains are 31 to 32 inches diameter and 100 to 115 inches length. Such mills can roll 175 tons of plates of common sizes in twenty-four hours, with eight

men on each of the two shifts; say, one roller, one engineer, one screw-man, one feed-roll engineer, two hook-men, and two scale-men.

The universal armor-plate train at Homestead has 20-inch vertical and 32×60-inch horizontal rolls, driven by reversing engines 30 × 54 inches and 40 × 54 inches respectively, and can handle 48 × 54-inch 25-ton ingots. It has a 3000-ton hydraulic shear, served by two pressure-pumps with 65-inch steam and 10-inch water cylinders, and 8-foot stroke. Using steam at 175 pounds, they supply water at 4000-pounds pressure per square inch.

APPENDIX.

PAY-ROLL OF MUGBY JUNCTION BESSEMER WORKS.

DISTRIBUTION OF LABOR, 1890.

Summary.

Occupation.	No. of Men.
Stocking iron,	24
Stocking spiegel,	10
Stocking coke,	8
Stocking limestone,	6
Stocking bloom- and rail-butts,	8
Stocking scrap,	10
Scalemen,	4
Stocking patching, loam, sand, etc.,	3
Cagers,	14
Sweepers,	2
Stage,	15
Charging-floor,	37
Patching cupolas,	5
Melting-floor,	27
Vessel-floor,	29
Ladlemen,	22
Pitmen,	54
Cindermen,	29
Repair of bottoms,	11
Crusher-house,	31
Yard-labor,	25
General labor,	68
Switchmen,	6
Drop,	4
Steel-weighers,	2
Handling pig-iron,	. .
Boiler-house,	6
Blowing-engine room,	10
Fan-room,	4
Converting-building,	5
Blacksmith-shop for converting-works,	4
Small locomotives,	9
Crusher-house,	2
Superintendents, timekeeper, and clerk,	4
	498
General repairs, from time of stopping Saturday afternoon till time of starting Monday morning,	81

Distribution of Labor, 1890.

	Number of Men.				
	1st Turn.	2d Turn.	3d Turn.	Total.	Grand Total.
Stocking Iron.					
Men in charge	1	1	...	2	
Men loading in buggies for chargers	11	11	...	22	24
Stocking Spiegel.					
Men to pile spiegel on buggies	2	2	...	4	
Men additional to break charcoal-iron	3	3	...	6	10
Stocking Coke.					
Men to load coke on buggies	4	4	...	8	8
Stocking Limestone.					
Men to haul stone from end of run to coke buggies	1	1	...	2	
Men to break and load on buggies	2	2	...	4	6
Stocking Bloom- and Rail-Butts.					
Men to load butts for cupolas	2	2	...	4	
Men to load rail-butts for vessels	2	2	...	4	8
Stocking Scrap.					
Men to load up steel scrap	2	2	...	4	
Men to load up other scrap	3	3	...	6	10
Scalemen.					
Men to weigh pig-iron for cupolas	1	1	...	2	
Men to weigh spiegel, patching, loam, coal and general supplies for cupolas	1	1	...	2	4
Stocking Patching, Loam, Sand, Etc.					
Men to fill bins on cupola-floor with patching, loam, sand, etc	2	...	...	2	
Man to fill bins on vessel-floor	1	...	...	1	3
Cagers.					
Men placing loaded buggies on elevator, for the charging- and melting-floors	7	7	...	14	14
Sweepers.					
Men to clear stockers' runway, pick up coke fallen from buggies, and other obstructions	1	1	...	2	2
Stage.					
Steel-blowers	1	1	...	2	
Stage-men	5	5	3	13	15
Charging-Floor.					
Men to charge cupolas with iron, scrap, butts, coke, etc	1	1	1	3	
Others, regularly	4	4	4	12	
Additional men, when running fast, at over fifty-five heats	1	1	...	2	
Men to place empty buggies on elevator	6	6	...	12	
Men to charge spiegel	1	1	...	2	
Men additional	1	1	...	2	

	Number of Men.				
	1st Turn.	2d Turn.	3d Turn.	Total.	Grand Total.
CHARGING-FLOOR—*Continued.*					
Men to run elevators	1	1	...	2	
Men to grease buggies	1	1	...	2	37
PATCHING CUPOLAS.					
Man to patch cupolas	1	...	...	1	
Men additional	2	...	...	2	
Men additional to help patch cupolas	2	...	...	2	5
MELTING-FLOOR.					
Iron-melters	1	1	...	2	
" "	1	1	...	2	
" "	1	1	...	2	
" "	1	1	...	2	
" " (additional)	3	3	...	6	
Spiegel-melters	1	1	...	2	
" "	1	1	...	2	
" " (additional)	1	1	...	2	
Men to weigh molten metal	1	1	...	2	
Men to run elevators	2	2	...	4	
Men to sweep, pick up scrap and carry to pile.	1	...	...	1	27
VESSEL-FLOOR.					
Men to pour iron from ladle to vessel	1	1	...	2	
Men to pour spiegel from ladle to vessel	1	1	...	2	
Men to patch and line ladles	1	...	...	1	
Men to help patcher, carry coal to vessels and locomotives, take scrap away to pile, and clean up	2	2	...	4	
Vessel-men	1	1	...	2	
" "	1	1	...	2	
" "	1	1	...	2	
" "	1	1	...	2	
Runner-men	1	1	...	2	
" "	1	1	...	2	
Men for heavy scrapping	1	1	...	2	
Test-boys	1	1	...	2	
Men to sweep and clean floor	2	2	...	4	29
LADLEMEN.					
Ladlemen to set stoppers	1	1	1	3	
Men to be with ladle when pouring heat	1	1	1	3	
Men to patch and put in nozzles	1	1	1	3	
Men to help	1	1	1	3	
Men to clean nozzles	1	1	...	2	
Men to catch tests	1	1	...	2	
Men to help patchers	1	1	...	2	
Men additional in hot weather	1	1	...	2	
Stopper-maker	1	...	...	1	
Man to help	1	...	...	1	
Men at receiving-crane	1	1	1	3	
Men at casting-crane	1	1	1	3	28
PITMEN.					
Men in charge of pit	1	1	1	3	
Men in second charge	1	1	1	3	
Other pitmen	8	8	8	24	
Helpers	1	1	...	2	

	Number of Men.				
	1st Turn.	2d Turn.	3d Turn.	Total.	Grand Total.
PITMEN—*Continued.*					
Steel-pourers	1	1	...	2	
Men to swing ladle	3	3	...	6	
Men to swab moulds	2	2	...	4	
Men on dog-house	1	1	...	2	
Boys to cool moulds	2	2	...	4	
Boys to catch tests	1	1	...	2	
Men additional in hot weather, to help cranemen	1	1	...	2	54
CINDERMEN.					
Men to carry slag from vessels	1	1	...	2	
" " " " "	2	2	...	4	
Men to carry slag from slag pitmen	3	3	...	6	
Cupolas cindermen	4	4	...	8	
Men to keep spiegel slag running clear	1	1	...	2	
Cartmen	7	...	...	7	29
REPAIR OF BOTTOMS.					
Foreman bottom-maker for vessels (total tonnage)	1	...	...	1	
Men to make bottoms	1	1	...	2	
Men to help	1	1	...	2	
" "	2	2	...	4	
Men to line new ladles	1	...	...	1	
Man to help	1	...	...	1	11
CRUSHER-HOUSE.					
Men to mix materials for bottoms and vessels	1	...	...	1	
Men to help	5	...	...	5	
Men to mix for cupolas	3	...	...	3	
Men to attend to the grinding	1	1	...	2	
" " " "	2	2	...	4	
Men additional	1	1	...	2	
Men to attend to crusher	1	1	...	2	
Men to break quartz	1	1	...	2	
Boy to dry clay	1	...	...	1	
Boys to dry clay	3	...	...	3	
Men to attend to carts	6	...	...	6	31
YARD-LABOR.					
Foreman	1	...	...	1	
Men to handle and unload scrap, loam, tuyeres, clay, sand, spiegel, etc	21	...	...	21	
Men to pick scrap at dump	3	...	...	3	25
GENERAL LABOR.					
Foremen	1	1	...	2	
Laborers to change bottoms, clean shop, etc	15	15	...	30	
Men to handle hot ingot butts, change moulds, etc	10	10	...	20	
Rigger to repair building, clean stacks and repair buggies	1	...	...	1	
Man to help	1	...	...	1	
Men to clean under the vessels and carry scrap to pile	3	3	...	6	
Men to keep track clean from vessels to bottoms	1	1	...	2	

	Number of Men.				
	1st Turn.	2d Turn.	3d Turn.	Total.	Grand Total.
General Labor—*Continued.*					
Men to clean shop	1	1	...	2	
Men to grease buggies	1	1	...	2	
Men to clean tracks and coal locomotives	1	1	...	2	68
Switchmen.					
Men to attend the engine in handling hot and cold moulds, to carry scrap from rail mill to converting-department, and drop	1	1	...	2	
Men to attend locomotives in carrying hot ingots to rail-mill	2	2	...	4	6
Drop.					
Man to break up skulls, old moulds and scrap	1	...	...	1	
Men to help	3	...	...	3	4
Steel Weighers.					
Men to weigh steel	1	1	...	2	2

	Number of Men.			
	Day.	Night.	Total.	Grand Total.
Boiler-House.				
First firemen	1	1	2	
Second "	1	1	2	
Water tenders	1	1	2	6
Blowing-Engine Room.				
First engineers	1	1	2	
Second "	1	1	2	
Third "	1	1	2	
Fourth "	1	1	2	
Men at pressure pumps	1	...	1	9
Fan-Room.				
Engineers	2	2	4	4
Converting-Building.				
Men in charge of machinery	1	...	1	
Handy-men	1	1	2	
Greasers	1	1	2	5
Blacksmith Shop for Converting-Works.				
Blacksmiths	1	1	2	
" helpers	1	1	2	4
Small Locomotives.				
Engineers	1	1	2	
Engineer	...	1	1	
" handling ingots	2	2	4	
" " moulds	1	1	2	9
Crusher House.				
Engineers	1	1	2	2
Superintendents, Timekeeper, and Clerk.				
Superintendent	1	...	1	
Assistant Superintendent	...	1	1	
Timekeeper	1	...	1	
Clerk	1	...	1	4
Total				498

General Repairs from the Time of Stopping on Saturday Afternoon till the Time of Starting on Monday Morning.

Yard.	Number of Men.				
	1st Turn.	2d Turn.	3d Turn.	Total.	Grand Total.
Charging Floor (patching cupolas).					
First patcher	1	...	...	1	
Second "	1	...	...	1	
Third "	1	...	...	1	
Laborers additional	2	...	...	2	
					5
Melting Floor.					
Men knocking out skulls, relining same, and putting in tuyeres	3	...	...	3	
Men knocking out skulls from vessel runners and relining same	2	...	...	2	
					5
Vessels.					
First vesselman to patch	1	...	...	1	
Second " "	1	...	...	1	
Third " "	1	...	...	1	
Fourth " "	1	...	...	1	
Laborers to help	8	...	...	8	
Vesselman at night to light up	1	...	...	1	
					13
Tool-House.					
Tool-keeper by day	1	...	...	1	
Tool-keeper by night	1	...	...	1	
Men to watch in bottom-house, Saturday night	1	...	...	1	
Sunday	1	...	...	1	
And Sunday night	1	...	...	1	
					5
General Labor.					
Foreman	1	...	...	1	
Men to clean shop	4	...	...	4	
Laborers to clean dumps, clean shop, make ready to light up, help repair machinery, change moulds, and do whatever is necessary to put things in shape	45	...	...	45	
					50
Cleaning Vessel-Stacks.					
Rigger to clean the vessel stacks when clogged with cinder	1	...	...	1	
Others to help	2	...	...	2	
					3
Total					81

Discussion.

Percy C. Gilchrist, London, Eng.: To Europeans the very low silicons given in Professor Howe's paper (page 1123) are extremely interesting. So far as I am aware, with the exception of Sweden, we are not able in Europe to produce steel successfully if

we use silicon as low as 0.6 to 0.9. We have silicon as low as that in the basic Bessemer, but then phosphorus is present besides. We cannot make Bessemer steel with a pig of that composition, and get it into the ladles, much less into the ingot. I suppose it is done in America by their attention to small details and by the blows following each other quickly. The very low silicon used largely explains how such quick work is possible. If we could get the stuff into the ingots, it would be utterly impossible to make bad steel with ordinary care. As we all know, the only trouble in making acid steel is the high silicon, which requires great judgment in the matter of scrapping or blowing the steel, so that it may regularly contain the same amount of silicon. When we have only 0.6 to 0.9, we cannot leave any silicon in, and therefore if we get the steel into the ingot-moulds (and that is a big "if"), it must be good, because we have only to add, at the completion of the blow, a small amount of ferro or spiegel—which is a very simple thing—in order to get a regular amount of carbon and manganese. Therefore, for any one to say that working quickly must necessarily make bad steel, is, I think, absurd.

The mixer, as described, I think would be extremely useful to us in England. At one of the works with which I am connected, the attempt has been made to use pig-iron direct from the blast-furnace for the basic Bessemer, and given up, because, in spite of the regularity which the blast-furnace managers claimed to have obtained, it was not found that the pig-iron direct from the furnace was sufficiently regular in composition to give the best results in the matter of yield. For example, it paid at those particular works to re-melt all the pig-iron, simply because, by remelting it, we got the least "defectives" and the best yield. The reason of the better yield was very simple. If the pig-iron varies much in silicon, the after-blow will vary with the silicon, and by mixing the pig-iron and re-melting it in the cupola, we get absolutely regular material running into the converter, and the blow will always be of approximately the same duration. The blowers should know that the next blow would require an after-blow of about the same duration as that required by the previous charge. They must arrange them on the Monday morning, so that the material can be kept the same all through the week. Special attention should be given to the Jones mixer, to which reference has been made.

The question of blowing in steam is specially interesting to basic people, because in the basic Siemens process we get rid of the phos-

phorus before the carbon is gone, and if we could, in the basic Bessemer, get rid of that after-blow, we should save from 3 to 4 per cent. in yield. Although we can make good material now, we could make even better with no after-blow. Therefore, any man who discovered how to do away with the after-blow would do one of the three things required to make the basic process perfect. It has always seemed to me that the reason why an after-blow is required is that we are not able to get base and oxygen simultaneously offered to the phosphorus. In the basic open-hearth furnace, oxygen and base are given at the same moment, and we get rid of the phosphorus before getting rid of the carbon. In the basic Bessemer process, we are unable to give a base until the carbon has gone. If some one would discover some vaporous or liquid base that could be blown in with the blast, then that person would do one of the three things required to make the basic process perfect.

William Tozer, Sheffield, Eng.: The experience which I have had in the basic Bessemer practice has been principally in the direction of the higher qualities of Bessemer steel which are used in the Sheffield district; and in studying Professor Howe's paper it has occurred to me that if we could obtain a Bessemer hematite pig-iron which would run lower in silicon than the iron that they offer us in the market, it would be a great advantage to us for certain classes of Bessemer steel. I allude to a silicon probably not so low as the American practice, but if we can obtain an iron with 1.0 per cent. or something of that sort, it would be a distinct benefit to us. As it is now, if we require anything of the kind, we have to use either No. 3 or No. 4 hematite, which is pretty high in sulphur for our purpose. I don't know whether the sulphur could be reduced by increasing the manganese. I suppose, if that were done, the cost would be increased at the same time. I think that our very high silicon irons are not at all suitable to the high quality of Bessemer steel which is made in several districts of our own country. There is one thing that struck me with reference to the enormous output. I think, if in our own country we were able to make anything like the wonderful output which has been mentioned by Mr. Howe, that about five of our rail-works would be all that would be necessary. I simply mention this to show that in America you have a stimulus for increasing your output because you have a market which is always getting larger and larger; whereas, in our own country, we have about fifteen rail-works, and if we were able to do anything like what you are doing here, I think we should only want

about five. I should be interested if Professor Howe would inform us whether the men here work three shifts of eight hours or two shifts of twelve hours. He does not mention that in the paper. It would be very interesting, because it struck me that the men must be very hard worked to get the large turnout. He also mentioned that the way of calculating wages in England is not the same as has been adopted in this country. I can only say that, as far as Sheffield is concerned, and the district there, we always include the wages of every man connected with the Bessemer department in arriving at the cost of our ingots. I mean the stock-men in the pig-yard, the boiler-feeders, engine-tenders, and so on. Mr. Howe has mentioned in his paper that that did not appear to be the custom at the works where he had had facilities for making investigation. I was very much surprised to learn that, for it certainly is the custom in the district that I represent.

WILLIAM F. DURFEE, Birdsboro, Pa.: It is well known to American engineers that I was responsible for the engineering of the works at Wyandotte. It is also well known that those works were abandoned after a few years of experimental existence. They were only intended as experimental works; they had accomplished their purpose, and in those works were made the ingots from which the first steel rails rolled in this country were produced. Those rails were of admirable quality; they went into one of the northwestern roads; they were used up after many years of service; and I believe that at the present day there is only one fragment of the rails in existence. I had the pleasure of seeing that fragment a few years ago mounted by my friend, Mr. Potter, of Chicago, and preserved in a very elegant case. Those works were intended simply as an experiment on a grand scale to practically test the question whether from Lake Superior ores, or from other American ores and metals, a satisfactory steel could be produced. At those works was erected the first laboratory, I believe, that was regularly operated in connection with an iron and steel works in America. Any one who wishes to ascertain precisely what was done there can do so by reference to the *Transactions* of the American Institute of Mining Engineers.*

It was the intention of those who furnished the means for carrying on those experimental works that the moment it was satisfactorily demonstrated that steel could be produced of proper quality, they would proceed with the erection of larger works, in which the busi-

* Vol. xii., p. 223.

ness could be prosecuted on a manufacturing scale. When that time arrived, they were not prepared, or they declined, to proceed in the way that had been agreed upon. I immediately resigned my position, and the attempt was then made, under other management, to develop the experimental work by putting in a second converter into a manufacturing establishment—a thing which I pronounced at the outset to be utterly absurd to undertake; and after one or two years of very unprofitable existence, the attempt proved a failure in a commercial sense, though there was a great deal of good steel made there.

With regard to the apparatus—when I designed that original apparatus, the steel that was made in England was being blown with only 8 pounds of pressure, and we had nothing else to go by in America; but I reasoned from a study of the subject at that time that it would be very much better if the pressure was increased. As a result of that reasoning, I designed an engine for blowing 16 pounds pressure of blast. At that time the highest pressure in use in America for any metallurgical purpose was 7 pounds for a blast-furnace. That engine answered its purpose; it blew 16 pounds successfully. It was used throughout all the operations there, and finally, I presume, it went into the scrap-heap, simply because it was not up to the pressure which was found to be desirable. But I was firmly persuaded, after a few blows, that the movement towards an increase of pressure was a mistaken one, and I am just as much persuaded at the present moment that the ponderous engines that have been designed to produce 20 to 25 pounds pressure are a totally unnecessary expenditure of capital, and that equally good results can be obtained with a much more moderate pressure.

Mr. Howe: Some of the shifts referred to in my paper were of eight and some of them were of twelve hours. In the Appendix is given a full statement of the number of men, and it is indicated, for each class of men, whether there are two or three shifts—that is to say, whether they are 8- or 12-hour shifts. They are ruled off into first, second and third turns. Where only the first and second turns are included, that means that there are only two shifts. Where there are first, second and third included, that means that there are three shifts of eight hours each. As to the limits of the Bessemer department, I think the gentleman is hardly correct, speaking generally; though doubtless as to his own district he knows perfectly. I cannot say just how it is in the books of the establishments, but as the total number of men have been given me by the managers of

Bessemer departments in Great Britain, there are between 32 and 38 per cent. of the men in the Bessemer department who are classed outside. Finally, as to Mr. Durfee's view as to low pressure, had his connection with our Bessemer industry been longer, his strong personality might well have left on it an impress comparable with that of the men whom I have named. The whole system depends upon extremely rapid blowing. The oxygen taken up by the iron must be taken away from that iron before the iron gets to the top of the bath. We have air blown in at the bottom of the bath, and the iron becomes oxidized by the air, which passes up very rapidly. The depth of metal above where the blast enters must be so great as to enable the iron in travelling up to become deoxidized before it reaches the top of the bath. The blast must travel a long distance through the bath in order to accomplish that. The distance which it must travel to permit that must, of course, increase with the rapidity of the blowing. If blowing slowly, a shallow bath might suffice to deoxidize the iron; but with large outputs, and blowing rapidly, we must have a deep bath. If we have a deep bath, we must have high pressure, as Mr. Durfee knows quite well.

Discussion Continued by Correspondence.

James Gayley, Braddock, Pa.: At the Edgar Thomson works the practice is to use iron containing a small proportion of silicon. The average content of silicon in the metal, as it is poured into the vessels, is 1.10 per cent. As all of the metal first passes through a metal-mixer, a considerable quantity of the initial heat is lost; but in cases where the iron is sent direct to the Bessemer department, *i.e.*, without passing through the mixer, it is found that 0.80 per cent. silicon will give the same result as to temperature of bath. For rapid working, low-silicon metal is not only a necessity, but it avoids, to a great extent, the scrapping of the heat in the vessel, which is an uncertain operation. The low-silicon iron spoken of is not, as it is often considered, a synonymous term for high-sulphur iron. As a matter of fact, the average sulphur in our iron, ranging from 1.00 to 1.30 per cent. silicon, is lower than when the silicon is 2.09 to 2.30 per cent. This is what might naturally be expected, inasmuch as the low-silicon metal requires less coke for its production; and as, in our case, the sulphur is derived entirely from the coke, there is less sulphur to be got rid of.

The grade of metal is in no wise deteriorated by keeping the sili-

con low; iron containing 1.00 per cent. silicon being as open-grained as that containing 2.00 to 2.50 per cent.; in fact, it is quite common to see a high grade of No. 1 iron with only 0.80 per cent., and even with silicon as low as 0.65 per cent. the metal is gray and has a good grain. It frequently happens that the Bessemer department will run for twelve hours on metal which does not exceed 0.70 per cent. of silicon, although the result from such metal is not as good as when silicon is higher.

JOHN FRITZ, Bethlehem, Pa.: The American Bessemer practice outlined in Mr. Howe's paper is framed upon facts readily accessible at our different works; and knowing him to be thoroughly efficient as a discriminating collector of such material, I have every confidence in his presentation of the case, especially since I find such facts as come within my own personal experience to be correctly stated.

The question raised as to the low silicon percentages used at one or two of our works, arises mainly, I think, from a want of familiarity with the close and accurate methods that obtain in the driving practice of these works.

Our own practice is by no means so rapid, and our pig is comparatively free from manganese; and yet, melting with anthracite in place of coke, we find no difficulty in working iron of 1.10 per cent. silicon. This we vary to as high as 1.50, to suit the conditions of different steels varying in carbon from 0.06 to 1.25 per cent.

Melting with anthracite is by no means so rapid as with coke; but preference is given to it owing to the lower and much more regular content of phosphorus and sulphur. All anthracite for cupola-melting is carefully selected from one or two mines, and frequent analyses are made in order to avoid, as far as possible, any variation.

E. C. POTTER, Chicago, Ill.: While I believe that the general average of American Bessemer pig-iron, as delivered to the converter, is lower in silicon than the established European practice, yet it seems to me that Mr. Howe has given undue prominence to practice which uses pig-iron exceptionally low in silicon, thereby unintentionally misleading his readers as to the usual and average American practice. Even the practice which he cites, as using iron with silicon considerably below 1 per cent. is not the ordinary or average practice at the works in question. If I translate Mr. Howe's cipher names of works correctly—and in this particular case I believe I do—the use of iron as low in silicon as stated, is, I believe, largely

confined to those occasions of "beating the record," for which these works have achieved a considerable notoriety. For these occasions, in order to attain the greatest possible speed, a sufficient quantity of low-silicon iron is selected out of stock, and reserved for this great effort, which extends over periods varying from a week to a month. In this way, certainly, many consecutive heats, and even many consecutive days, of very low-silicon practice may be obtained. But when the record has been sufficiently broken, a return to the ordinary stock of iron is necessary, the average of which is considerably higher in silicon because of the withdrawal of all the low-silicon iron. I have known of these works being compelled to run largely on iron carrying 3 per cent. to 4 per cent. of silicon, after a month spent in "record-breaking." Their average therefore, is no lower than the rest of the country, which I should place at 1.75 per cent. to 2.00 per cent. of silicon. It is impossible for any works, or any group of works, to run continuously on iron below 1.00 per cent. in silicon, simply because it is impossible for any blast-furnace to make this character of iron regularly and uniformly, and of sufficient purity, especially as regards sulphur, for steel-making. I would not be construed as condemning the use of low-silicon iron, or controverting Mr. Howe in his general deductions. I merely desire to correct what seems to me to be an unintentionally misleading impression to the reader of his admirable paper, that the usual or typical American practice is the use of this very low-silicon iron.

Upon one point, however, I must squarely take issue with Mr. Howe—namely, on the question of rapid running as affecting quality. Upon the general proposition that of two plants of equal size, one American, the other European, the former will yield a considerably larger tonnage than the latter, and in that respect is a "rapid" plant, I am firmly of the opinion, with Mr. Howe, that our quality does not suffer with the larger output, for reasons clearly set forth in the paper under discussion. But when a plant which, for example, would make 700 tons per day in Europe, or 1000 tons per day in America, is forced in America to make 1200, 1400, or even 1500 tons per day, I insist that the result as regards quality is unsatisfactory, not to say disastrous. I am speaking now of that strictly American practice of "beating the record." The utter recklessness with which operations are carried on on these occasions is sufficient proof that the results must be bad—and they *are* bad. I never knew of a record broken in which the record for high percentage of second quality was not also broken. The desire for tonnage, tonnage, nothing but

tonnage, by both manager and men, leads to slovenly, careless work. In the mad struggle for supremacy everything goes with a rush—scrapping carelessly done; blowing more or less inaccurate; heats poured hurriedly and in a slovenly manner; nozzles in ladles badly set and imperfect streams resulting; moulds carelessly and irregularly set in their stools, causing large "fins" on the bottoms of ingots; moulds insufficiently cooled; and ingots stripped too soon. Any or all of these deficiencies would seriously impair the quality of the product. Then, when the rail-mill is reached, we find the steel either insufficiently heated or scorched; the rolls are not changed as soon or as often as they should be; saws are allowed to run until dull, resulting in rough and inaccurate cutting; rails are straightened so hot that they cannot be handled without hand-leathers, consequently they continue to warp after they have left the mill, reaching the purchaser's hands in bad condition, and often unfit to lay in track. It is in the finishing department of the mill that the sins of "beating the record" are most largely developed. A hot-bed designed to deal with 1500 rails per shift cannot well deal with 2000 rails per shift with equal care. There are two ways of meeting the emergency of a heavy surplus of rails on the hot-bed: one, to pile the surplus of rails, unstraightened, in the yard, to be straightened when not engaged in "beating the record," which is expensive and not particularly good for the rails; another way is to *cool all the rails on the hot-bed with water*, which certainly facilitates matters, but cannot be recommended as good or safe practice. It has actually been done, though, at a works noted for its "distinctive American practice," use of low-silicon pig, and frequency of "beating the record." It is unnecessary to add that these rails so cooled were not popular upon the railway for which they were intended, the manager of which railway was singularly unappreciative of the necessity for "beating the record." These same works, after a month's run, in which the records for the month, week, day and shift were hopelessly smashed, and the works and its manager surrounded with a halo of glory, took *three months* to straighten and ship the surplus of rails piled in the yard, working Saturday nights and Sundays. During that time the accounting department was entirely in the dark as to whether certain contracts were filled, or where they stood with their orders.

There is one individual about the plant who does not enter upon a "record-breaking" spurt with the same enthusiasm as his co-

laborers. This is the master-mechanic. A broken record has no charms for him. He knows who will have to "pay the fiddler," and thirty-six hours of steady hard work on his machinery to get it in any kind of shape for the following Monday morning is not alluring. He is the man who is compelled to beat all kinds of records to get ready for next week, and he can testify feelingly that our machinery *does* suffer by reason of the over-driving incident to "beating the record."

This is a cursory glance at some of the salient features of this peculiarly American institution, whereby many managers gain renown for themselves and their works. I fail to see any features in it to commend it to universal practice. It simply is a "spurt," by which a rate of speed is attained that could not be continued, and would not be if it could. The manager, therefore, has not proved anything as to his capacity, and has simply succeeded in deteriorating the quality of his product to gratify a foolish and eventually extravagant pride. What, then, is there to praise? I believe most heartily in the spirit of emulation and *esprit du corps*, which is so important a factor in the successful management and operation of our works, but I believe these things should be tempered with reason and good judgment.

Mr. Howe: Mr. Potter's strictures demand a very full reply. In speaking of the Ningunaparte practice of having but from 0.6 to 0.9 per cent of silicon in the whole vessel-charge as "most characteristically American," I thought that the immediately preceding clause, stating that in the majority of American mills the vessel-charge had less than 1.75 per cent. of silicon, made it clear that by "characteristically American" I did not mean usual. The percentage of silicon at Ningunaparte is decidedly lower than at most American mills.

That I have given the Ningunaparte practice undue prominence I cannot admit. It is one of those clouds like a man's hand, whose importance is not readily overrated.

The latest information which I had received from the officers of Ningunaparte gave their usual silicon-content as between 0.6 and 0.9 per cent. It is still their favorite practice; but as it is not always convenient to obtain iron with so little silicon, and as, owing to temporary delays, it is sometimes desirable to use a little more silicon, the average proportion of silicon taken over long periods is a little higher. The last returns which I now have, covering a period of

nearly a year, give the average proportion of silicon for the whole period as a little less than 0.98 per cent., 10 per cent. of all the heats having less than 0.73 per cent. of silicon.

The Bessemer superintendent writes me (March, 1891): "I should have had it lower if I had had the low-silicon pig to melt; as it was, I was obliged to use all kinds of scrap to keep it down."

He states further, that if the rail-mill would take the steel as fast as he could make it, he could run continuously with from 0.66 to 0.76 per cent. of silicon in the molten metal—I believe further reducing its effective calorific power, by charging in addition cold steel scrap, and blowing steam through the bath.

As far as the calorific power of the charge is concerned, the above average of 0.98 per cent. of silicon is equivalent to an average probably of less than 0.90 per cent. of silicon in the case of charges wholly melted before their introduction into the vessel; for about 4 per cent. of the charge is introduced in a cold state into the vessel; and a great deal of steam is blown through the charge to lower its temperature. By like reasoning, the average silicon-content of the first four months of 1890, at the Union Mill, as given in Table IV., is equivalent to an average of about 0.85 per cent. of silicon in the case of wholly molten charges.

I will now take up some of Mr. Potter's statements in detail.

1. He states that the average silicon-content at Ningunaparte is not lower than from 1.75 to 2.00 per cent. He is misinformed. I have just given the average for a period of nearly a year as less than 1 per cent.

He says that *the average of silicon in other American works is from* 1.75 *to* 2.00 *per cent.*

I had stated that the majority of mills use less than 1.75 per cent. I have before me information from seventeen American Bessemer works, including all the principal ones.

Number using less than 1.30 per cent. of silicon, ...	7
Number using less than 1.75 per cent. of silicon, ...	13
Number using more than 1.75 per cent. of silicon, . . .	4

My information from those using more than 1.75 per cent. of silicon is, with one exception, some years old, and it is not unlikely that some of these have followed the general current of American practice, and reduced their silicon. Messrs. Fritz and Gayley have just given you the practice of Bethlehem and Edgar Thomson.

Mr. Potter may be misled by taking as the silicon-content of the

vessel-charge that of the pig-iron used. This, however, is first diluted by the addition of scrap before melting in the cupola, then impoverished by the removal of silicon in the cupola, and finally further diluted by the addition of cold scrap in the vessel itself.

2. He says, *It is impossible to run continuously with less than* 1 *per cent. of silicon, because the blast-furnace cannot make iron with so little regularly, and free enough from sulphur.*

Mr. Gayley has only just told us that the average silicon-content of the Edgar Thomson metal (direct-metal, I take it), as it goes to the vessel, is 1.10 per cent. If this metal were to be remelted in cupolas, as in common American practice, and if the manager sought to bring the silicon-content of his vessel-charge continuously below 1.00 per cent., he could certainly do it by so mixing the iron that each charge would have nearly the average composition of the metal from the blast-furnace, or 1.10 per cent. If, now, the metal further lost 0.20 per cent. of silicon in melting in the cupola, and if to the melted charge 8 per cent. of scrap-bloom and rail-ends were added, the silicon in the vessel-charge would be only 0.83 per cent. Mr. Gayley further tells us that at Edgar Thomson the sulphur decreases with the silicon, instead of rising, as Mr. Potter implies.

But the loss in the cupola is often greater than I have here taken it, and the proportion of scrap to be remelted may be higher; so that it should be wholly possible, under such conditions (I do not say convenient), for the Bessemer manager continuously to obtain in practice vessel-charges even lower in silicon than 0.83 per cent., if he sought them.

3. Admitting that reasonably rapid running does not injure quality, Mr. Potter takes issue squarely with me, insisting that record-breaking efforts injure quality. This is quite a feat, as I had never intimated that they did not. Then follows a vigorous denunciation of record-breaking, and of the Ningunaparte record-breakers. To make sweeping charges against a class of men is easy; to rebut them in detail necessarily difficult. To prove a negative is rarely easy. Many of Mr. Potter's charges cannot, in their very nature, be disproved. Mr. Potter himself has been a notorious record-breaker. I find it difficult to believe his charges, even as applied to himself. As I have no interest, beyond a love of truth, in collecting evidence to free other record-breakers from his charges, and you none to hear detailed evidence in rebuttal, let it suffice if I insist generically that his picture of what occurs during record-breaking is, as regards those record-breaking mills with which I am familiar, including Ninguna-

parte, most grotesquely and unrecognizably exaggerated and distorted; and if, replying specifically, from such evidence as is at hand, I meet with exact numerical data enough of his unsupported assertions to give the reader some idea of their credibility, and of the care exercised in their preparation.

4. *Mr. Potter never knew a record broken which did not also break the record for high percentage of second-quality rails.* I know many. Indeed, where, as at Edgar Thomson, the rails are chipped between the passes at the rail-train, and where every bloom that even gives the least promise of producing a second-quality rail causes serious delay, while its incipient cracks are chipped out, it is evident that, if the output of rails is to be great, the proportion of second-quality rails must be small. That this is true, witness the fact that the greatest "runs at these works have been made when the percentage of 'seconds' has been lowest," as I am assured (March 4th, 1891) by the most unquestionable authority. Witness the statement of Captain Jones, at the time general manager of these works, that "it is impossible to attain great speed in working while making bad steel." *

At the Union Works the proportion of seconds for the months of November, 1887, and May, 1888, both of them, I believe, record-breaking months, was 1.53 per cent. and 1.42 per cent., respectively; by no means excessive. I have known mills to make over 10 per cent. of second-quality rails when running at moderate speed.

The output of the Union Works for several of the 17 consecutive months represented in Table IV. breaks almost all records existing prior to the first of them. If there were any necessary connection between the output at these great speeds and the proportion of second-quality rails, it should be apparent in this table. Yet it gives no such indication. Confining ourselves to the last 11 months, May, the slowest but one, has 60 per cent. more seconds than the average. The average proportion of seconds for the fastest 6 months is actually less than that for the slowest 5 months.

5. He says that, *in record-breaking, scrapping is carelessly done.* Careless scrapping implies irregularity in the weight of the charge, and hence an excessive proportion of butt-ingots, *i.e.*, of pieces of ingots which fall below that weight needed for making an aliquot whole number of rails. Yet in the banner-month of October, 1889,

* *Journal of the Iron and Steel Institute*, 1881, i., p. 137.

at the Union Works (cf. Table I.), so far as I know still the champion month for the whole world, the proportion of butt-ingots was only 0.47 per cent. of the total weight of metal charged. If this be madness, there's method in it.

So, too, the average proportion of butt-ingots in the last 11 consecutive months in Table IV., all of them high-speed months, was only 0.66 per cent.

6. He says that *in record-breaking the blowing is more or less inaccurate.* So it is; decidedly less inaccurate than to justify his apparent implication. Turning again to the last 11 months, Table IV., we find that the variation in the proportion of carbon in the steel is actually less in the fastest 6 than in the slowest 5 months. That is to say, with the irony of fate, the rapid work here, even at this tremendous pace, had more accurate blowing than the relatively slow work.

During the famous month of October, 1889, at the Union Works, the carbon in the steel lay, with the exception of a single doubtful heat out of the (about) 3600 heats, within a range of 0.06 per cent. above, and 0.07 per cent. below the average. Considering the large number of heats represented, considering that, in order to save iron, it is desirable to arrest the blow at the first marked indication of shortening of the flame, is this more, or is it less inaccurate blowing?

Now, does this inaccurate blowing, which, Mr. Potter says, arises during record-breaking, consist in blowing sometimes too long and sometimes not long enough? In that case the insufficiently blown heats would give steel which would be too hard, and the resulting rails should either be classed as of second quality or rejected. But, as we see elsewhere, there is no indication of an excessive proportion either of second-quality rails or of rejected rails.

Or does the inaccurate blowing consist solely in sometimes blowing too long? In this case, the loss of iron should be excessive. Yet, in the banner-month of October, 1889, at the Union Works, the loss was only 10.85 per cent.; the average loss for the last 11 rapid months of Table IV. was only 9.60 per cent.

Finally, inaccurate blowing should cause irregularity in the weight of the blown charge, with consequent increase in the proportion of butt-ingots; yet we have seen that this proportion is so insignificant as to indicate extreme accuracy in all the conditions affecting the weight of the blown metal, whether as to weighing, blowing or pouring.

7. He says that during record-breaking *the steel is poured hurriedly and in a slovenly manner; the nozzles are badly set, causing imperfect streams; the moulds are irregularly set, so that large fins form on the bottoms of the ingots.* If this is true, it should manifest itself by increasing the proportion of second-quality rails, of pit-scrap, and of butt-ingots. If the pouring is irregular, the weight of the ingots must vary, and hence a considerable weight of butt-ingots must arise. We have seen that their weight is inconsiderable.

Actually, only 2.30 per cent. of scrap altogether was made in the month of October, 1889, at the Union Bessemer Works, of which 2.21 per cent. was pit-scrap, and 0 09 per cent. was iron scrap. So, too, in the last 11 consecutive rapid months given in Table IV., only 2.11 per cent. of pit-scrap was made.

The effect of record-breaking on the proportion of second-quality rails we have already considered.

8. He says that *during record-breaking the moulds are insufficiently cooled.* The effects of insufficient cooling should be, first, to cause the ingots to stick in the moulds, thus delaying operations hopelessly, and so preventing record-breaking, and causing cracks in the ingots, and thus leading to increasing the proportion of second-quality rails; and second, to hasten the destruction of the moulds.

But we find that in October, 1889, at the Union Works, the moulds lasted unusually long, viz., an average of 92.63 heats each. This agrees with common experience and common sense—that it is only by the closest attention to such details as the proper cooling of the moulds that great speed can be maintained.

9. He says that *during record-breaking the ingots are stripped too soon.* If the ingots are to be heated on end, which is thought the best practice, and which is done at the Union Works, they cannot be stripped too soon, so far as quality is concerned. The only danger in very early stripping is that the ingots may bleed; but the low proportion of pit-scrap, 2.21 per cent., already referred to, shows that no serious amount of bleeding can have occurred.

10. *Any of the evils, 5 to 9 inclusive, would seriously impair the quality of the steel.* The statement that either careless scrapping, or irregular setting, or insufficient cooling of moulds, or early stripping of ingots would seriously injure the steel, points to that "going with a rush" into print, which Mr. Potter deplored as applied to Bessemer practice.

11. *During record-breaking the ingots are either over- or underheated; rolls are not changed often enough; neither are the saws, so*

that they cut roughly and inaccurately; rails are straightened so hot that they thereafter warp in cooling. Charges easy to make, hard to refute. Each of these evils would be likely to lead to rejection by the purchaser's inspectors, and should thus diminish the shipments of rails. For if the steel were not at the desired temperature on entering the rolls, neither would it be on leaving them and on sawing: hence its contraction after sawing would be either too great or too little, and the rails would not be true to length, and should be rejected. If the rolls were not changed often enough, the rails would not be true to section, and should be rejected; so, too, if the saws cut irregularly, the rails, not being true to length, should be rejected.

Turning now to the facts, we find that in the record-breaking month of May, 1888, at the Union Works, the shipments of rails were 96 per cent. of the weight made, which tallies ill with the Bedlamite reign of terror which Mr. Potter so ruthlessly exposes.

12. It is hard to comment on Mr. Potter's remarks about insufficient hot-beds. Nobody applauds insufficient hot-beds, nor do they form a feature of American rapid practice. There is room in our country for all the hot-beds needed. Nor is it common American practice during record-breaking to cool rails with water. It has been done after the rails have become perfectly black, but never, so far as I know, injuriously. In stating that, at the mill he clearly refers to, it took three months to straighten and ship the excess of rails piled in the yard during a single record-breaking month, I have it on direct authority that Mr. Potter is wholly misinformed.

13. He says that *our machinery does suffer from over-driving incident to beating the record.* A bare assertion, wholly unsupported by evidence. To reconcile it with the detailed statements as to the actual endurance of our machinery given in the body of my paper, and with the almost record-breaking output kept up during the greater part of the 17 successive months given in Table IV., few will find easy.

Looking at it now in a general way, it should be apparent to every one experienced in executive work that, where great speed is to be kept up for a month at a time, the very administrative difficulties which must arise demand as a fundamental condition that there shall be no mad rush; that every detail shall not only be carefully thought out beforehand, but carefully executed. "The utter recklessness," the "slovenly careless work," "the mad struggle" which exist, accord-

ing to Mr. Potter, must necessarily defeat themselves. He who would run a rod at highest speed may indeed rush madly; he whose army spans a continent the swiftest, husbands his resources, looks well to diet and shoeing; while he must not waste time, neither dare he waste strength. What protracted struggle against time or man or beast was ever won by utter recklessness, mad rushes, carelessness and slovenliness?

While, like Mr. Potter and every other sane man, I would not commend record-breaking "for universal practice," I am confident that its occasional practice is desirable; for, in preparing for a record-breaking effort and in carrying it out, expedients are devised in the way of organization and of detail of procedure; short cuts are established, which are used after the effort has ceased, and which may introduce great economies. Witness our low-silicon practice. A speed is reached which, indeed, cannot be long sustained then; but the experience gained permits a greater permanent speed than was possible before the spurt, and, when we return to normal speed, it is no longer that which was normal before. Little by little the organization adapts itself to this new speed, till it is as easy as the old was. Then comes a new spurt, with another rise of the normal speed. After a number of these elevations of the normal, we find that it is far, far beyond the maximum speed of our early spurts. Thus the average output of the 17 consecutive months in Table IV. is 78 per cent. greater (or, if we omit the 3 months affected by extraordinary delays from strikes, orders from the directors, and the annual stoppage for general repairs, 90 per cent. greater) than the 15,000 tons which Captain Jones looked forward to, in 1881, as the probable limit which we might some day reach.

I am confident that these spurts have, on the whole, been productive of good, both to mill-owners and the public; that it is in no insignificant degree due to them, and to the talent which has directed them, that the cost of the Bessemer process has diminished so greatly; that in general—certainly in the cases which Mr. Potter singles out for attack—haste *has* been "tempered with reason and good judgment."

I am surprised that Mr. Potter thinks that either the intelligence or the character of the men who have broken our records would permit them to sacrifice the interests of which they are the trustees, in the way he depicts, especially when the President behind has his finger ever on a button, which would blow the earth from beneath the manager's feet.

TABLE IV.—*Work done at the Union Bessemer Mill with two 10-ton Vessels.*

Month.	Output per Month, Tons of Ingots.	Turns.	Tons of Rails per Month.	Turns.	Average per cent. Silicon.	Average Length of Blow, Minutes.	Average Blast Pressure, lbs. per Square Inch.	Average number of Heats per Hour.	Average Life of Bottoms, Heats.	Relative number of Heats per Hour, 1.00 the Lowest.	Relative Proportion of 2d Quality Rails, 1=the average.	Relative Variation of Carbon, 1.00 = the Lowest.
1889.												
July,	28,212.9	47	22,983.5	47	...	...	...	...	...	...	...	...
August,	30,264.4	50	25,107.3	50	...	...	...	...	...	...	...	...
September,	26,264.3	46	21,151.4	46	...	...	...	...	...	...	...	...
October,	36,232.8	54	28,491.6	54	...	...	...	...	...	...	...	...
November,	20,699.1	36	17,771.3	36	...	...	...	...	...	...	...	...
December,	13,235.5	35	12,091.9	26	...	...	...	...	...	...	...	...
1890.												
January,	32,546.0	50	26,555.6	50	0.887	9.98	25.28	5.07	23.75	1.29	1.07	1.66
February,	28,454.5	44	23,053.5	44	0.876	9.39	25.10	4.93	24.34	1.25	0.97	1.55
March,	27,494.7	47	22,756.0	47	0.967	9.60	24.40	4.42	23.35	1.12	1.23	1.22
April,	28,825.4	48	23,151.8	48	0.952	9.19	25.60	4.57	22.51	1.16	1.19	1.22
May,	21,389.8	39	17,469.4	39	0.939	9.23	26.00	4.04	20.70	1.03	1.60	1.88
June,	25,016.3	46	20,298.5	46	1.128	9.84	26.52	4.24	19.64	1.07	1.04	1.55
July,		...	...	...	1.103	9.80	26.60	4.20	19.00	1.06	1.03	1.33
August,		...	...	...	1.086	10.00	27.50	4.20	17.73	1.06	0.60	1.44
September,	135,708.0	...	...	...	1.067	9.69	27.60	4.56	22.00	1.15	0.65	1.22
October,		...	...	...	1.035	10.02	27.34	4.64	21.10	1.18	0.87	1.11
November,		...	...	...	0.966	10.11	26.12	3.94	19.50	1.00	0.84	1.00

Proportion of ingots, etc., from Jan. 1, 1890, to Dec. 1, 1890.

Ingots,	87.43
Butt-ingots,	0.66
Pit-scrap,	2.11
Iron scrap,	0.20
Loss,	9.60
	100.00

NOTE.—The following corrections of Mr. Howe's paper were received after the pages to which they belong had been finally printed:

On page 1160, the number of ladle-men should be 28 instead of 22, and the total should be 503 instead of 498.

On page 1165, the total of men at pressure-pumps should be 1 instead of 4, and the grand total should be 503 instead of 498.

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